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The Outlook.

When exact statistical data on the production of the metallurgical and chemical industries in 1905 are available, the claim that the past year has been a red-letter year in the industrial almanac of this country will undoubtedly be found justified. To mention only an instance, the market for metals has been very high and is very high now, while this note is being written—a most remarkable phenomenon for the week after Christmas. What is more important, however, prosperity has been coupled with progressiveness towards new developments, and since prosperity is Nature's acknowledgement of activity in the past, we may hopefully look into the future. The specific American trait of working on a large scale and of pushing the output to the utmost limit, has been manifest during the past year in every direction. We cite as examples the reverberatory and the blast furnaces of enormous tonnage at Anaconda, as well as the fact that the output of our electrolytic copper refineries is now twice what it was six years ago. But besides this tendency of working on a large scale, there has been a marked desire to improve methods. This has not always been so. Huge scale working tends towards standardization, and means the investment of large amounts of money in installations. It is most gratifying to see that in spite of this fact our engineers are endeavoring to adapt their equipment and methods to the progress of the art. The Carnegie Steel Co. are equipping its new furnaces with Mr. Gayley's dry blast system. Several large blast furnace plants of the United States Steel Corporation will put in large gas engines, operated by blast furnace gases, for the development of electric power. By-product coke ovens are growing in favor. In the lead industry the adoption of the Huntington-Heberlein process has been the most important development of the last year, while Mr. Betts' electrolytic lead refining process has achieved international importance through the erection of English works. In the special field of elctrochemistry the first days of the new year will witness the opening of the new chlorine-alkali works, using the Townsend process, and of the Acheson silicon works at Niagara Falls, as well as the starting of the 80-ton electric Héroult furnace at the Halcomb steel works. These are only a few examples which could be easily multiplied. They indicate a new life and a new spirit in the industrial and metallurgical industries of this country.

Separation of Copper and Nickel

The treatment of the nickel-copper ores of the Sudbury district is a problem of the greatest importance to the metallurgical industry. Roasting and smelting, followed by [illegible] treatment, yields a high grade nickel-copper matte, but the problem is to separate the nickel from the copper. This is done on a very large scale by the Orford process, [illegible]

of the Orford Copper Co., in Bayonne, N. J. The principle is to subject the molten matte to contact with sodium sulphide, whereby the heavier nickel sulphide separates by gravity from the copper and iron sulphides. The process is cheap and has been very successful from a commercial standpoint. Its greatest drawback is that the separation is not sharp, that the resulting nickel is impure. To get pure nickel, electrolytic methods early suggested themselves, on account of the analogy with copper refining. For a complete understanding, however, it is not sufficient to look at the engineering side of the problem, but the commercial situation must also be considered.

* * *

Before the International Nickel Co.—the "Trust"—was formed, it was in the interest of independence that the Canadian Copper Co., in Copper Cliff, endeavored to develop a process different from the Orford process. For a series of years experiments went on to devise an electrolytic method. An unsuccessful experimental trial was first made with the process of the late C. Hoepfner, but success was obtained later with the electrolytic process of the chief chemist of the Canadian Copper Co., D. H. Browne. At the same time the Orford Copper Co., while using the alkaline sulphide smelting process on a large scale, was compelled, for a number of years, to meet the demand for very pure nickel for certain purposes by having a portion of its nickel product refined at the Balbach works in Newark, by the electrolytic process of their chief engineer, William Thum. Later came the consolidation of the different nickel companies in one trust, which then controlled both the Orford Copper Co. and the Canadian Copper Co., and the Browne process was given up for the time being, since its cost was found to be higher than that of the Orford process. A large item in the cost sheet of the Browne process is the cost of electrical power, and it is expected that when the electric transmission plant to the Copper Cliff works of the Canadian Copper Co. will be completed, cheap electric power will be available, and the prospects for the Browne process will be greatly improved. In the meantime considerable experimental work has been done at the Bayonne works of the Orford Copper Co. According to a note mentioned in the Synopsis of our present issue, electrolytic nickel refining (starting evidently with the impure product of the Orford process) is now carried out on a small scale at the Orford works. Concerning the experiments made on electrolytic copper-nickel matte treatment at the same works complete secrecy has been maintained. Information on one process, worked out thereby N. V. Hybinette, is, however, now available from a patent recently issued, and discussed in the department on Recent Metallurgical Patents in our present issue.

* * *

It is interesting to compare in a general way the Browne process and the Hybinette process. Browne (whose process was described in detail in our issue of July, 1903,) uses a chloride solution, Hybinette a sulphate solution (with a weak acid). Both inventors use a cyclic process, and both pay special attention to the important problem of keeping their solutions pure. In the Browne process it is chlorine, which goes through a cyclic process, in the Hybinette process it is the nickel-sulphate electrolyte. Browne deposits by far the larger part of the copper and all the nickel electrolytically, and removes the rest of the copper and the iron by chemical means. Hybinette deposits only the nickel by electrolysis, while the copper is precipitated by cementation. Browne uses simple electrolytic cells, Hybinette a diaphragm cell of novel construction. It was recently stated that the Hybinette process is to be tried outside of the nickel trust, at Sault Ste Marie. The results will be watched with interest.

Depreciation Due to Advance in the State of the Art.

Our friends and co-workers, the electrical engineers, have made use of the phrase, "depreciation due to the advance in the art," in their methods of accounting, and especially in the accounting arising from law suits. There is a very good reason for this in the electrical profession, and that is that the marvelous application of fixed mathematical laws by a brilliant coterie of engineers has resulted in changes proceeding at the rate of an express train. Good, excellent generating machinery of twelve years ago is now antiquated for a large central station, because it is brought into competition, for instance, with the high-speed alternating-current turbo-generator, and it is not impossible that the advent of the osmium and tantalum lamps and of the graphitized filament may place many a small lamp plant in uncomfortable circumstances. In applied chemistry—or as we like to term it, applied physical chemistry—such changes have not been so marked as in electrical engineering. The reason for this is plain. Progress is made in metallurgy and chemical engineering more by the methods of "cut and try" than by rigid theoretical deduction. Metallurgical experimenting is costly and slow—slow, very slow. Many processes successful in the laboratory are abject failures in the works, because of faulty engineering, or more often, because of ignorance of the real conditions for success. Perfect theory would, of course, have rendered all easy.

* * *

However, there have been many changes in metallurgy, such as the Bessemer process, which restricted the advance of the puddling furnace. The basic steel processes which destroyed natural monopolistic value of the "Bessemer ores," the cyanide gold process, which rendered low-grade gold mines available, are both instances of epoch making changes. The electrolytic copper process not only made high conductivity copper commercially available, but opened up vast mines of Montana and Arizona. All these changes, however, went slow working, because of the great "economic friction" in metallurgy, which is an art rather than a department of applied science. However, due to the great progress made in physical chemistry and the general and widespread propagation of its tenets at the present, metallurgy is gradually coming to the same state, which is typical of electrical engineering.

* * *

No conservative system of accounting has been devised to cover this all-pregnant phrase, "depreciation due to the advance in the state of the art." The nitre beds of Chili have an enormous value now. But when the electrochemical fixation of nitrogen from the air is a demonstrated success, perhaps this value will be annihilated over night. The Huntington Heberlein roasting process will wipe off the book value of

thousands of dollars of capital tied up in hand roasters. The electric steel furnace will do the same for the present inefficient crucible steel furnace. Then, too, the advance of analytical chemical methods and their widespread diffusion will obliterate the fictitious value of many "brands" whose "good will" has largely a sentimental worth only. There is just one way for the enterprising manufacturing company to succeed and discount this uncertain but sure depreciation. Adopt the methods used by many, but most of all by that canny American, Andrew Carnegie, and make your plant the most modern and efficient in the world, by the aid of able and enthusiastic partners. Raise the personal equation to the highest power. There the depreciation will be written off on the capital account of your competitor's book and not on your books. In this twentieth century, modern business methods are the only right ones.

"When all Treasures are Tried, Truth is the Best."

We hesitate to approach the subject of "graft" in business and politics, when the editorial page of every journal of the country is replete with similar warnings and forbodings. But the question is so germane to engineering and the great flood of prosperity that is blessing our land is so intimately connected with it, that we feel that there is little apology needed for considering the basic causes of what we may term "commercial leprosy." Anthropologists tell us that human nature has changed but little in 2,000 years, and it is well to confess this. Then we can trace the causes directly to the changing environment and sociological conditions of the twentieth century. There is much truth in the song of the Broadway theatre ending with the refrain, "In the good old days of old when people did just as they ought to do—or at least they never told." In these days of "modern inconveniences," the limelight of the daffodil-colored press, by aid of photograph, telegraph and telephone, brings out every bit of scandal into bold relief. Not that we did not have public scandals, as the "Credit Mobilier" in the seventies, nor that other countries did not have them, as France in the "Little Napoleon's" reign, England in the time of Clive, or Prussia in the days of Frederick the Great. But modern-applied science has developed the means of bringing many a nastiness to the public vision.

* * *

Aside from the part played by applied science in showing up dishonesty, we believe that to her door can be traced the fundamental causes of dishonesty. These causes are two in number, one intellectual and moral, the other material and sociological. Galileo, Copernicus, Newton, Darwin and a host of other scientists have brought forth fundamental truths of nature that are seemingly at variance with the doctrines of the church. To a thinking and logical man, the tenets of the ecclesiasticism are but the forms of an outgrown creed, and no set philosophy has arisen to take the place of old faith. Thus, every thinking man rejecting the forms of theology, finds out by his thirtieth year a working moral code. But, nevertheless, in his formative young manhood he is subjected to the moral strains that often reduce a thinking Puritan to cold-blooded materialism. The second cause, the material one, is due to great increase in material wealth brought about by inventions —especially metallurgical inventions. For instance, Sir Henry Bessemer's process of making steel at a low cost connected the broad acres of the West to the world and the entire commercial world. And as cheap metals are the fundamental necessities of all mechanical arts, we can say without fear that the application of the chemistry of Dalton and Lavoisier has been the prime cause of this excess of material wealth. This extravagance of the princes of fortune has caused moral deterioration of the less fortunate. Concisely stated, the increase of national wealth has subjected our national conscience to a strain nearly up to its elastic limit. Individuals get fortunes made by luck on good management, and these are the desire of the jealous and covetous. Integrate this covetousness between national limits and you have "graft."

* * *

Three young boys once used to gather mushrooms together and being straight, nice, honest boys, they made an agreement, let us call it a "gentleman's agreement." This was to the effect that he who first "sighted" a "big bunch" had all in the vicinity of 10 feet of the biggest mushrooms. This scheme worked beautifully, and much pocket money was gained by selling the product of their gleanings at 75 cents a pound. But one Saturday, having walked long, they came to a virgin pasture where the mushrooms were so thick that there was there at least $15 for each. Then the rule which worked so well in the lean fields was slightly transgressed by one enthusiastic youth. This lead to a second infraction. Retaliation, recrimination and bickering ensued in quick succession; then a fight—*bellum omnium contra omnes*. Finally, disgusted and sorrowful, for fully half of their mushrooms were destroyed, these little fellows put all their mushrooms in a pool and divided them equally. It seems to us that this story is illustrative of the sociological changes that are being enacted before our very eyes; that our great increase of material wealth has blinded our eyes to justice, and that a mad race for wealth at any price has followed. Whether we will have the recourse to socialistic division of property that our three little friends had will be read in the future histories. At all events, we believe that as Carnegie, Morgan, Hill and Rockefeller have found in the "trust" the most efficient way of producing wealth, their successors must find the way to efficiently and justly distribute it. The genius of America has always been ready for the problem of the day, and signs are not wanted that a reaction against the ignorant sin of graft is started. But of all men, engineers and technical men, who know that "you cannot lie to science or bluff a chemical equation," should be in the lead in the fight for honesty. The moral idiot who defined "an honest man as one smart enough to keep out of jail," has made many young men cynics by that remark. Honesty is the best policy. Yes, it is far more than that, it is right and true as nature is right and true to her children that love her. Benedict Arnold was a noble man in comparison to him, who is an example of material success by prostitutional methods and has silently prostituted his powers to a commercialism. As our journal tries on these pages to portray the fundamental laws of physical chemistry and commerce it seems to us the best of our ability, so we are glad to record our views on the present evils of an "immoral wealth." Honesty is the only basis of human happiness.

William B. Rankine.

In our November issue we were sorry to report the death of Mr. William B. Rankine, to whom Niagara Falls owes so much. While in that issue we already gave a short sketch of his life and work, we are now enabled, through the courtesy of a personal friend of the deceased, to publish his portrait with the following appropriate notes on Mr. Rankine as a man:

It was Mr. Rankine's foresight which conceived the immense industrial importance of "Niagara harnessed," and it was his and Mr. Edward D. Adams' energy and foresightedness that provided the capital necessary for this great enterprise. Before it had been shown that electrical horse-power on a large scale could find a ready market, millions of dollars had been sunk in the gigantic wheel-pit and tunnel of this great company. Not only was capital timid in advancing money for such a project, but even the engineering talent at that time was somewhat afraid that power could be commercially developed on so large a scale.

With the completion of the power plant at Niagara the engineering end of the plan proved successful in every point and vindicated the confidence of capital. There remained the question of a market. It had been expected by many that cheap horse-power involved in the plan would attract great textile mills and other factories of a similar character to Niagara. But the true market for big blocks of power came in the establishment of entirely new industries. It was the Pittsburg Reduction Co., making aluminium, the Carborundum Co. and the Union Carbide Co. who became the great consumers of this power, thus creating an almost entirely new field of industry.

WILLIAM B. RANKINE.

Mr. Rankine's connection with the electrochemical world has always been intimate. Under his supervision the power lands at Niagara became covered with electrochemical factories. An opportunity was offered manufacturers and inventors to work out, on a commercial scale, plans which had hitherto been confined to laboratories. Mr. Rankine's stand, as encouraging the development of this class of industry, will always be held in remembrance by electrochemists.

Mr. Rankine was a man of immense activities. Since the successful utilization of Niagara's flow, Mr. Rankine has been actively engaged in building up the plant he fathered and in duplicating it on the Canadian side. He lived to see the Canadian plant finished and in operation.

Mr. Rankine had time left from his many businesses to cultivate the artistic side of his nature. He was well read in general literature, fond of practical charity, and devoted much of his time to church work, he being the chancellor of the Diocese of Western New York at the time of his death. His friends were innumerable, and he probably met more distinguished foreigners in the course of a year than any man in American private life. No distinguished party of visitors come to the United States that did not come to Niagara Falls, and coming to this city they invariably inspected the power house of the Niagara Falls Power Co. They met as host Mr Rankine, and America could hardly have had a more polished and cultured representative.

His activity is evidenced by the number of important positions he held. He was the second vice-president and treasurer of the Niagara Falls Power Co., Niagara Junction Railway Co., and Niagara Development Co., vice-president of the Canadian Niagara Power Co., vice-president of the Francis Hook & Eye & Fastener Co., of Niagara Falls; secretary and treasurer of the Cataract Power & Conduit Co., director and chairman of the executive committee of the Natural Food Co., of Niagara Falls; of the Cataract Power & Conduit Co., of Buffalo; of the Tonawanda Power Co., and of the Suburban Power Co., secretary of the Tesla Co., a director of the Bell Telephone Co., of Buffalo; a director of the Ramapo Iron Works, Niagara Tachometer & Instrument Co., and Niagara Research Laboratories, and a trustee of the Equitable Trust Co., of New York. Mr. Rankine was a trustee of De Veaux College and a life trustee of Union College, a member of the Buffalo Club, of Buffalo, and of the University, Metropolitan, Lawyers and Alpha Delta Phi Clubs, of New York City; of the Bar Association of the City of New York, and of the New York State Bar Association.

Electric Smelting of Magnetic Iron Ore.

As was already mentioned on page 445 of our November issue, 1905, a series of tests have recently been made by Dr. David T. Day, of the United States Geological Survey, on the utilization of black sands and magnetic iron ores obtained from various points of the Pacific Coast. Dr. Day makes the statement that conditions for the electric production of steel are fully as good in Oregon as they are in Germany. From Dr. Day's preliminary report the following notes are taken:

The electrical experiments were made by Mr. C. E. Wilson. He brought from the East twenty-five carbon electrodes—each 48 inches long and 4 inches square—such as are ordinarily used in electric furnaces. The rest of his equipment was obtained in Portland from materials kept in stock or easily made at a foundry.

The first experiments were made with a "small furnace." In building this furnace a course of ordinary Carnegie firebricks was laid upon the ground. Upon this single course was laid a cast-iron plate, five-eighths of an inch thick, 3 feet long and 3 feet wide. On this was placed an oval sheet-iron drum, of No. 16 iron, 3 feet long by 3 feet high. The sides of this drum were lined with fire-bricks, to form a crucible 18 by 18 inches and 24 inches high. The bottom of the crucible was covered from the cast-iron plate up to the tapping hole, with broken carbon electrode. The carbon electrode to carry the current was suspended by a pulley above this furnace and connected with a balanced axle and wheel, by which it could be readily raised or lowered. The top of the furnace was covered with two double plates of riveted wrought iron, between which cold water was run. In the center of this water-jacketed cover an opening was left sufficient to allow the free play up and down of the carbon electrode.

From a transformer sub-station of the Portland Electric Co., alternating current of 1,000 to 2,000 amps., at 50 to 20 volts, were available at the terminals of the furnace.

The first run was made on Oct. 17. The charge was made up of a mixture of magnetite, coke and lime. This consisted of 200 pounds of magnetite, obtained from the sand at Hammond Station, near Astoria, Ore., at the mouth of Columbia River (analysis Fe_3O_4 79.06, TiO_2 16.00, MnO_2 2.45, silica, moisture and undetermined matter 2.40); 44 pounds of "Fairfax" coke, containing about 25 per cent of ash, and 24 pounds of lime. About 150 pounds of this charge was slowly introduced into the furnace, and within an hour there was tapped from the furnace 70 pounds of steel and slag which contained 8 per cent of iron.

On the following day the furnace was again heated, and filled with a mixture similar to that used on the first run, except that it contained less lime. Steel was successfully cast twice, making, for that day's run of 2 hours, a product of 90 pounds of steel from 300 pounds of iron ore.

The heat was sufficient to keep the entire slag in a fluid state, whether much or little titanic acid was present. No titanium went into the iron.

The slags first obtained consisted of fused iron silicates, fused oxide of iron and silicate of titanium. Later in the experiments these slags grew lighter in color and in specific gravity. It became possible also to lessen the quantity of slag produced, which was unduly large, owing to the great quantity of ash in the coal. The coke used showed an analysis 41 per cent of ash. It is difficult to procure, in that locality, coke that is well adapted to metallurgical needs.

Experiments with the small furnace having been successful, it was thought desirable to build a larger furnace, with thicker walls, in which higher temperatures might be obtained and maintained. An iron plate, 2 inches thick, 5 feet wide and 6 feet long, was therefore procured and laid upon two courses of fire-brick, to form the base of a furnace, on which was set a wrought-iron cylindrical shell one-quarter of an inch thick, 5 feet in diameter and 4 feet high. This was lined with fire-brick, the bottom having the usual lining of one course of carbon electrode bricks 4 inches in diameter. Two carbons clamped together with a water-jacketed head or clamp formed the electrode for introducing the current. The voltage was run up as high as possible—that is, from 75 to 90 volts, the limit of the current obtainable over the wires. In all respects, except these mentioned, this second furnace is identical with the first.

Iron ore from Aptos, Bay of Monterey, Cal., was smelted in this furnace on Nov. 10. This iron ore is very fine grained and contains a notable percentage of manganese. It is not so rich in titanium as the other sands that had been used. From the start this furnace made a satisfactory run, maintaining easily a high temperature and turning out a very smooth product. After a few trials the slag became as light in color as that from any well regulated blast furnace. The later products of steel were much denser than those first made, which would seem to indicate that, at the higher temperature, the process of reduction is complete, even in the short time that elapses between the beginning of reduction and the tapping. In every case, however, small blow-holes were observable in the steel. These were due to gases which formed wherever grains of magnetite were still entangled in the steel in process of reduction. The capacity of this furnace with a current of 125 volts, 1,200 amps., would be 2,000 pounds in 24 hours.

From the daily records of both furnaces it appears that the energy consumption per unit of weight of steel produced varied greatly with the amount of the charge and other factors; 1-hp. day produced from 2 to 24 pounds of steel.

News and Notes.

American Electrochemical Society.—At the meeting of the board of directors of the American Electrochemical Society, held on Dec. 2, in Philadelphia, the following gentlemen were elected members of the society: Louis Ruhl, New York; Arthur Haug, Peekskill, N. Y.; H. A. Prosser, Elizabeth, N. J.; A. W. Lewin, New Orleans, La.; Francis R. Pyne, South Bethlehem, Pa.; Harry R. Lee, South Bethlehem, Pa.; John S. Bridges, Jr., Boston, Mass. The names of the following gentlemen will come up for election at the January meeting of the board of directors: M. Fukuda, Japan; Joseph H. Goodwin, Penn Yan, N. Y.; Chas. S. Palmer, Upper Montclair, N. J.; Albert Petersson, Alby, Sweden; Washington Devereux, Philadelphia, Pa.

Franklin Institute.—In a section meeting of the Franklin Institute of Philadelphia, on Dec. 2, Mr. S. S. Sadtler presented a paper on new laboratory methods and appliances, and Dr. Henry Leffmann a paper on analytical [illegible] Lehigh University, lectured on electrochemical calculations.

Chemists' Club.—The Chemists' Club, of [illegible] elected, at its annual meeting on Dec. 13, the following [illegible] for the coming year: William Jay [illegible] president; Reginald C. Woodcock and Victor G. Blode, vice-presidents; Charles Baskerville, secretary; Maxmilian Toch, treasurer; also for trustees, terms expiring 1909, Alexander C. Humphreys and Carl H. Schultz.

Palladium.—The uses of palladium are few. It is introduced at present into use as a "contact" material in the automatic gas igniters now coming into vogue. A pellet of asbestos is covered with a mixture of finely divided palladium with platinum black; as soon as illuminating gas strikes the pellet, combustion is brought about. Gas-lighting engineers are making the most of this device, since it spoils the argument of electrical engineers that the electric lamp is the only one which needs no match. Palladium has the lowest melting point of any of the platinum metals, 1,500° C.; it is more malleable than platinum, is steel-white in color, and does not tarnish. On account of its resistance to oxidation and to the action of hydrogen sulphide, it is used in the construction of the mechanical parts of fine watches, fine balance beams, scales on delicate apparatus, etc.

Niagara Falls.—The Pittsburg Reduction Co. has contracted with the Niagara Falls Hydraulic Power Co. for the delivery of 27,000 hp., and has also leased 4 acres of land extending along the river bank. The Acker Process Co. is stated to intend enlarging their plant by 30 per cent, and to negotiate with the Niagara Falls Hydraulic Power & Mfg. Co. for the increase of power.

Mining in the Butte District.—The annual output for 1905 (the month of December being estimated) by the leading mining companies of the Butte district amounted to 5,110,000 tons. From this total tonnage of ore in excess of 338,000,000 pounds of copper was produced. Of this amount the Amalgamated Company is credited with 264,000,000 pounds; United Copper, 30,000,000; Clark, 20,000,000; North Butte, 20,000,000. The Amalgamated also obtained as by-products 9,000,000 ounces of silver and 60,000 ounces of gold; United Copper, 2,000,000 ounces of silver and 6,000 ounces of gold; Clark, 1,000,000 ounces of silver and 1,600 ounces of gold; North Butte, 1,500,000 ounces of silver and 1,800 ounces of gold. The output of the [illegible] smelter of the Amalgamated Company was 14,000,000 pounds of copper a month, and the Boston and Montana smelter 8,000,000 pounds a month. Ninety-two per cent of all Amalgamated ore yielded 4 per cent copper and 8 per cent of it went 8% per cent in copper. During the last six months the Anaconda ore averaged better than 3 per cent.

A Remarkable Engineering Achievement.

By J. Ramsay Wilson.

A notable piece of engineering was recently completed at Niagara Falls, Ont. The Park & River Railway Co., finding that the amount of water passing through their canals was [illegible] to supply the [illegible] Seeing that the in-take of the company is [illegible] the river from the point of the junction has had to be excavated, this part of the work, it will be seen, is [illegible]

After consideration it was resolved to build a concrete [illegible] sluice and tapping the river. A [illegible] was erected of sufficient height to allow the [illegible]

when tipped, to lie about 15 feet from the shore, and on top of this was erected a concrete pillar 50 feet high and 6 feet 1 inch square. In order that the dam might reasonably conform to the bed of the river, there were inserted every 8 feet or so wooden wedges extending half-way through the column, and to insure the pieces remaining together a heavy chain was

FIG. 1.—CEMENT TOWER STANDING.

embedded in the center, due allowances being made at the joints.

On Nov. 9, after a month's drying in the air, at 3.08 p. m., hydraulic jacks were inserted below the timber base, and shortly after, the huge mass, weighing about 300 tons, com-

FIG. 2.—MOMENT OF STRIKING THE WATER.

menced to tip, and about 4.35 p. m. fell into its place with a grand splash.

Immediately after soundings were taken, and it was found the water in the intake had been raised 10½ inches, as was anticipated.

Fig. 1 shows the cement tower standing before it was

FIG. 3.—PORTIONS LYING IN THE WATER.

dropped into the rapids. Fig. 2 is a snapshot of the tower just as it struck the water. Fig. 3 is a picture of the portions as they lie in the water.

Great credit is due to Mr. Isham Randolph, C. E., Chicago, who planned and carried through this novel piece of engineering to a most successful finish.

CORRESPONDENCE.

Water-Cooling of Furnace.

To the Editor of Electrochemical and Metallurgical Industry:

Sir.—I have read with considerable interest your editorial in the December issue on "Novelties in Water-Cooling Furnaces." Such ideas as you have expressed in it are in line with the general subject of refractory materials, in which subject, I notice, your journal has evinced much interest during the past year.

Without doubt it is a matter of great importance to metallurgy to have our knowledge systematized about the engineering construction of furnaces, and especially of the refractory material with which the molten or hot charge is in contact. In the zinc business, for instance, it can be almost laid down as an axiom that the fire-clay and general management of the pottery must be "like Cæsar's wife, above reproach."

In any metallurgical furnace, care expended upon the subjects of proper refractory materials and the metallic backing which holds them in place is most important. A furnace that will endure a long campaign means that the cost of repairs and renewals in the cost sheet is kept a low figure. Besides, we have a general feeling of satisfaction among all the workers of the plant at the continuous and successful operation of the furnace.

This general subject I studied with some care while metallurgist of the Lanyon Zinc Co. where I had at my disposal a research laboratory in which to test out practically my theories. In addition, while associated with Mr. Fred. W. Gordon and Mr. E. A. Uehling, while in the employ of the Lungwitz Reduction Co., I had the pleasant opportunity of hearing these two pioneer metallurgists attack the subject in the light of their own varied and prolific experience.

The ideas you express apropos of the subject of water cooling is about the result of their own experience, and I have learned from them that there is no better way of preserving the walls of the furnace than cooling the plates, against which the bricks rest, with a stream of water. In fact, the gentlemen tell me that on many occasions they repaired the bosh of an iron-furnace by simply putting in a plate and turning on the hose against this plate. Such a repair is extremely efficient and shows the general lines along which work should be carried on.

In a letter I sent to your journal in June, 1905. (Vol., III., p. 214) I stated that it was impractical, as a general principle, to water-cool an electric furnace. This I have now learned is only a "part-truth," and I now believe that where a large continuous electric furnace is in operation with an enormous output in tons of material per cubic foot of area, the number of heat units lost per ton of charge treated should not amount to over 5 or 10 per cent of the total energy required for the operation. For electrical purposes water cooling would be somewhat dangerous, but with low voltages this difficulty would not be any great factor. Where it is, it would be quite possible to cool with crude oil and carry by a pipe of insulated material the oil to a coil of iron pipes in which the oil could be cooled.

Another interesting development in the line of refractory materials for metallurgical furnaces would be, in my opinion, the use of Acheson graphite for the tap hole for either the slag or the metal. While this would not stand any rough mechanical treatment of the drill or tap bar, yet under certain conditions this material is practically indestructible, and I have used it with success for running slags.

I note that Mr. Hiram W. Hixon has used what he calls electrode carbon in his work at the smelter of the Mond Nickel Co., of the Victoria mines, of Ontario.

New York. WOOLSEY McA. JOHNSON.

The Injurious Effect of Acid Pickles on Steel.

BY PROF. CHARLES F. BURGESS.

A most interesting phenomenon attendant upon the pickling of iron and steel articles by the use of acid solutions is the influence which such treatment exerts upon the physical properties of the metal. The attention of the metal worker is called to this repeatedly by the ruin of steel springs and other tempered articles, which, accidentally or by design, have been subjected to the action of acid solutions.

While for many years it has been known that when an acid attacks an iron article it weakens it by dissolving some of the iron, it is not so generally recognized that there is a much more serious and a far deeper seated effect, resulting in an alteration of the physical properties of the iron below the surface. The influence of the acid penetrates to some depth, so that the temper of steel may be destroyed by a few moments' immersion, and the tensile strength of untempered iron is usually diminished. This phenomenon is commonly designated by the practical worker as "rotting." It was first brought to the attention of the writer in the course of removing scale from a short length of steel wire by immersing it in a dilute sulphuric acid solution. Before immersion, this wire could be bent in a circle of less than 1 inch diameter. The jar containing the acid solution was of such dimensions that the wire had to be bent in an arc of about 8 inches diameter, the ends being near to the bottom of the vessel. While noting the action of the acid for a few seconds, the wire suddenly snapped in two, throwing some of the solution into the eyes of the watcher, thereby reducing him, for the rest of that day at least, to a mood more conducive to contemplation than to observation.

Newman (Metallic Structures, Corrosion and Prevention) speaks of this phenomenon, and refers to the investigations made by Ledebur as being an excellent experimental study of the subject. Among other things Ledebur has shown that by pickling various kinds of iron in a 1 per cent sulphuric acid solution, the susceptibility of the iron to become brittle by pickling is least with cast iron and silicon steel and most with wrought iron and high carbon steel. Combined carbon seems to increase the action and silicon to diminish it.

The effect of sulphuric acid upon iron, by which the metal is made brittle, is a subject that has also been investigated by Prof. Hughes and others, reference to whose work will be made later.

In the process of wire drawing, where the rods are frequently annealed and pickled, it has long been recognized that the strength of the metal gradually decreases with the length of the process. This, by some, has been ascribed to the decarbonization which the air exerts upon repeated heating. It is more probable, however, that the acid treatment is alone responsible for such deterioration.

It is a fact commonly accepted that galvanized iron wire, in spite of its increased section, is never so strong as the black wire from which it is made. Some investigators have held that this is due to the alloying of the zinc with the iron, but such a view is contradicted by others, and from investigations made by the writer it would appear that such alloying is extremely slight and quite insufficient wholly to account for the marked weakness produced. The heat treatment may exert some influence in the weakening process, but it seems more likely that if the pickling which is done prior to the immersion in the molten zinc bath could be dispensed with, galvanized wire would be quite as strong as black wire, barring, of course, any alteration of the temper incident to the heat treatment.

In the Transactions of the American Society of Mechanical Engineers (Vol. IX., 1888, p. 715) may be found an interesting discussion of the question, "To what extent does electroplating reduce the temper of highly hardened steel?" In this discussion it was held by some of the speakers that the idea that steel could be injured in the electroplating process was a fallacy, and it was maintained that to destroy a temper imparted to steel by heat treatment a [illegible] is necessary, and that, therefore, nickel plating, which involves no such treatment, cannot be [illegible]. Others claimed that certain [illegible] is liberated upon the metal [illegible], and it was stated that the acid treatment [illegible] to steel in the course of preparing it for the reception of a nickel deposit, was largely responsible for the deterioration which took place.

Steel springs, such as are used in [illegible] isms and the like, steel tape, piano wire, and many other tempered steel articles having but a small section dimension, cannot be subjected to the ordinary plating processes on account of the deterioration which takes place in the [illegible] such as these were called repeatedly to the attention of the writer, eventually leading to an experimental investigation into the underlying causes and the extent of the deterioration produced by different methods of treatment, the hope being, to determine if possible, some satisfactory way of obviating the difficulty.

The concensus of opinion among those who have investigated the matter is, that the alteration of the physical properties of iron and steel by the action of acids is due to the hydrogen which is liberated. The only dissension from this view is that which holds that the acid itself penetrates into the metal through capillary or other force, the destructive action being thus continued after the metal has left the pickling solution.

That hydrogen has a capacity for exerting a marked influence on the properties of iron is demonstrated in the electrodeposition of that metal. The iron thus produced has associated with it, either in chemical or physical combination, a relatively large quantity of hydrogen. The gas may be expelled by bringing the metal to a red heat, but this expulsion involves a change in the property of the iron, which from being hard and brittle becomes soft and ductile. An apparent anomaly is the fact that steel possessing certain physical properties may be exposed to hydrogen gas under varying conditions of temperature and pressure without undergoing any material physical changes and without absorbing any appreciable quantity of the gas. If, however, the metal is exposed to hydrogen under that peculiar condition of activity, commonly known as the nascent state, the absorption of hydrogen becomes most marked. The action of acid upon iron liberates hydrogen in this peculiar condition of activity, and the iron therefore absorbs the gas.

This absorption of occlusion of hydrogen by iron has been made the subject of much research. The gaseous element is usually present, though in almost immeasurable quantities in all forms of iron. Like all other fluids, molten iron is capable of absorbing gases, and the solidified metal retains some of the gases thus taken in. Ledebur determined the amount of hydrogen which would be retained by solidified iron, and found that Martin iron (poured) containing 1 per cent of carbon and .14 per cent silicon retains .0017 per cent by weight of hydrogen. Graham (Collected Works, p. 279) found hydrogen able to diffuse through a red hot iron pipe, and on cooling this down in an atmosphere of hydrogen the metal occluded 2 per cent of its own volume thereof. Cailletet (C. R., LXXX., p. 319) by depositing metal from its ferric chloride solution obtained metal hard enough to scratch glass and containing .028 per cent of hydrogen. Johnson (Proceedings of the Royal Society, 1875, Vol. XXIII.), Hughes (Journ. Soc. Tel. Eng., 1880, p. 103), Backher (Zeit. d. Ver. deutsch. Ing., 1888, p. 1861), and Ledebur (Stahl und Eisen, 1885, p. 745 and 1888, p. 743) found that iron corroded by [illegible] hydrochloric acid absorbs hydrogen. Graham [illegible] observes that when hydrogen is [illegible] the gas can diffuse through sheet iron and [illegible] in detective places, such bubbles being [illegible]

siderable pressure. Sir Roberts Austen (Fifth Report of Alloys Research Committee, Inst. of Mech. Eng., 1889) described the marked influence of hydrogen in electrolytic iron.

To test the correctness of the theory that the deterioration of steel in a pickling bath is due to the hydrogen which is liberated and also to find, if possible, a practical method for preventing such deterioration, was the incentive which led to an experimental investigation of the subject in the electrochemical laboratories of the University of Wisconsin. Granting that the hydrogen liberated upon the iron is alone responsible for the rotting produced in the pickling process, the remedy would, naturally, be sought in some method which would suppress the hydrogen and at the same time would not interfere with the effectiveness of the pickle in removing the scale. Various methods for attaining such a result suggested themselves, among them being the following:

The use of nitric acid of such strength as to liberate no hydrogen.

The use of acid solutions containing an additional material capable of being readily reduced by the liberated hydrogen.

The use of arsenic in the solution.

The employment of a bath containing no hydrogen.

The employment of the electric current.

Nitric acid presents objections as a pickling material for the removal of scale on account of the slight solubility which it has for black iron oxide, the tendency which iron has to become passive in solutions of certain concentrations and the absence of liberated hydrogen which would serve physically to remove the loosened scale. The reason that no hydrogen bubbles are liberated during the action of nitric acid upon iron is that the hydrogen which is replaced by the iron reacts upon the NO_3 radical, reducing it to a lower oxide. In other words, the acid itself acts as a depolarizing agent as it does when employed in a primary battery.

Materials other than the acid itself may act as the depolarizer, such being illustrated by copper sulphate, sodium nitrate and the like.

Arsenic in an acid solution exerts a remarkable influence upon the rate at which sulphuric or hydrochloric acid will attack iron. This was pointed out over half a century ago by E. Millon (*Comptes Rendus*, 1845, Vol. XXI., p. 37), who showed that the presence of certain salts and other impurities in an acid solution exert a marked effect. While iron turnings are readily acted upon by dilute sulphuric acid solution, a few drops of platinic chloride greatly increases such activity. On the other hand, arsenious acid arrests all action, and it is claimed that the iron may be left for several months in a solution containing a few drops of anaqueous solution of As_2O_3 without any appreciable action taking place. The effect of arsenic upon hydrochloric solutions is said to be similar. A Meurice (*Bulle. de l'Asso. Belge. Chemists*, Vol. IX., p. 343) states that a good and a bad acid is determined by the percentage of arsenic which it contains, claiming that an acid containing .5 per cent of arsenic will clean iron wire well without effervescing and give a wire of even gauge which will not break in drawing, while with a 1 per cent solution the effect is quite the reverse. From results obtained by him it appears that the use of arsenic is advantageous in acid pickles, reducing the consumption of acid by one-half. This is contrary to the conception in which arsenic usually is held by the pickler, who generally considers its use decidedly disadvantageous on account of the discoloration and other defects which it produces upon the pickled work.

An example of a bath containing no hydrogen, and consequently introducing conditions impossible for the liberation of the gas, is one made of fused borax, such as is frequently used as a flux in the brazing process. It is possible that certain organic compounds might likewise meet this requirement.

The electric current, while facilitating the liberation of hydrogen at the cathode, has the opposite action at the anode, and by suitable regulation of its density the formation of hydrogen may be entirely prevented. By making the object exposed to the action of the pickling solution the anode, this suppression can be obtained.

Methods of Investigation.

In attempting to remove the thin black scale from some small tempered steel springs of a U-shape, it was found that all the ordinary acid pickles during a few moments' action produced such a rotting effect as to make the spring useless. This deteriorating effect was readily noted when an attempt was made to straighten the spring. Before having been subjected to the treatment, it could be pulled out straight without rupture, but after the acid had acted upon it rupture occurred during the straightening process.

After noting that different strengths of the ordinary acids always produced this effect various other pickles were tried, one of which was a solution of stannous chloride, such as is recommended by Brannt (*The Metal Workers' Handy-Book*, p. 246). Here it is claimed that iron may be cleaned of its oxide by immersion in a nearly saturated solution of chloride of tin, the duration depending upon the thickness of the film, 12 to 24 hours being sufficient. The solution should not contain too great an excess of acid, since in such case the iron itself will be attacked. It is recommended that the article, after its removal from the bath, should be washed in water and then in ammonia, this being followed by rapid drying.

Upon trial, a freshly prepared tin chloride solution was found to be effective in removing the thin scale from the steel spring, 1 to 2 hours being required for the operation. Whether this solution exerts a solvent action upon the scale or upon the iron itself, with a resultant loosening of the scale, was not determined, but it was found that the solution produced little brittleness. It was noted, however, that after the solution had been in use for some time, the rapidity of its action diminished, though this could be easily regenerated by the addition of a very small percentage of free hydrochloric acid.

To determine quantitatively the deteriorating influence of various solutions under consideration, it became necessary to devise other means for making observations. Two such suggested themselves. One of these has been referred to previously. It consists in bending a steel wire or a flat strip in the manner shown in Fig. 1. The support holding the bent wire is immersed in the solution, the action of which is under test. After a certain length of time, depending upon the chemical composition and strength of the solution, the wire becomes so brittle as to break. By observing the length of time necessary to produce this effect in different solutions the relative deteriorating effects of each can be approximately stated as being inversely proportional to the periods thus measured. While this method gave good results, its practice was not found satisfactory, on account of the length of time required to take each measurement and the necessity of careful watching in order to note the exact moment of rupture. The testing of one solution required a number of measurements, since different pieces of iron cut from the same strip gave somewhat varying results, necessitating the taking of a number of such determinations from which an average may be secured.

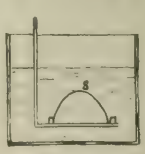


FIG. 1.—METHOD OF TESTING.

A more satisfactory method of making such tests consisted in determining the amount of flexure which a strip of steel will stand before and after being subjected to various corroding influences. Fig. 2 shows a testing device which was designed and constructed for this purpose. The piece of wire or steel strip is held in the manner shown, and by steadily turning the screw head until rupture occurs and noting the number of turns which may be taken, a determination of the amount of bending which the strip will stand is thus secured. In

making the tests, the screw is turned as uniformly as possible, since if it is turned by jerks or is left stationary at any position for an interval of time, considerable variation in the results are obtained. The distance between the two outside knife edges of the apparatus is ½ inch, and consequently small samples may be tested, or, on a piece of wire 6 inches long several measurements may be made and the average of these secured. The screw has a pitch of 50 per inch. Five complete turns of the screw, therefore, cause a deflection of one-tenth of an inch between the middle and the two ends of a ½-inch length. In using this piece of apparatus the decrease in flexibility was taken as being a measure of the deteriorating effect of hydrogen, and it was assumed that this decrease in flexibility was approximately proportional to the difference between the number of turns of the screw required to break the sample before and after its subjection to the action of the solution under test. These assumptions are not entirely justified, since the flexibility is not strictly proportional to the distance through which the central knife edge moves. This error is so small, however, as to be masked by other experimental errors.

A steel ribbon was used in preference to wire, as reducing such errors to a minimum. This ribbon was used in the investigations, the results of which follow: "It was obtained

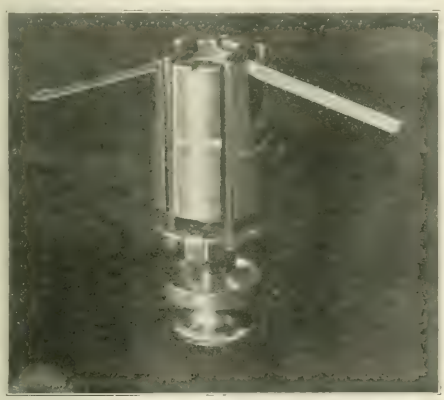


FIG. 2—TESTING SHEET.

directly from the factory, being a high-grade spring steel, untempered, 5 to melt well and .070 inch thick. It carried a thin black scale. Pieces 7 to 8 to 12 inches long were used in making the tests, and these were immersed half their length in the solution the action of which was to be observed.

While this steel strip has the appearance of being uniform, and by means of a micrometer was found to have uniform dimensions, when subjected to the bending test some variations between samples cut from the same strip were discovered.

To state quantitatively, or to denote units the amount of deterioration which is produced in a given time under conditions is, of course, impossible, and the term "percentage deterioration" will be employed to designate measurements which show roughly the amount of deterioration by means of which comparison between the different samples may be made. The percentage deterioration was derived in the following manner: The samples upon which measurements were made were all cut from the same coil of steel ribbon, pieces 8 to 12 inches in length being used, and these were immersed vertically to a distance of about 6 inches in the solution under test. Three or more breaking tests were made upon the exposed and upon the unexposed portions of each sample and the average of each reading calculated. The difference between the values thus given for the two ends show approximately the amount of brittleness. The percentage which this difference bears to the number of turns required to break the unexposed part of the specimen is the "percentage deterioration." The procedure is not entirely satisfactory, as the deposition of the gas on the steel while removing the scale would somewhat offset to increase the apparent flexibility, as there is a point where the deteriorating action [illegible] counteracted by the increase in flexibility due to the removal of the scale. The error in this respect would, however, be small, and probably offset the [illegible] effect of [illegible] the measurements of the last angle to be [illegible] would not be considered, except in cases where nitric acid was employed.

RELATIVE EFFECT OF VARIOUS ACIDS.

For the purpose of determining which of the three common acids, sulphuric, hydrochloric and nitric, produce the least amount of brittleness, the following measurements were made. Normal solutions of these three acids were used. Samples of the test ribbon were immersed for various lengths of time and the percentage deterioration in each case measured. It was found that the nitric acid had the least satisfactory effect in removing the scale, though it dissolved the iron at a much greater rate than did the other two acids:

Acids Used.	Time of Immersion, Minutes.	Per Cent Deterioration.	Weight of Iron Dissolved.	Loss
n H_2SO_4	10	3		
n HCl	10	50	Grams.	Grams.
n H_2SO_4	15	51	.0810[illegible]	.0[illegible]17
n HCl	15	67	.0810[illegible]	.0[illegible]14
n HNO_3	15	15	.0[illegible]8	.0[illegible]34
n H_2SO_4	20	55		
n HCl	20	69		
n H_2SO_4	30	57		
n HCl	30	68.5		

From this data it may be concluded that sulphuric acid produces less deterioration than does hydrochloric, though the difference between the two is not great. The brittleness is shown to be produced mainly during the first few minutes of immersion, the values upon longer immersion remaining somewhere near a constant. Some gas was evolved in the nitric acid solution, though not so much as in the others. The 15 per cent deterioration given for this acid is probably too low, since the metal which was dissolved made the strip thinner, and therefore capable of a greater amount of bending without rupture. In fact, if the strip had been left longer in the nitric solution, the bending of the corroded sample would have become greater than that of the uncorroded piece, as is brought out in the data which follows:

	Percentage Deterioration							
Minutes Immersed	1	3	5	10	15	20	30	60
n H_2SO_4	10	38	48.2	25.0	[illegible]	[illegible]	[illegible]	[illegible]
n HCl	68	94	82.0	44.0	[illegible]	[illegible]	[illegible]	[illegible]
n HNO_3	11.0	12.7	5.3	0.0	[illegible]	[illegible]	[illegible]	[illegible]

It is seen [illegible] acid produces less brittleness than does hydrochloric; that the lower amount of brittleness in the nitric acid bath is [illegible] to lack of hydrogen evolution [illegible] the fact that the rate of the dissolution of the metal is much greater, the deposition of hydrogen [illegible] as shown by the second measurement, [illegible] case [illegible] the other acids was about [illegible].

[illegible] accidental effect or whether

Influence of Arsenic.

It has been pointed out previously that the presence of arsenic causes sulphuric acid to act much less rapidly than does the pure solution, and it was sought to determine whether this would decrease the amount of brittleness as well as the rate of attack by the acid. A five normal solution (containing about 245 grams of acid per liter of solution) was employed. To a portion of this solution was added a very small quantity of arsenic in the form of arsenious acid. The resultant data follows:

Percentage Deterioration.

Minutes of immersion....	3.	10.	15.	60.
5n.H_2SO_4	52.0%	54.8%	61.0%	64.0%
5n.H_2SO_4 + As_2O_3.....	.8	8.0	2.0	9.2

These figures show most strikingly the advantage which the presence of arsenic in such pickling solutions affords. The scale was removed completely by either of the solutions and apparently with equal facility. It is frequently urged by practical picklers that arsenic is a material to be rigorously avoided in making up pickling solutions, owing to the undesirable surface which, they claim, it leaves after the removal of the scale. Such objection to its use was not borne out by the experiments cited. Indeed, the solution containing arsenic produced a surface even somewhat smoother than that left by the other pickle.

It was noticed that in the solution containing arsenic there was almost no evolution of gas, in which respect it differed from the arsenic-free solution. Measurements were then made to determine the relative rate of attack of the two solutions in question. The surfaces to be exposed to the acid were cleaned with emery cloth. The exposed surface comprised 3.75 square inches. The resultant data follows:

Time of Immersion.	H_2SO_4Aq	$H_2SO_4Aq + As_2O_3$
20 minutes.....	.0472 gram loss	.0052 gram loss
60 minutes.....	.2060 gram loss	.0022 gram loss
Total	.2532 gram loss	.0074 gram loss

These figures show the marked retardation of corrosion where arsenic was employed. On diluting the above solutions by one-half and removing scale from the samples by immersing, the losses in weight, including the weight of the scale which was removed, were .0252 grams and .0187 grams respectively, the greater loss being that noted where the pure solution was employed. The smoothness of the surface which was treated by the solution containing arsenic was much more pronounced and the brittleness of the sample much less than in the other case.

Just what is the reason for the protective effect produced by the arsenic I shall not attempt to say. Certain it is that the action of a small percentage of it in pickling solutions is remarkable. It is possible that certain other impurities might be introduced to even greater advantage, and this is a matter worthy of further investigation. The presence of antimony in a sulphuric acid solution seems to have somewhat the same effect as that exerted by arsenic, though in a lesser degree, while nitrate of bismuth has an effect only slightly less than that given by the arsenic, so far as the few measurements taken in this connection demonstrate.

Mellon claims that the effect of arsenic is noticeable equally in nitric and in sulphuric solutions. Trials were made to determine whether the arsenic produced as pronounced an effect in a five normal hydrochloric acid solution as it did in a sulphuric acid solution of the same strength.

Percentage Deterioration.

Time of Immersion.	5n.HCl	5n.HCl + As_2O_3
5 minutes.....	60.4%	40.0%
10 minutes.....	71.7	53.9

Investigation showed that the tin chloride solution, the efficacy of which as a means for removing scale from tempered steel, has been pointed out previously, produced a deterioration of about 3 per cent upon the standard steel strip after an immersion of two hours' duration. To determine whether the tin itself exerts a beneficial action, a stannous chloride solution was added to the five normal sulphuric acid pickle. It was found that the deterioration was reduced in a 5 minutes' exposure by from 55 per cent to 18 per cent, and that the tin, therefore, acts somewhat in the same manner as does arsenic.

It occurred to the writer that an aluminium chloride solution might be effective also, and it was found, after making such an experiment, that spring steel could be cleaned in about 4 hours without appreciable injury to its properties. On continued immersion for several days, however, the steel became rotten. The action of the stannous chloride and the aluminium chloride solutions appeared to be somewhat similar, the advantage being in favor of the first named. It was found, however, that while the scale is completely removed the surface is not left bright, so that a subsequent scouring or other treatment becomes necessary in order to make it ready for an electro-deposit.

Brittleness Produced by Nickel Plating.

It has been suggested that the process of nickel plating is responsible for the deterioration produced in various kinds of steel and iron articles, and that such deterioration is to be traced almost directly to the effects of the pickling that is necessary. It was decided to make further tests to determine, if possible, whether the ordinary nickel plating bath plays its part in such deterioration. These tests were made by preparing three samples of spring steel for receiving a nickel coating, the following methods being employed: Sample No. 1 was freed of scale by rubbing it with emery cloth. No. 2 was pickled in the ordinary sulphuric acid solution. No. 3 was immersed in a pickle of similar nature, to which had been added a small amount of arsenious acid. The samples were then plated in the nickel ammonium sulphate solution, and were afterwards tested after the manner described above. The results obtained follow:

Percentage Deterioration in the Nickel Plating Process.

Cleaning With Emery.	H_2SO_4Aq	H_2SO_4 — Aq —As_2O_3
13.2%	18.6%	4.8%
6.9	10.5	4.8
9.3	18.6	7.2
11.9	16.2	12.0
14.3	14.0	4.8
15.5	....	7.2
Ave.. 11.85	15.58	6.8

Sample No. 2 showed the greatest amount of deterioration, and while the deterioration of No. 1 was less, yet it showed an appreciable weakening, demonstrating that the nickel plating solution in itself exerted a harmful effect on the thin steel strip. This was due, possibly, to the presence of a small amount of free acid in the solution. Whether a neutral solution would show the same effect is a question that was not put to test at the time, but it is probable that such an investigation would be of interest and, perhaps, of some practical importance. The sample which had been subjected to the pickle containing arsenic was weakened materially less than the piece which had been cleaned mechanically, and whether the arsenic not only decreased the harmful effect of the pickling solution itself but also that exerted by the plating solution, or whether the results as given may be traced to inaccuracies in the experiment, it is impossible to state without farther investigation. There seems to be little doubt, however, but that the entire nickel plating process produced harmful effects in certain

classes of work, where the section of the metal treated is so small as to make the affected part a considerable percentage of the whole.

SUPPRESSION OF HYDROGEN EVOLUTION.

The experiments above given, as well as many others which are not recorded here, seem to show almost beyond question that the brittleness or "rotting" of iron and steel is due to the penetration into its pores of hydrogen rather than of acid, so that any method which would prevent the liberation of hydrogen would do away with the usually consequent brittleness. If nitric acid of such strength that no hydrogen is evolved be used, a bent steel wire may be completely dissolved without breaking, while when other acids are employed the wire snaps in two before corrosion has proceeded to any appreciable degree. If sodium nitrate be added to a sulphuric acid solution the liberation of hydrogen can be prevented and it may thus serve a most useful purpose. An objection to its use, however, would seem to lie in the fact that the removal of the black oxide scale proceeds less rapidly in its presence.

Another experiment which was performed, but which was not original with the author, was the "annealing" of a piece of steel wire which had been made brittle through the liberation of hydrogen upon it. This brittleness had been produced by making it the anode in a sodium hydroxide solution from which the oxygen was liberated. The influence of this oxygen appeared to extend a slight distance below the surface of the iron, neutralizing the hydrogen which had been absorbed previously. In this way electrolytic annealing may be effected without the application of heat. Such a method is not of any practical value, however, on account of its incompleteness and slowness of action.

The suggestion has been made that the corrosion of acids may be advantageously replaced by electrolytic corrosion, a method which, by the way, has been made the subject of a number of patents. If a steel article from which a surface scale is to be removed be suspended as the anode in a dilute sulphuric acid or a sodium sulphate solution, corrosion may take place without the liberation of hydrogen. While this might be of advantage in preventing the weakening of the metal through the influence exerted by the hydrogen, it has its patent disadvantages, since it interferes with the chief purpose of the pickling process, *i. e.*, the removal of the scale. Since this removal is effected not so much by the actual solvent action of the pickle as by destroying its adherence to the underlying metal, a process which is assisted in marked degree by the liberation of the gas bubbles, it is usually disadvantageous to attempt a suppression of hydrogen liberation.

To avoid all necessity of pickling tempered steel springs, preparatory to plating, it may be possible that the tempering could be done in such a way as to prevent the formation of the oxide scale. With the idea of bringing about such a result an experiment was undertaken by covering the melted lead tempering bath with a layer of borax. Upon immersing a steel spring into the lead, through the layer of borax, the oxide was dissolved, and after subsequent immersion in oil and water a bright surface, ready for electroplating, was obtained. In some such way the disadvantages attendant upon the usual pickling process might be avoided.

The harmful effect which hydrogen exerts is an additional argument for the substitution of the sand blast for the ordinary pickling method, but its universal adoption can hardly be looked for on account of the greater expense that would be entailed, especially in the handling of small work.

Perhaps the most important feature brought out by the work here recorded is the demonstration of the marked influence which small quantities of impurities exert in acid solutions, while the fact that much additional work is necessary before a complete understanding of this will be available, is likewise shown.

Chemical Engineering Laboratories, University of Wisconsin.

Metallurgical Calculations.

BY J. W. RICHARDS, PH. D.
Professor of Metallurgy in Lehigh University.

ARTIFICIAL FURNACE GAS.

MOND GAS.

In the Mond producer, an [illegible] the heated air used for combustion: the [illegible] much colder than the ordinary producer, [illegible] dioxide is present in the gas. In fact, [illegible] ordinary producer gas, is very high in nitrogen (about 50 per cent.), very high in hydrogen (20 to 30 per cent.), very low in carbon monoxide (10 to 15 per cent.), [illegible] (40 to 50 per cent.), and carries as [illegible] undecomposed moisture. The fuel used is low-grade bituminous slack, and the object of using so much [illegible] the temperature in the producer so low that a maximum amount of the nitrogen in the coal is [illegible] gas. The gas is cooled to ordinary temperature by [illegible] water spray, so that all but a small amount of moisture is condensed, the ammonia is removed by dilute sulphuric acid, and the cold nearly dry gas is then used for gas engines or for furnaces.

The calorific power of the gas is not low, because the high proportion of hydrogen compensates for the low carbon monoxide, while the great heat evolved by the large formation of carbon dioxide has been mostly absorbed in decomposing steam, and is, therefore, potentially present in the gas in the form of hydrogen.

Problem 16.

Bituminous slack coal used in Mond producers for generating gas for a gas-engine power plant contained:

Moisture	8.60 per cent
Carbon	62.69 "
Hydrogen	4.57 "
Oxygen	10.89 "
Nitrogen	1.40 "
Ash	10.42 "

Calorific power, determined in a bomb calorimeter, water condensed, 6,786 Calories per unit of dried fuel. Ashes produced 208 pounds per ton (2,240 pounds) of moist slack used, contains 12 per cent of carbon.

Air used for running is heated to 300° C. by the waste heat of producer gases, and carries in 2[illegible] lbs of water as steam (at same temperature) for every ton of fuel burnt. The steam is generated by a tubular boiler run by the escape gases from the gas engines, but is heated from 100° C. to 300° C. by the waste heat of the producer gases. The latter escape from the producer at 350° C.

Composition of waste gases passing out of condensers at 15° C.:

Carbon monoxide (CO)	11.0 per cent
Hydrogen (H^2)	27.5 "
Marsh gas (CH^4)	2.0 "
Carbonic oxide (CO^2)	16.5 "
Nitrogen (N^2)	41.3 "
Water vapor (H^2O)	1.7 "
	100.0

Assume all the nitrogen of the fuel to [illegible] ammonia gas, NH^3.

Required:

(1) The calorific power of the Mond gas.
(2) The volume of gas produced per [illegible]
(3) The efficiency of the producer.
(4) The weight of steam which is decomposed [illegible] producer.
(5) The proportion of the calorific power of the [illegible] to the gas by the decomposition of steam.

(6) The proportion of the heat generated in the producer which is saved to the gas by the decomposition of steam.

(7) The proportion of the calorific power of the coal lost from the producer by radiation and conduction.

Solution: (1) One cubic foot of the gas at 0° C. would generate the following amounts of heat (water uncondensed).

$CO = 0.110$ cubic feet $\times 3062 =$ 336.8 ounce Calories
$H^2 = 0.275$ cubic feet $\times 2613 =$ 718.6 " "
$CH^4 = 0.020$ cubic feet $\times 8598 =$ 172.0 " "

Total = 1227.4 " "
= 76.7 pound "
= 138.1 B. T. U.
Per cubic meter = 1227.4 kilo. "

If measured at 15° C. (60° F.) the above values will be reduced by the factor $273 \div (273 + 15)$, and become

Per cubic foot = **72.7** pound Calories
Per cubic meter = **1163** kilo. " (1)

(2) Carbon in 1 cubic foot of gas at 0° C.:

In CO 0.110×0.54 ounces
In CH^4 0.020×0.54 "
In CO^2 0.165×0.54 "

0.295×0.54 " = 0.1593 ounces
= 0.009956 pounds

Carbon going into gas per pound of fuel burnt:

Carbon in fuel 0.6269 pounds
Carbon in ashes $\frac{268}{2240} \times 0.12 = 0.0144$ "
Carbon in gas = 0.6125 "

Volume of gas (at 0° C.) per pound of fuel used:

$$\frac{0.6125}{0.009956} = 61.52 \text{ cubic feet}$$

Per ton of 2240 pounds = **137.805** cubic feet
At 15° C. (60° F.)

$$= 137.805 \times \frac{273 + 15}{273} = 145.375 \text{ " " (2)}$$

(3) The calorific power of the dried fuel is given as 6,786 Calories per pound, water condensed. Since one pound of wet fuel contains 100 — 8.60 = 91.40 per cent dried fuel, the calorific power of 1 pound of wet fuel, moisture condensed, is

$6{,}786 \times 0.9140 = 6{,}202$ pound Calories.

But, 1 pound of wet fuel would produce, on combustion,

$0.0860 + 9\ (0.0457) = 0.4973$ pounds moisture,

which, remaining vaporized at 15°, would retain

$0.4973 \times 596 = 296$ Calories,

leaving the net metallurgical calorific power as

$6{,}202 - 296 = 5{,}906$ Calories.

The 61.52 cubic feet of gas produced per pound of moist fuel will have a calorific power, burnt cold, of

$61.52 \times 76.7 = 4{,}719$ Calories,

making the efficiency of the producer, on fuel consumed in it,

$$\frac{4{,}719}{5{,}906} = 0.799 = \mathbf{79.9} \text{ per cent.} \quad (3)$$

The above figure is true only on the assumption that the steam used is obtained from waste heat, and therefore does not require the combustion of extra fuel.

(4) The gas produced contains, per cubic foot, the following amount of hydrogen, free and as CH^4:

As H^2 0.275 cubic feet $\times$ 0.09 = 0.02475 ounces
As CH^4 0.040 cubic feet $\times$ 0.09 = 0.00360 "

Sum 0.02835 "
= 0.001772 pounds
Per pound moist fuel $= 0.001772 \times 61.52$ = 0.1090 "
Present as ammonia gas $0.0140 \times (3 \div 14)$ = 0.0010 "

Total (not including H as water) = 0.1100 "
Hydrogen in 1 pound of coal = 0.0457 "

Hydrogen from decomposition of steam = 0.0542 "
Water decomposed in the producer = **0.5787** " (4)

Since the steam introduced weighs 2.5 pounds for every pound of fuel burnt, we see that only

$$\frac{0.5787}{2.5} = 0.2315 = \mathbf{23.15} \text{ per cent}$$

of the steam introduced is decomposed.

[In the writer's opinion this unused 76.85 per cent can only pass in and pass out carrying out sensible heat, and it seems a very wasteful method of keeping down the temperature in the producer. From the standpoint of regarding the 2 pounds of steam as a mere absorber of sensible heat, it could probably be replaced by some of the gases of combustion from the gas engine or open-hearth furnace. The products of combustion of the above gas would contain approximately

Nitrogen ..... 68.7 per cent
Carbon dioxide ..... 14.7 "
Water vapor ..... 16.6 "

And if the ½ pound of water vapor decomposed in the producer were thus supplied, it would bring in, per pound of coal burnt,

Nitrogen ..... 41.2 cubic feet
Carbon dioxide ..... 8.8 "
Water vapor ..... 9.9 "

And if the CO^2 thus introduced were reduced to CO, as it probably would be, the gases would receive from this source

Nitrogen ..... 41.2 cubic feet
Carbon monoxide ..... 27.5 "
Hydrogen ..... 9.9 "

Thus producing gas quite up to standard as regards combustibles, while the heat absorbed in the reduction of CO^2 to CO would cool the fire down quite as effectually as the extra steam now used.]

(5) The steam decomposed, 0.5787 pounds per pound of fuel burnt, may be assumed to be at 100° C. on entering the fire, and to therefore absorb 3,196-pound Calories per pound of steam decomposed. The heat absorbed in the producer, and thus transferred into potential calorific power is, therefore,

$3196 \times 0.5787 = 1850$ pound Calories,

which expressed in per cent of the calorific power of the fuel is

$$\frac{1850}{5906} = 0.313 = \mathbf{31.3} \text{ per cent.} \quad (5)$$

(6) The heat generated in the producer equals the calorific power of the coal minus the heat lost by carbon in the ashes, minus the calorific power of the gases, plus the heat absorbed in decomposing steam. The loss by carbon in the ashes is

$$\frac{268}{2240} \times 0.12 \times 8100 = 117 \text{ Calories.}$$

The heat generated in the producer is, therefore,

$5906 - 117 = 4719 + 7850 = 2920$ Calories,

equal to $2{,}920 \div 5{,}906 = 49.4$ per cent of the calorific power of the coal. Of this, 1850 Calories is absorbed in decomposing steam, which is $\frac{1850}{2920} = \mathbf{63.4}$ per cent of the total heat generated. (6)

(7) There are 2920 Calories generated in the producer, of which 1850 are absorbed in decomposing steam, leaving 1070 Calories to supply radiation and conduction and as sensible heat in the hot gases, to which must be added the sensible heat in the hot air and steam used at 300° C.

The air used per pound of coal is found from the nitrogen in the gases:

Nitrogen in 1 cubic foot of producer gas = 0.413 cubic foot
Nitrogen in 61.52 cubic foot of producer gas = 25.40 "
Air used = 25.4 ÷ 0.792 = 32.08 "
Steam used = (2.5 × 16) ÷ 0.81 = 49.40 "

Heat in steam and air at 300° C. (from 15° C.):

$32.08 \times 0.3116 \times 285 =$ 2850 ounce Calories
$49.40 \times 0.3872 \times 285 =$ 5450 " "

Sum = 8300 " "
= 519 pound Calories

Total heat radiated, conducted and in hot gases:

1070 + 519 = 1589 Calories.

The heat in the hot gases is as follows, per cubic foot of gas produced:

CO	$0.110 \times 0.313 \times 335$	= 83.7
H^2	$0.275 \times 0.313 \times 335$	
N^2	$0.413 \times 0.313 \times 335$	
CO^2	$0.165 \times 0.450 \times 335$	= 24.9
CH^4	$0.020 \times 0.400 \times 335$	= 3.1
	Sum	= 111.7 ounce Calories
		= 7.0 pound Calories

Per pound of coal burnt:

$7.0 \times 61.52 = 430$ pound Calories

To this must be added the heat in undecomposed water vapor in the gases, as follows:

Steam used per pound of coal = 2.50 pounds
Steam decomposed per pound of coal = 0.58 "
Steam remaining in gases = 1.92 "
Moisture in coal = 0.086 "
Total in gases = 2.006 "
Volume = (2.006 × 16) ÷ 0.81 = 39.6 cubic feet

Heat contained in this at 350° C.:

$39.6 \times 0.395 \times 335 = 5240$ ounce Calories
= 328 pound Calories

Total heat in producer gases:

430 + 328 = 758 Calories

Heat lost by radiation and conduction:

1589 — 758 = 831 Calories

Proportion of calorific power of coal thus lost:

$$\frac{831}{5906} = 0.141 = 14.1 \text{ per cent} \qquad (7)$$

When Mond gas is heated in the regenerator of an open-hearth furnace it changes considerably in composition, as is shown by the following analyses made by Mr. J. H. Darby, on gas dried before analysis:

	Before Regenerator.	*After Regenerator.*
Carbonic acid gas, CO^2	17.8	10.5
Carbon monoxide, CO	10.5	21.0
Ethylene, C^2H^4	0.7	0.4
Methane, CH^4	2.6	2.0
Hydrogen, H^2	24.8	17.7
Nitrogen, N^2	43.6	47.8
	100.0	100.0

The above changes are very interesting, and their discussion profitable. Before heating, the gas burns with a non luminous flame; after heating, it burns with a brilliant white flame. Let us inquire:

(1) What chemical change occurs during the heating?
(2) What are the relative volumes of the gas before and after heating (excluding water)?
(3) What change in the calorific power is produced by the heating?

(1) An inspection of the analyses shows undoubtedly that at a high temperature the CO cannot hold all its oxygen in the presence of such a large amount of hydrogen, and that the following reaction must occur:

$$CO^2 + H^2 = CO + H^2O$$

The figures given do not check exactly, but assuming that the above reaction takes place, we can find to what extent, by expressing the composition of the gas after heating for the same amount of *nitrogen* as was in the gas before heating, since this gas is unchanged:

	Before Heating.	*After Heating.*	*Loss or Gain.*
CO^2	17.8	9.6	— 8.2
CO	10.5	19.7	+ 9.2
H^2	24.8	16.1	— 8.7
N^2	43.6	43.6	0.0

Since, according to the reaction written, the volume of CO^2 reduced to CO will be equal to the volume of H^2 thus consumed, and will produce an equal volume of CO; the above table proves, within the probable limits of error, that the reaction written actually takes place.

The separation of luminous carbon is probably due to the splitting up of C^2H^4.

The relative volumes of the heated and unheated gases will be inversely as the percentage of nitrogen in each (since this gas is unchanged), viz.: as 47.8 to 43.6, or as 100 to 91.2. The contraction, 8.8 parts, would again correspond almost exactly to the amount of water formed in the assumed reaction, which would be equal to the hydrogen so used, thus giving another check on the validity of the reaction assumed to take place.

The calorific power of 1 cubic foot of original gas is:

CO	$0.105 \times$	3,062 =	321.5 ounce Calories
C^2H^4	$0.007 \times$	14,480 =	101.4 " "
CH^4	$0.026 \times$	8,598 =	223.5 " "
H^2	$0.248 \times$	2,613 =	648.0 " "
		Sum =	1294.4 " "
		=	80.9 pound "
Per 100 cubic feet		=	8090.0 " "

The calorific power of 1 cubic foot of heated gas is:

CO	$0.216 \times$	3,062 =	661.4 ounce Calories
C^2H^4	$0.004 \times$	14,480 =	57.9 " "
CH^4	$0.020 \times$	8,598 =	172.0 " "
H^2	$0.177 \times$	2,613 =	462.5 " "
		Sum =	1353.8 " "
		=	84.0 pound "

For 91.2 cubic feet the calorific power would be $84.0 \times 91.2 = 7715$ pound Calories.

The net conclusion is that the heated gas (aside from its sensible heat) gives less heat by combustion in the furnace per unit of coal gasified in the producers than the corresponding quantity of unheated gas; but the difference is only some 3 per cent. On the other hand, the calorific power of the heated gas per cubic foot is some 5 per cent greater than that of the unheated gas, and, therefore, its calorific intensity will have been increased by the heating—quite aside from the question of its temperature being higher, and therefore its sensible heat greater.

4. WATER GAS.

This gas is made intermittently, by first burning part of the

fuel until the fire is very hot, and then introducing steam, cool or superheated, into the fire. The reaction producing the gas is:

$$\underset{\text{I}}{C} + \underset{\text{I}}{H^2O} = \underset{\text{I}}{CO} + \underset{\text{I}}{H^2}$$

And if the reaction is complete, and only carbon is present as fuel, the gas produced is theoretically composed of equal parts by volume of CO and H^2, the volume of each of these being equal to that of the steam used (if measured at the same temperature and pressure.)

Deviations from this ideal composition are caused in practice by the occurrence of undecomposed steam in the gas, also of CO^2 and N^2, which come from the residual gas formed during the heating up of the fire, some of which will get into the first portion of the water-gas produced, and also hydrocarbons, tar and ammonia from the distillation of the fuel. A typical analysis of water-gas, given by Mr. W. E. Case, is:

Hydrogen (H^2)	48.0
Carbon monoxide (CO)	38.0
Methane (CH^4)	2.0
Carbon dioxide (CO^2)	6.0
Nitrogen (N^2)	5.5
Oxygen (O^2)	0.5

The production of water-gas is more expensive than that of other artificial producer gases, because of the large amount of steam necessarily required, the heat lost during the heating up, and the essentially intermittent character of the operation. Its essential advantages are its very high proportion of combustibles, averaging 90 per cent, and the consequent high calorific intensity which it is capable of producing. (Uncarburetted, it is well known that water-gas is a valuable domestic fuel, and illuminant when used in mantle burners; carburetted, it forms our principal illuminating gas, and as such is manufactured on an immense scale. We will treat here only of its metallurgical uses.)

Discussing the manufacture of the gas, the first step is the heating up of the fuel. This is accomplished by blowing air through it. At this point we can distinguish two systems. The older one is that of blowing in air under moderate pressure, which passes through the fire at moderate velocity, and produces a fair grade of ordinary producer gas. This gas is either wasted, or burned in furnaces requiring such quality of gas, or burned in regenerators where heat is stored up to be utilized in the next stage in superheating steam. The newer system is to blow in air at high pressure, such that a large percentage of carbon dioxide remains in the gases, thus nearly completely burning what carbon is oxidized, and storing up a corresponding quantity of heat in the remaining carbon. The incombustible gases produced are passed through a recuperator or regenerator, where their sensible heat is partly communicated to the steam used, so as to superheat it when forming water gas. This system consumes the minimum of carbon in "heating up" the fuel, saves time in this unproductive period, and allows the producer to be run longer "on steam." In practice the fuel is probably raised to an average temperature of 1500° C. during the heating up.

We can calculate the efficiency of this heating up, that is, the proportion of the calorific power of the carbon burnt which is stored up in the remaining fuel, if we know how much fuel is in the producer, how much air is blown through, the composition of the gases produced, and the average rise in temperature of the fuel. Since this operation is, however, only supplementary to the real formation of water-gas by the decomposition of steam, we will first make calculations upon the cooling down period, during which water-gas is made.

During the use of steam the reaction absorbs heat, and the producer rapidly cools. Steam is passed through until the temperature of the fuel is 800° C., and must then be stopped, because between 800° and 600° the reaction is mostly

$$C + 2H^2O = CO^2 + 2H^2$$

resulting in a rapid increase of CO^2 in the gas. The heat to decompose steam is furnished partly by the oxidation of carbon to CO, and partly by the sensible heat in the carbon itself. Since the carbon, from temperatures of about 1000° C. up has a specific heat of 0.5, we can easily calculate how much steam can be decomposed before the temperature falls to 800°.

Problem 17.

A Dellwick-Fleischer water-gas producer contains 3 tons of coke (90 per cent carbon), heated up to 1500° C. Steam, heated to 300° C., is passed through for 8 minutes, until the temperature of the gas escaping is 700° C., or 800° at the fuel bed. The composition of the gas is (Prof. V. B. Lewes):

Hydrogen	50.0
Carbon monoxide	40.0
Carbon dioxide	5.0
Oxygen	1.0
Nitrogen	4.0

Assume 9000 Calories per minute lost by radiation and conduction.

Required: (1) The amount of steam used in the 8 minutes. (2) The volume of gas produced in the 8 minutes.

(1) The amount of steam which could have been used is limited by the available heat to decompose it. The latter is furnished by

(*a*) Oxidation of carbon to monoxide.
(*b*) Oxidation of carbon to dioxide.
(*c*) Sensible heat of the carbon and ash of fuel.
(*d*) Sensible heat of the steam used.

While the items of heat absorption and loss are:

(*e*) Decomposition of the steam.
(*f*) Sensible heat in the gases.
(*g*) Radiation and conduction.

The simplest way to arrive at a solution is to make a few necessary assumptions, and to then let X represent the weight of steam used. The assumptions are that the average temperature of the fuel bed falls to 1000° C., that the average temperature of the escaping gases is $(1500 + 700) \div 2 = 1100°$ C., that the average specific heat of carbon in the range 1000 — 1500 is 0.5, and of the ash of the fuel 0.25. Then, casting up a heat balance sheet in terms of X, we can finally arrive at an expression for the heat which was available during the 8 minutes for decomposing steam, and thus at the weight of steam decomposed, and (making this equal to X) at a solution.

(*a*) The analysis of the gas shows that eight times as much carbon was burnt to monoxide as to dioxide, making the equation of combustion:

$$9C + 10H^2O = 8CO + CO^2 + 10H^2$$

By weight, $8 \times 12 = 96$ parts of carbon was burnt to CO per $10 \times 18 = 180$ parts of steam used, or 0.533 parts per one part of steam. The heat generated in forming monoxide is, therefore:

$$0.533 \times X \times 2430 = 1296\ X \text{ Calories}$$

(*b*) Only one-eighth as much carbon burns to dioxide, giving, therefore, as the heat evolved:

$$0.533 \times 8) \times X \times 8100 = 540\ X \text{ Calories}$$

(*c*) 3000 kilos. of fuel, representing 2700 kilos. of carbon and 300 kilos. of ash, cool from 1500° to 1000°:

$$2700 \times 0.50 \times 500 = 675{,}000 \text{ Calories}$$
$$300 \times 0.25 \times 500 = 37{,}500 \quad "$$
$$\text{Sum} = 712{,}500 \quad "$$

(*d*) $X \times 0.81$ will be the volume of the steam used at normal conditions, which brings in considered only as vapor at 300°:

$$(X \times 0.81) \times 0.385 \times 300 = 93.5\ X \text{ Calories}$$

The sum total of (a) + (b) + (c) + (d) gives the total available heat, viz.:

1929.5 X + 712,500 Calories

(e) The steam used requires for its decomposition, considered theoretically as cold steam, producing cold products:

(X ÷ 9) × 29,042 = 3227 X Calories

(f) The gas being 50 per cent hydrogen, and the latter being equal to the volume of steam used, the volume of gas must be 2 × (X ÷ 0.81), which multiplied by the mean specific heat of gas of this composition between 0° and 1100° per cubic meter, and by the temperature, will give the heat thus carried out of the producer:

2(X ÷ 0.81) × 0.347 × 1100 = 942.5 X Calories

(g) 9000 × 8 = 72,000 Calories

The sum total of (e) + (f) + (g) gives the total heat distribution, viz.:

4,169.5 X + 72,000 Calories

Since the heat available equals the heat distributed:

1,929.5 X + 712,500 = 4,169.5 X + 72,000

or X = **286** kilograms (1)

(2) The volume of this steam, at assumed standard conditions, would be

286 ÷ 0.81 = 353 cubic meters,

and of gas, since it is 50 per cent hydrogen,

353 × 2 = **706** cubic meters

The above figures represent the maximum attainable production, on the assumption that sufficient steam-generating power is available to furnish the steam, and that the fuel in the producer is of small size and the bed so uniform that the production of gas is regular all over it. In practice, figures considerably below this are attained, but it is always well to know the possible maximum which is attainable.

Problem 18.

In a Dellwik-Fleischer water-gas producer the heating up is accomplished in 2 minutes by blast from a Root blower, furnishing air through a 9.5-inch pipe at a total water-gauge pressure of 19 inches of water, temperature of air 15° C. The gases escaping from the producer analyze:

Carbon dioxide	17.9 per cent
Carbon monoxide	1.8 "
Nitrogen	78.6 "
Oxygen	1.7 "

Temperature of waste gases 900° C.; heat lost by radiation and conduction 9000 Calories per minute; assume producer to contain 3000 kilos. of fuel, consisting of 90 per cent carbon and 10 per cent ash.

Required: (1) The average rise in temperature of the fuel bed.

(2) The proportion of the heat generated which is thus stored up as useful heat for producing water gas.

(3) Assuming that the production of water-gas lasts 8 minutes, during which 2500 Calories are absorbed from the fuel bed per kilogram of steam used, what should be the steam supply in kilograms per minute?

(4) The ratio between the volume of air supplied during the blowing up period and the weight of steam used in the gas-making period.

Solution: (1) We must first find the amount of air furnished by the blower. To do this, we calculate the pressure head in terms of air at 15° C., instead of water, and then apply the well-known formula $V = \sqrt{2g.h}$. Water is 772 times as heavy as air at 0° C., and, therefore, 19 inches of water pressure would represent 19 × 772 ÷ 12 = 1222 feet of air pressure (that is, a column of fluid as light as air, 1222 feet high). But air at 15° is lighter still than air at 0°, in the ratio 273 to 288, so that the air pressure corresponding to 19 inches of water at 15° will be

$$1222 \times \frac{288}{273} = 1290 \text{ feet}$$

The velocity of the air supplied will therefore be $\sqrt{64 \times 1290}$ = 287 feet per second,

and the volume delivered, per minute, at 15° C.:

287 × 60 × 0.7854 × 0.5 × 0.5 × 144 = 17,220 × 0.462 = 8472 cubic feet,

which, in terms of air at 0° C., would be

$$847 \times \frac{273}{288} = 8050 \text{ cubic feet}$$

228 cubic meters.

The volume of waste gases produced in the 2 minutes can be found from the relative percentages of nitrogen in the air (79.2), and in the gases (78.6), as follows:

$$228 \times 2 \times \frac{79.2}{78.6} = 459.5 \text{ cubic meters}$$

Containing, therefore, from its analysis:

Carbon dioxide	82.25 cubic meters
Carbon monoxide	8.27 " "
Oxygen	7.81 " "
Nitrogen	361.15 " "
	459.48 " "

The carbon burnt to CO^2 and CO will be

C to CO^2 82.25 × 0.54 = 44.42 kilos.
C to CO 8.27 × 0.54 = 4.47 "

Sum = 48.89 "

And the heat thus generated:

C to CO^2 = 44.42 × 8100 = 359,800 Calories
C to CO = 4.47 × 2430 = 10,860 "

Sum = 370,660 "

To find the amount of this heat left in the producer at the end of the 2 minutes blowing up, we must subtract the 2 × 9000 = 18,000 Calories lost by radiation and conduction, and then, in addition, the heat carried out by the hot gases, at an average temperature of 900° C., which latter will be:

CO, O^2, N^2 377.25 × 0.327 × 885 = 109,150 Calories
CO^2 82.25 × 0.571 × 885 = 41,565 "

Sum = 150,715 "

Heat left in the fuel bed:

370,660 − 168,715 = 201,945 Calories

Heat capacity of the fuel bed per 1° C.:

3000 × 0.9 kilos. carbon × 0.5 = 1,350 Calories
3000 × 0.1 kilos. ash × 0.25 = 75 "

Sum = 1425 "

Average rise in temperature of the fuel bed:

$$\frac{201,945}{1425} = 142° \text{ C.} \quad (1)$$

(2) The useful heat thus stored up in the fuel bed corresponds to the following proportion of the total heat generated during the blowing up:

$$\frac{201,945}{370,600} = 0.545 = 54.5 \text{ per cent} \quad (2)$$

(3) Steam which can be decomposed in the gas-producing period:

$$\frac{201{,}945}{2500} = 80.8 \text{ kilograms}$$

$$= 10.1 \text{ kilograms per minute} \qquad (3)$$

(4) Air supplied in 2 minutes = 456 cubic meters. Steam used in 8 minutes = 80.8 kilograms.

$$\text{Ratio} = \frac{80.8}{456} = 0.177 \text{ kilos. steam per 1 m}^3 \text{ air} \qquad (4)$$

$$= 0.117 \text{ ounce steam per 1 ft}^3 \text{ air}$$

$$= 1.1 \text{ pounds steam per 100 ft}^3 \text{ air}$$

[The eleven instalments of this treatise on Metallurgical Calculations which have so far been published are now in press, and will appear as Part I. of the treatise in book form (See publisher's announcement on another page.) Subsequent instalments appearing during 1906 will appear as Part II., and those of 1907 as Part III., concluding the treatise. The parts will be paged consecutively, so as to permit of combining into one volume, with index, when completed.]

Electrolytic Copper.[1]

By Lawrence Addicks.

The advent of the dynamo made electrolytic copper at once possible and necessary. The consumption of copper pursuant upon the generation and transmission of electric power in ..... never dreamed of in the days of primary batteries, ... times exceeded the output obtainable from the few ..... the native metal which will yield a product of satis- ..... nductivity without electrolytic treatment.

..... that certain mines in the region of the Great Lakes .... neral so free from objectionable impurities that a high-grade copper is obtainable directly by a simple furnace scorification, early gave Lake copper a premium in the market that electrolytic competition even yet has not entirely wiped out. For almost all purposes there should be no hesitation whatever to-day in specifying electrilytic copper. In conductivity it is superior to nearly all brands of Lake, and in mechanical properties it leaves little to be desired.

There are a few classes of work in which the metal is subjected to very severe punishment, such as in the making of cartridges, where Lake seems to stand up better than electrolytic, although even here it is a question how much to allow for the trade prejudices of the older generation of mill foremen who clung so tenaciously to the perplexing system of wire gauges of but a few years past. Any such advantage in Lake copper must be attributed to the fact that it is not so clean as electrolytic. A small quantity of arsenic, for example, while exceedingly detrimental to the conductivity, will appreciably improve the mechanical properties of copper. At one time arsenic was considered necessary in English specifications for fire box copper. Each year a greater proportion of the world's output is electrolytically refined, and it seems probable that before long there will be but two grades of copper on the market—high conductivity copper, which will include electrolytic and picked brands of Lake, and casting copper, covering all material which will not pass a conductivity requirement of 98 per cent annealed.

The ideal electrolyte for refining copper would be one chemically inert as a solvent but capable of electrochemically dissolving copper, and of zero specific resistance. These conditions are most nearly met by an acidulated sulphate solution, and this has the additional commercial advantage of employing one of the cheapest of chemicals—sulphuric acid. A sufficiently high percentage of sulphuric acid may be carried to make the specific resistance very low. There is a slight redissolving of the cathode copper, but not in serious amount. Gold is untouched and silver practically so.

It is customary to carry a minute percentage of some soluble chloride as a constituent of the electrolyte which would throw down as chloride any silver which might be dissolved and also tend to slime antimony as oxychloride. Some claim that smoother cathodes result from the presence of chlorine. While the actual benefits from this ingredient are open to question, it certainly does no harm, and the refiner is often morally sustained by its presence.

In American refineries high-grade anodes are universally used. Usual compositions are:

Copper	98 to 99.5	per cent	
Silver	0 to 300	ounces	per ton
Gold	0 to 40	ounces	per ton
Arsenic	0 to 2	per cent	

Small amounts of antimony, bismuth, iron, nickel, sulphur, selenium, tellurium and silicon.

Impurities with soluble sulphates go entirely into solution and grow cumulatively. Selenium and tellurium and the precious metals go entirely into the slimes. We have, therefore, a triple separation. The usual products are only copper at the cathode, silver and gold from the slimes, and sometimes copper and nickel sulphates from the electrolyte. Selenium and tellurium, especially the latter, are easily recoverable from the slimes, were there sufficient market. Tellurium is an element almost without use in the arts. Small amounts are used in medicine, but the output of one of our copper refineries for one day would stock a large chemical supply house with tellurium enough to last a year. Arsenic, antimony and bismuth go partly into solution, partly into the slime, depending on the nature of the form of combination in which they exist in the anode and various secondary reactions taking place in the electrolyte.

Arsenic is the most difficult impurity with which to deal. If much is present in the anode, say over 1 per cent, it rapidly grows in the solution and necessitates a large expenditure for purifying to hold it down. With small quantities it seems to slime more rapidly, and no purifying at all is necessary. With other conditions properly cared for, satisfactory cathodes can be produced in an electrolyte running 2 per cent arsenic, though this is considerably above the customary figure.

The purification of solutions is almost always done by working up ..... certain quantity regularly into bluestone and adding ..... acid to the electrolyte. If the purification requirement is heavy the bluestone department becomes disproportionately large, and consequently copper running high in arsenic is most unwelcome. Antimony and bismuth are seldom present in sufficient quantity to cause trouble. Various chemical methods of purification have been tried with the object of regenerating the electrolyte and returning it to the circulation, but there are few substances that will precipitate arsenic in an acid solution, and further the action of such reagents does not seem reliable, probably due to differences in form of arsenic compounds present at different times. For example, the writer has boiled electrolytes running high in arsenic with metastannic acid[2] and obtained a remarkable cleansing action, and upon other trials found hardly any precipitation.

The insoluble anode slime from which the gold and silver are recovered is very variable in composition, as might be expected, depending on the grade of material being handled. It is chiefly metallic in nature, and a common composition would be as follows:

Ag 40 per cent, Au 2, Cu 25, Se and Te 5, As and Sb 10, Pb, silica, sulphuric acid, etc., 18 per cent.

The presence of so much copper in the slime is objectionable

[1] A paper read before the Franklin Institute of Philadelphia.

[2] U. S. Patent of F. B. Badt.

in the silver refinery process, and must be removed before cupelling the dried slimes to avoid making very rich slags. The other impurities give but little trouble, except possibly tellurium, which requires prolonged furnace treatment to burn it off. The copper is present in very finely divided form, as is shown by experiments in fine screening to reduce the copper contents. Almost all of the copper which will pass a 40-mesh wire screen will also pass a 200-mesh. This would indicate that the coarser screen removes the fine crystals that have dropped from the cathodes, and that the remainder is in the form of a chemical cement. Wohlwill's experiments[8] in-

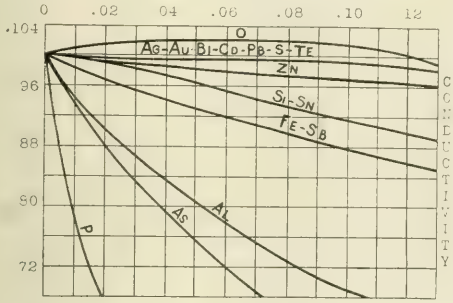


FIG. 1.—INFLUENCE OF IMPURITIES ON ELECTRIC CONDUCTIVITY OF COPPER.

dicate that this is due to the formation of some cuprous sulphate at the anode with subsequent oxidation to cupric sulphate and the liberation of metallic copper:

$$2\dot{C}u = \dot{C}u\ddot{} + Cu$$

The amount of cuprous sulphate formed is a problem in chemical equilibrium dependent upon many factors. The copper dust is usually removed by leaching it out in hot concentrated sulphuric acid, taking care not to carry the action too far, as this would result in dissolving some of the silver. If a more rapid reaction is made use of, such as that with ferric or silver sulphates, the copper is dissolved almost immediately, owing to the enormous surface exposed.

The cathode copper is exceedingly pure, usually running about 99.93 per cent copper, with hydrogen a the chief impurity. Objectionable cathode impurities are of two classes—those which depress the electrical conductivity and those which make the metal brittle. Arsenic and antimony represent the first class; tellurium and lead the second. Good cathode copper should show but a few thousandths of a per cent of arsenic and antimony. The writer's experiments have indicated that it takes but 0.0013 per cent of arsenic or 0.0071 per cent of antimony to lower the conductivity 1 per cent. Any conductivity troubles in electrolytic copper can almost invariably be traced to the presence of undue amounts of one or both of these elements.

Impurities of the brittle-making class are very rarely met with, and if present are due to mechanical contamination of the cathode, either in the bath or in the subsequent furnace treatment.

A third class of cathode impurity concerns the refiner, comprising silver and gold. Cathodes usually show from one-tenth to 1 ounce of silver and a trace of gold. This seems to be due entirely to the mechanical fouling of the cathode by particles of anode slime. In fact, it is a question if most of the arsenic and antimony found in the cathodes does not have a similar origin. The slime is very finely divided, and the continual circulation of the electrolyte necessary to prevent polarization maintains a very slight turbidity. These particles are probably electrostatically attracted to the cathode. Fig. 1 shows in a general way the effect of [illegible] on the electrical conductivity of copper.

The output of a refining tank depends upon the total current passing through it, the number of electrodes in series, the time the current flows and the electrochemical equivalent. In the present case the theoretical amount per ampere day per pair of electrodes in series is almost exactly [illegible] pois ounce. The amount actually deposited is always [illegible] than this, due to chemical solution of the cathode, grounds and short circuits between electrodes. The redissolving of the cathodes will amount in practice to 0.5 to 1.0 per cent of the amount deposited. This action is chiefly at the surface of the bath. In the presence of oxygen copper is oxidized to cupric sulphate by sulphuric acid.[9] The reaction is quickened by heat, but in so far as the writer's experiments extend, is independent of the composition of the electrolyte. Metallic copper is also slightly soluble in pure water.

If the tanks are well insulated from the supporting piers, and care is taken to break the circulating pipes carrying the electrolyte from tank to tank, by rubber tubing or miniature waterfalls, the current shunted around the tanks by grounds should not average over 1 per cent of the total. Short circuits between electrodes, caused either by direct contact between anode and cathode or by indirect contact between the electrodes and the tank, will amount to some 5 per cent under best commercial conditions. In this way the net current efficiency is usually from 90 to 95 per cent.

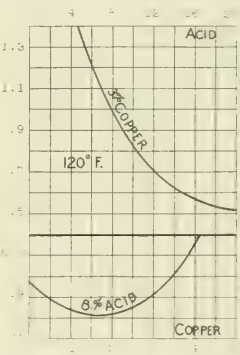


FIG. 2.—SPECIFIC RESISTANCE OF ELECTROLYTES OF VARYING COMPOSITION.

The voltage at the switchboard required to force the current through the tank-house depends upon a number of factors. The resistance is made up of metallic resistances, liquid resistances, contacts and counter e. m. f.

The metallic resistances can be figured on Thomson's law that the cost of power lost should equal the interest on the investment in copper. As the continuous operation gives a load factor of 100 per cent, the rule can be applied without the usual allowances. As regards the investment, the copper can be considered as so much extra metal tied up in the process. Any density under 1000 amps. per square inch will run cool enough.

The liquid resistance of the electrolyte is largely influenced by its chemical composition. Fig. 2 shows the approximate specific resistances of electrolytes carrying varying amounts of copper and free acid. The conductivity of an electrolyte depends upon the number of ions present, the speed of migration and the charge of electricity each carries. The charges carried is a constant for the various compositions as the valency does not change. The ions concerned are copper, hydrogen and sulphion. Hydrogen is much the swiftest of these, and hence the effect of the free acid present in reducing the resistance.

The number of ions present depends upon the concentration and the degree of dissociation. [illegible]

[8] *Zeitschrift fur Elektrochemie*, April 23, 1903, p. 311, *et seq.*

[9] See also ELECTROCHEMICAL AND METALLURGICAL INDUSTRY, Vol. III., 1905, p. 314.

[illegible] Metallurgie, Aug. [illegible]

[6] *Zeitschrift fur Elektrochemie*, Aug. [illegible]

ture of two electrolytes with a common anion, SO_4, and the proportion of one present controls the degree of dissociation of the other. This is shown in the lower curve of Fig. 2, giving the effect of varying the copper contents of the solution. Starting with a fixed percentage of free acid present, additions of copper sulphate at first increase the conductivity, due to the increased number of ions present. After a certain quantity of copper is reached, however, further additions have the reverse effect. This is due to the driving back of the dissociation of the acid by the increased proportion of the SO_4 ions claiming a copper mate in accordance with the laws of equilibrium.

It will be seen from these data that the copper should not exceed 3 per cent at the most, or 12 per cent if figured as bluestone. The acid may advantageously be run up to about

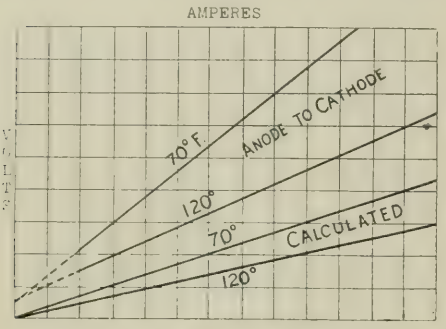


FIG. 3.—VOLT-AMPERE CURVES.

13 per cent. If carried higher, polarization troubles are likely to offset the gain in conductivity. These figures are for pure electrolytes. In practice impurities cause the resistance to be 10 to 15 per cent higher than shown.

Transfer resistance is the name which has been given to a liquid resistance, the nature of which is not fully understood. If we make a series of measurements of the voltage drop between a pair of electrodes at varying current densities, Ohm's law requires that a current-voltage plot should be a straight line. Where this line cuts the ordinate of zero current is a measure of the counter e. m. f. present.

If we analyze the results, however, we shall find that the indicated specific resistance of the electrolyte is higher than it should be, and that the discrepancy is greater the nearer were the electrodes spaced during the measurement. These facts are illustrated in Fig. 3, and indicate a high ohmic resistance near the electrodes. Exploration of the potential gradient between anode and cathode with a potentiometer confirms this, and shows that it exists at both electrodes.[1]

The most probable explanation seems to be that there are minute bubbles of gas in the neighborhood of the electrodes, due to slight generation of hydrogen and oxygen. The phenomena can also be explained by assuming that there is an irreversible polarization present, which is a function of current.[2] The transfer resistance is greatest in the high acid electrolytes and at low temperatures. Its practical effect is to reduce the gain to be expected by spacing electrodes close together and from increasing the acidity of the electrolyte, and to increase the gain to be expected from heating the electrolyte.

The temperature coefficient of electrolytes is large and varies with the temperature. Fig. 4 shows the effect of temperature change on a typical copper refining electrolyte. The practical effect of the change in temperature upon the resistance is rather complex. The transfer resistance exhibits a very large temperature coefficient, while the contacts and metallic conductors are not appreciably affected. The resultant coefficient figured from the switchboard is approximately 0.5 per cent per degree F.

Contact resistances are met with at the joints in the main bars and at the connections between bars and electrodes. The joints in the main bars should be equal in conductivity to the bar itself. This standard can easily be attained if the bars are properly faced. Three or 400 amps. per square inch of bearing area will give no trouble. The contacts between the electrodes and the main bars are very variable. A single contact will run from 0.000005 to 0.0005 ohm, according to the cleanness of the engaging surfaces and the pressure.

Finally, we have a certain amount of counter e. m. f. present, due to the greater concentration of the electrolyte at the anode than at the cathode. This is in general very small—about 0.02 volt per tank in the multiple system. The anode slimes cause a certain amount of polarization, and if very bulky and adherent it may become serious, starting intermittent gasing and making the tank "crazy."

Tank resistance is therefore made up of a number of factors, and, as none of the power applied is absorbed chemically, disregarding the small counter e. m. f., it is possible to make any of these factors as small as desired, and it is a commercial problem to determine the most profitable value for each. A rough summary of the relative values of each in practice is as follows:

	Per Cent.
Metallic resistance	15
Electrolyte, including transfer	60
Contacts	20
Counter e. m. f.	5
	100

The question of what current density to carry is largely one of power cost. Present American practice runs all the way from 12 to 35 amps. per square foot. The steady full load on a large power house gives ideal conditions for the generation of cheap steam power. With high-current density the tank room requires much closer supervision, as there is a greatly increased tendency toward polarization.

It is customary to circulate the electrolyte from tank to tank to maintain uniform composition throughout. If this were not done the heavy liquor from the surface of the anode would form a horizontal layer of increased specific gravity at the bottom of the tank. Different strata would then have different conductivities, crystals of blue vitriol would form on the bottom edge of the electrodes, and the operation of the tank be thoroughly disorganized. The higher the current density the greater this tendency and the more active must be the circulation.

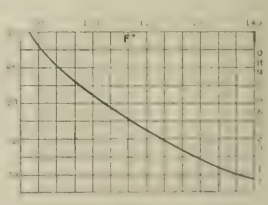


FIG. 4.—EFFECT OF TEMPERATURE.

An excessive circulation tends to stir up the silver mud and results in an increased silver loss in the cathodes. It is safe to say that roughly, the silver contents of the cathodes made at a density of 30 amps. per square foot will be double that of cathodes made at half the density. High-current density also means rougher deposition and necessitates more frequent renewal of the cathodes to maintain the current efficiency. Most of the Eastern refineries run at from 17 to 20 amps. per square foot.

The size of tank is chiefly a matter of construction. The number of electrodes placed in parallel does not seem to bear

[1] Transactions American Electrochemical Society, Vol. VII., p. 61.
[2] Transactions American Electrochemical Society, Vol. VII., p. 33.

any direct relation to the current efficiency. This is probably because low efficiency is caused rather by the condition of the deposit or its tendency to sprout and form trees than by local mechanical troubles between single pairs of electrodes. Sixty electrodes in a single multiple tank give no trouble.

Various dopes have been tried for controlling the character of a deposit. Ammonium sulphate undoubtedly exerts a beneficial effect in encouraging smooth deposits. This may be due to some secondary cathode reaction in which the ammonium radicle is momentarily freed. Ammonium sulphate increases the resistance of the solution, however, by driving back the dissociation of the free sulphuric acid present.

Organic reducing agents, notably gelatin, have marked effect in restraining the crystalline nature of deposits, but are not easily controlled in copper solutions. The peculiar structures sometimes seen at the cathode are doubtless due to the effect of various soluble impurities in the anode. Often a total change in the character of a deposit can be induced by simply lowering or raising the temperature of the electrolyte.

Comparatively little copper is put upon the market in cathode form, although this is an ideal shape from which to make brass and copper castings. The wire bars and ingots never give a conductivity equal to that shown by samples from the cathodes from which they are made. This is partly due to the absorption of some impurities from the furnace lining and products of combustion, and partly to mechanical pocketing of impurities in the cathodes during the electrolytic process. These particles of impurities are associated with nodules, and a smooth part of the cathode is naturally selected as a sample to draw out into wire.

The furnace operation consists in melting down the charge rapidly in large reverberatory furnaces and exposing it to an oxidizing treatment until the bath contains about 6 per cent of cuprous oxide in solution.[9] This exerts a strong scorifying action on such impurities as can be slagged. The metal at this stage exhibits a characteristic fracture, and is called "set" or "dry" copper. Carrying the oxidation further only results in needlessly slagging copper.

The reaction is then changed to a reducing one by means of hydrocarbon gases evolved from green poles thrust beneath the surface of the bath and the copper brought back to "pitch," or when it contains some 0.6 per cent of cuprous oxide, and gives castings the surface of which neither rises nor falls when "setting" in an open mould. The metal in this state develops the best mechanical properties under subsequent treatment. Underpoled copper simply contains an excess of cuprous oxide as an impurity and suffers in both electrical and mechanical properties. Overpoled copper looks like slightly underpoled copper under the microscope. The mechanical properties are satisfactory, but there is a slight decrease in conductivity and castings are bad, due to the expulsion of gases as the metal sets. An overpoled bath is, however, in a very unstable condition, and it is usual in such a case to start at the beginning of the refining operation and oxidize the metal to set copper again. The chemical reasons for these facts are not fully understood. The fact that molten copper is so sensitive chemically to its surroundings, and that it chills so readily, due to its high thermal conductivity, makes it a difficult metal to handle.

This paper has been written from the point of view of the multiple system of refining. The main points of difference between this system and the series system are in power cost, compactness and cost of preparing anodes. The power required is practically half as much again in the multiple tank. The series tank has relatively no contacts or conducting bars, and the electrodes are very close together, the anodes being thin, even plates. To produce such anodes they must be either rolled or specially hand-cast, and the grade of material used must be good. The interest on the metal tied up in process

[illegible] supervision to keep the quality of the cathodes up. [illegible] lined tanks cannot be [illegible] high voltages [illegible] item. The fact that large refineries on both systems are being [illegible] the pros and cons in each case, although much more material is refined by the multiple than by the series process.

[9] Transactions American Institute Mining Engineers Vol. XXXIV., p. 671.

R. Threlfall on Electrochemical Theory.

In his recent presidential address to the [illegible] tion of the (Brit.) Institute of Electrical Engineers, R. Threlfall made the following remarks:

"As the electrochemical industry rests ultimately upon electrochemical theory, it should be my duty to refer to any advance in our knowledge of the general mechanism of electrolysis. I say 'should be,' because so far as I can judge no very definite advance has been made for several years. It is quite possible, however, that such progress has really been made, but has been buried among those portentious papers which, unfortunately, are so common in electrochemical literature.

"The most pressing want at the moment among laboratory electrochemists is a sense of humor in those who contribute accounts of their work to the various journals. Many of these papers appear to have been written with the idea that people interested in the subject enjoy reading for its own sake, and actually prefer fifty pages to five. I do not know how it may be with those who are exclusively employed in developing the laboratory side of the subject, but I am sure that I shall have many sympathizers with my opinion that if electrochemists would study the art of brevity more than they do, a large number of other people would have a better knowledge of the merits of their discoveries.

"It is a thousand pities that the subject of electrochemistry owes so little to French chemists—had it been otherwise we should no doubt have had a standard of exposition, which would have reduced the volume of the literature in the ratio of, say, 10 to 1. Seriously, the exuberant prolixity of physical chemists is a grave and serious drawback to the study of the subject, for it makes it impossible for anyone to keep abreast of the work unless he rigidly confines himself to a small branch of the science.

"To pass on. I am afraid we must admit that the high hopes exhibited by the preliminary successes of the dissociation theory, say, ten years ago, have ended in disappointment. Those of us who can remember the dreary array of uncoordinated and superficial experiments which formed the subject of electrochemistry in the latter part of the last century will easily understand how eagerly a theory was welcomed which promised to reduce all to order and regularity. It did not appear too much to hope that a theory specifically limited to dilute solutions, in which domain it certainly was epoch-making, would soon receive such additions as would fit it for use with everyday solutions.

"No one who has tried to formulate some satisfactory account of electrolytic phenomena in the light of the Arrhenius-Ostwald, Nernst and van't Hoff [illegible] that of gratitude to those pioneers who succeeded in having straightened out a plain path of calculation through the maze of observations.

"What it was not so to be. Further investigation, particularly with solvents other than water and with [illegible] solutions, has shown that the theory which regards the solvent as having nothing in the way of properties except three dimensions in space [illegible] The simple ionic hypothesis [illegible]

memory of what is doubtless one aspect of the phenomena of electrolytic conduction—predominant in solutions of small concentration.

"I wish I could go so far as to say that the hypothesis in its extreme form is only a memory. Unfortunately, this is not the case, for what was once a good servant has grown in its old age into a bad master. I wonder how many young minds engaged in electrochemical research are being stunted and paralyzed by the fashionable necessity of finding an ionic explanation of the phenomena they encounter?

"Who is not weary of the constant recurrence of chemically abhorrent reactions supposed to occur between ions often invented *ad hoc*, and prompted with all solemnity into equations of equilibrium, and that whatever the strength of the solutions? I have seen cases where ions are apparently manufactured from chemical formulas on the *totidem literis* principle as expounded by Swift in the "Tale of a Tub." For instance, Cl might be split on this system into the ions C and l.

"Not only is considerable ingenuity misapplied in the determination to have an explanation of electrochemcial phenomena limited by the dissociation theory at any cost, but the real meaning of the experiments is apt to be masked in the process, and the progress of inquiry arrested by the stifling of curiosity incident on a happy conjunction of gratuitous ions. This is much to be regretted, for I suppose everybody who has worked at these subjects for any number of years must have come to the conclusion that electrochemical phenomena are mainly chemical, and that their elucidation depends rather on the good old plan of close observation and accurate analysis than on dexterity with a pen and paper.

"The crying experimental need of the moment is to find a method of separating and accumulating the layer of electrolyte which is within molecular 'striking distance' of the electrode. Perhaps some kind of minutely porous electrode through which the electrolyte can be drawn at a rate proportional to the current density, or some fortunate discovery in the domain of membranes used as electrodes, may supply us with the all-important instrument of investigation.

"Mr. W. D. Bancroft, in a paper before the American Electrochemical Society (Electrochemical and Metallurgical Industry, 1905, Vol. III., p. 370), has given many illustrations of the application of pure chemistry to electrochemical phenomena, and has, in fact, anticipated much of what I had intended to say on this occasion. In view of this paper, I will only say here that the complexity of electrochemical phenomena is such that a good chemist may be profitably employed for years in elucidating the all-essential details of the chemical reactions occurring at the electrodes between intermediate products of a single electrolysis, and till this is adequately performed no theorizing is of any lasting importance.

"It is no use to be satisfied with the remark that such and such a reaction is an 'oxidation' or a 'reduction' process, or that this or that effect is due to 'over-voltage"; what is wanted is the purely chemical detail free from the ambiguity of cant phrases.

"An investigator starting on such researches as are indicated above will do well to make sure that 'indifferent' electrodes are really indifferent and not catalytic agents. The discovery by Luther and Brislee (*Zeitschrift für Physikalische Chemie*, Vol. XLV., 216, 1903) of the activity of platinum in certain cases is a matter to be taken into serious consideration, and goes far to suggest an explanation of the disappointing results obtained by many observers who have attempted to apply Nernst's method of use of current and voltage to everyday electrolysis.

"Although the subject of the conduction of electricity through gases has hardly begun to claim the professional attention of the engineer, it is of so much importance that it cannot be passed over. At the present moment it is no exaggeration to say that we know more known of the mechanism of gaseous than of liquid conduction, and it does not require much penetration to foresee that an adequate knowledge of conduction in liquids is at least as likely to come from an extension of the principles obtaining in gaseous conduction as by loading the original ionic theory with additional hypotheses.

"In view of the continuity of the gaseous and liquid states it is hard to believe that some of the properties on which gaseous conduction depends do not pass over to, and persist in, the liquid state. I know of no experiments on gaseous conduction extended over the critical point and continued into the liquid state. With a sufficient range of data covering a sufficient number of substances it is hard to believe that some insight into the mechanics of conduction would not be obtained in this way.

"A corresponding research covering the conduction of solutions with less and less solvent and higher temperatures, till ultimately the pure fused salt is alone in question, would possibly—but less probably—throw light on the whole subject. It would at least be interesting to trace out how the migration velocities become equalized as we approach the state of fusion conductivity.

"If we consider the case of molten sodium chloride, for instance, I am unable to understand how the rate of migration of the anions and cations could possibly be different, as there is here no opportunity for change of concentration. Unless the mechanism of conduction is entirely different in fused salts from what it is in solutions, it might be argued that differences in migration velocities depend upon the solvent more than upon the anions and cations themselves—a view perfectly consistent with the known fact that in a particular solvent water each anion and cation appears to have a relative velocity independent of the other and more or less constant.

"Some work on migration velocities of a few selected anions and cations extended to a large number of solvents could hardly fail to be illuminating. The material may be actually in existence, but if so, I have failed to come across it."

Other portions of Mr. Threlfall's suggestive address will be dealt with in our next issue.

Electrolytic Bleach and Caustic.

By Geo. B. M. Zerr and Jay M. Whitham.

No electrolytic plant can live unless its operation is successful from a commercial standpoint; by "successful" we mean that after charging power, supplies, labor, supervision, taxes, interest, repairs and depreciation to the electrolytic plant, its product must readily sell so as to yield at least a 15 per cent return on the installation costs.

By the electrolysis of brine solutions bleaching powder and caustic soda are produced. These products are in great demand for bleaching pulps, rags, cotton goods, etc., and for digesting woods, manufacturing soaps, refining mineral oils, softening waters, etc.

Bleaching powder, having 35 per cent available chlorine, sells at about $25 per ton, f. o. b. cars at the factory, as compared with $45 in 1902. This decrease in cost is due to the use of electrolytic processes of production. Caustic soda is now worth about $35 per ton, while in 1900 it cost $50. This decrease is due to the use of electrolysis in its manufacture. The caustic produced by electrolysis cannot only be produced at less cost, but is purer than that produced chemically.

The following notes refer to diaphragm cells for the production of bleach and caustic. A diaphragm cell consists of an inner and an outer compartment separated by a porous diaphragm. In one compartment, generally the inner, is an anode, usually composed of graphitized carbon, and placed in a strong brine solution (sodium chloride). In the outer compartment, against the diaphragm, is a cathode, usually of perforated iron

or steel plate. At the start, the outer compartment contains water or weak brine. The chlorine set free at the anodes is led through vitrified pipes to a tower or towers, where it is absorbed by lime water, producing bleach liquor, or it may be conducted to chlorine chambers, usually made of lead, where it is absorbed by slacked lime, producing bleaching powder. At the cathode the discharged sodium ions produce caustic soda in solution. The most successful diaphragms are now made of asbestos cloth, asbestos board or asbestos paper, or of two of these combined.

These cells must be insulated. Direct current, at about 4 volts per cell, is used, and the current varies usually from 300 to 400 amps. for cells of usual size. A bleach and caustic plant is composed of several cells arranged in series for reducing the voltage of the generator to about 4 volts per cell. Thus, a dynamo operating under 120 volts would supply current for thirty cells in series, provided it delivered from 300 to 400 amps.

Several of these series are arranged in parallel to consume the amperage of the generator. Thus, a 250-kw, 120-volt, 2,000-amp. generator will furnish current for six series, each of thirty cells, operated at 4 volts per cell and at 330 amps. per cell.

The following notes give the results obtained with a 3.3-ton bleach per day plant after three months operation:

The bleach solution made in 24 hours referred to.

35 per cent bleaching powder, was	7,619 pounds

The electric current used in the 96 cells was:

Volts	205
Amperes	706
The switchboard horse-power made was	220
The engine horse-power made was	260
Salt used per day	2.34 tons
Lime used per day	0.83 "

The actual installation costs of this plant were:

Engine	$5,000
Generator	5,175
Condenser pump and condenser	1,600
Foundations	500
Building for engine	900
Piping	1,050
Boilers	4,000
Stokers for boilers	1,000
Boiler house, part chargeable to electrolytic plant	700
Feed pump, part chargeable to electrolytic plant	100
Breeching, part chargeable to electrolytic plant	200
Chimney, part chargeable to electrolytic plant	1,500
Bleach and brine tanks, etc.	1,450
Storage tanks	600
Piping	350
Brine engine	500
Cell room building	4,000
Lime tank	300
Salt house	700
Evaporator (caustic) house	150
Evaporator and pumps	2,500
Wiring and lights	420
Brine pumps	225
Caustic tanks	500
Ninety-six cells	14,400
Cables, electric	500
Switchboard	600
Chlorination towers	350
Concrete flow and tank	530
Total (about)	$50,000

That is, the installation cost of a 3.3 ton bleaching plant was about $15,151 per ton.

The operating costs per year are:

Fixed charges 10 per cent on costs	$5,000
223 boiler horse-power at steam at $40	8,920
Wages	1,350
Repairs	2,000
Lights	200
Salt	4,611
Lime	1,141
Total	$25,121
Yearly 35 per cent bleach produced 2,425 tons at $28	67,900
344 tons caustic at $37	12,727
Value of product	$40,616
Yearly saving	15,495

Return on the investment for a ... per cent bleach and caustic as produced 31 per cent.

For a 7½-ton bleach plant to bleach 50 tons of sulphate pulp per day we get the following data. The installation costs are:

250 cells	$37,500
Evaporating apparatus	3,250
Tanks	3,000
Towers and chlorine mains	700
Small pumps	700
Piping at cells and tanks	1,000
Engine and dynamo	20,000
Wiring and switchboard	1,000
Boilers	7,000
Steam and water pipes	1,850
Chimney and breeching	4,000
Buildings	10,000
Total	$100,000
Installation cost per ton	13,333

The annual operating costs of this 7.5-ton (bleach) plant are:

Steam, 400-boiler horse-power at $40	$16,000
Wages and superintendent	6,000
Fixed charges, 10 per cent on $100,000	10,000
Repairs	4,000
Fuel for evaporating	3,300
Salt, at $6.75 ton	14,749
Lime, at $5.25 ton	4,095
Oils, etc.	1,000
Total	$59,312

The yearly profit is:

2,425 tons bleach at $28	$67,900
974 tons caustic at $30	29,220
Yearly income	$97,120
Yearly profit	37,808

The yearly profit is 38 per cent on the investment.

These estimates are based on the actual operations of the bleach plant just described.

Notes on Electrochemistry and Metallurgy in Great Britain.

From Our Special Correspondent.

THE FARADAY SOCIETY MEETING.

The Faraday Society opened its winter session on Oct. 30 with a four paper evening. Lord Kelvin presided. There was an exceptionally good attendance of members and students of the Institution of Electrical Engineers being present

together with practically the whole of the Council of the Society. The proceedings began with Prof. Ernest Wilson's paper on "Alternate-Current Electrolysis," which had been previously published in the August issue of the Transactions of the Faraday Society. Prof. Wilson gave a somewhat lengthy lecture on his paper, in connection with which the only defect seemed to be that the electrolytes were not in all cases described with sufficient accuracy, the terms "dilute" and "strong" being somewhat vague.

Lord Kelvin then asked Mr. W. R. Cooper for his paper on "Alternate-Current Electrolysis as Shown by Oscillograph Records," and announced that this and the first paper should be simultaneously discussed. The meeting then followed with keen interest a series of oscillograph records, which showed the effect of polarization in retarding the current wave of an alternating current at low voltage, and some records of an electrolytic rectifier were also shown.

The discussion was opened by Mr. A. P. Trotter, of the Board of Trade, which is *inter alia* the legally constituted protector of the interests of the gas and water companies in regard to corrosion arising from earth currents. The speaker naturally diverted the discussion from the laboratory to outside practical experience, by speaking of the risks of the electrolysis of gas and water pipes by alternating currents. He has found that lead subject to corrosion became encrusted with an insoluble salt, perhaps a carbonate. He thought that one phase in each cycle caused the corrosion, and that the deposit being insoluble, the succeeding phase of the cycle was unable to reverse the condition caused by the first. He had buried two lead pipes, each having an area of 40 square inches, in ordinary moist soil, 4 inches apart, the current passing through being kept at 1 amp. The deposit did not cover the whole of each pipe, but ceased on about one-third of the circumference on the part remote from the adjacent pipe. He had also tried the effect of smaller current densities, and had coated four lead plates with varnish, leaving on each an unvarnished area, the plates being buried in the earth and arranged in series across 100-volt mains, and an alternating current was supplied, the respective current densities in amperes per square inch being $\frac{1}{40}$ $\frac{1}{100}$ $\frac{1}{640}$ $\frac{1}{2560}$ Judged visually the amount of corrosion was proportional to the current density, but no weights had been taken to confirm this, as a scraping off of the carbonate film would have entailed the removal of part of the metallic lead. In view of the fact that the current density was as low as 0.00006 amp. per square centimeter, he was of the opinion that so far as electrolytic corrosion, due to alternating currents, was concerned, there was no such thing as a minimum current density. With regard to lead pipes and lead sheathing generally, if electrolysis did occur through stray currents, the insoluble salt forming on both poles would seal up the fault.

Passing then over to a discussion of direct-current electrolysis, Mr. Trotter regarded all accounts previously published concerning this as unsatisfactory. Perhaps the National Physical Laboratory might investigate the matter. So far as tramways in the United Kingdom were concerned, no electrolysis had been reported during the past twelve years where the regulations laid down by the Board of Trade had been complied with. They must also remember that the character of the soil itself was of supreme importance in regard to the life of both lead and iron, for extensive corrosion often occurred when no tramways existed.

Mr. Alexander Siemens stated he had found, twenty years ago, that alternating current did dissolve lead pipes placed in the ground, but that no measurements of the action occurring had been made. Mr. James Swinburne, referring to Prof. Wilson's estimate of the energy lost per period, pointed out that the loss of energy could not be purely chemical as these processes were reversible; the loss must, therefore, be due to diffusion, which would naturally mean a loss decreasing with the frequency. Mr. Cooper's curves tended to confirm this view. As to Mr. Trotter's remarks concerning insoluble lead salts, in the early days of accumulator design, Lord Kelvin and others had pointed out some curious phenomena connected with the reduction of lead sulphate. Pure lead sulphate was not reduced, but lead sulphate in connection with lead was reducible so long as crystallization did not occur. He deprecated the suggestion that the National Physical Laboratory should study the electrolysis of lead as singularly unfortunate. If there was anything to which they objected at Teddington it was earth currents in any shape or form. Dr. F. Mollwo Perkin mentioned that Le Blanc confirmed Prof. Wilson's views, that with high frequency practically no corrosion occurred. In some experiments of his own, Dr. Perkin stated that with platinum plates in sulphuric, he had, with alternating currents, obtained what was apparently a colloidal solution of platinum. In alkaline solutions he had found a similar result. In reply to the discussion, both Prof. Wilson and Mr. Cooper waived their rights of verbal reply, deferring this to a communication to the Transactions.

Prof. A. K. Huntingdon then read an abstract of his "Note on the Crystalline Structure of Electrodeposited Copper." The only remark made by way of discussion was a contribution by Mr. J. G. A. Rhodin, who expressed his regret that Prof. Huntingdon had not dealt with the current density employed in the production of the samples examined. He (Mr. Rhodin) suggested that the peculiar crystalline formation to which attention had been drawn, had its parallel in magnetism. If a V-shaped groove was cut in one end of a bar magnet, the adhering filings arranged themselves in such a manner as to show a distinct line of repulsion. Were the lines of crystalline stress shown in the microphotographs due in the first instance to an electrostatic action?

Lord Kelvin then announced that, on the author's suggestion, the fourth paper, by Mr. W. Pollard Digby, on "Some Observations Respecting the Relation of Stability to Electrochemical Efficiency in Hypochlorite Production," in view of the lateness of the hour, would be taken as read. Dr. S. Rideal then stated that his interest in the paper was chiefly chemical. The commercial problem of the electrolytic hypochlorites was that of producing a stable solution with a large conversion of chlorine. Mr. Digby had carried an important subject a decided stage further, and he hoped that the investigations would be continued by the author, and that further communications would be submitted to the Society.

The meeting then closed with a vote of thanks to Lord Kelvin, moved by Mr. Siemens, which was carried with hearty acclamation.

The Financial Side of the Electrolytic Alkali Trade.

Thanks, partly to improved trade conditions and partly also to economies in the process of manufacture, the directors of the Castner Kellner Alkali Co. have been able to declare an interim dividend for the six months ending Sept. 30 last, at the rate of 4 per cent per annum. As a result, the market price of these shares, which in January last were £13.10 to £15.10, are now quoted at £1 to £1.1s ex dividend. The condition of their chief rivals, the Electrolytic Alkali Co., which works the Hargreaves-Bird process, is scarcely so happy despite the trade improvement. Preference dividends have been in arrears, but this year a payment is to be made. The report for the year ending Aug. 31 states that the net profit, after allowing for depreciation, mortgage, debenture interest and expenditure on maintenance of buildings, plant, etc., is £4,746, to which is added £1,374 from last year, making £6,120. The directors propose payment on the cumulative preference shares of one year's dividend, on account of arrears of 7 per cent on all the preference shares issued prior to April, 1902, and at the same rate upon all the preference

shares issued since that date. This will leave £519 to be carried forward. The process continues to work satisfactorily, and there is a good demand for the company's products. The great depression which existed in the market price of bleaching powder during 1902, 1903 and 1904 has continued, though to a less extent during the past year. Towards the close of the year the erection of additional manufacturing plants was continued.

At the recent general meeting, Mr. Hargreaves (one of the directors) opposed the adoption of the report, which criticised the policy of the board, and maintained certain salt plants which the company possessed were not adapted to their requirements. He opposed the nomination of Mr. G. H. Harrison as a new director, but, in spite of the opposition, a five to one majority elected Mr. Harrison. Mr. Hargreaves then announced his own resignation from the board. It seems a pity now that trade prospects are improving that the board could not work amicably together.

THE INSTITUTION OF MINING AND METALLURGY.

The first (or October) meeting of the 1905-06 session was devoted to the first of four papers on the agenda. This was Mr. F. Dietzsch's paper on the "Treatment of Tin-Tungsten-Copper Ores at the Clitters United Mines." The paper in itself was a lengthy one, and being contentious and sweeping in its assertiveness, elicited a vigorous discussion. The system which has become used in this reopening of an abandoned mine for the sake of the tungsten present, is one depending essentially in classification in graduated crushing and electromagnetic separation. The details of working are too lengthy for abstracting in this letter. The paper closed with an attack on Cornish mining methods; an attack keenly resented in several of the speeches and communications received.

The remaining three papers were respectively entitled: "The Computation of Assay Values," by William Crosley; "A Graphic Calculator," by William Crosley, and "Ore Valuation of a Witwatersrand Mine," by E. J. Way, and were held over for discussion at the succeeding meeting.

HYPOCHLORITE PROCESSES.

The registration of the British Hosiery & Electrolyte Bleaching Co., Ltd., marks the advent of another electrolytic hypochlorite process on the market, to compete with chloride of lime prepared by the older chemical or newer electrolytic methods. Apparently the new company proposes to reap its dividends from two spheres, the first being that of certain of the Vogelsang patents, the second being some patents for improved knitting and splicing machines. Why the two should be yoked to the same plow is not clear. Separate organizations would surely be desirable. Existing processes in use in England are three in number. The Hermite installations at Netley and Ipswich are, I understand, still in operation for sewage treatment. The Atkins process (owned by Oxychloride, Ltd.), to the trials of which at Guildford I have alluded, is under consideration for sewage treatment by certain municipalities. The Electrozone process, which now uses the Crawford instead of the Woolf patents, has a factory in the north of England, used for turning out stable solutions for domestic and hospital use. The Poplar Borough Council is considering the installation of a plant for municipal disinfecting work. Quite apart from this, experiments are being, and have been, carried out in regard to the bleaching of paper pulps and textile fabrics. In respect to these, the electrochemical efficiencies are generally satisfactory, but the salt consumption remains very high. To compete effectively with existing conditions in paper mills with chloride of lime, a recovery and reworking of the electrolyte seems advisable. This will mean a certain amount of complication, and mill owners distrust complications which they do not understand.

THE SEVENTH REPORT OF THE ALLOYS RESEARCH COMMITTEE.

The subject of the seventh report of this important body was the properties of a series of iron-nickel [illegible] particular property of any one alloy can be referred to with as little loss of time as possible. The actual preparation was carried out at the Hecla Works, Sheffield—it goes without saying that Mr. Hadfield is a member of the committee—and the base of the material used was Swedish charcoal iron of special purity. The percentages of nickel varied from nil to 10.01 per cent, the carbon [illegible] between 0.41 and 0.52 per cent. The manganese [illegible] stancy in any alloy it is notoriously difficult to maintain) varied between 0.82 and 1.03 per cent. The bars generally showed that an increase in the nickel contents raised the maximum stress but limited the elongation. But to attempt to deal adequately with the report would extend over several letters. The committee have presented a report which epitomizes the properties of these alloys, a report of which every decent scientific library will keep copies, and which every metallurgist worthy of the name will place upon his own shelves.

MARKET QUOTATIONS DURING NOVEMBER.

Prices have weakened in only one direction during the month, ammonia sulphate having receded from £13 to £12.12.6 per ton; but in other cases rises are reported. These include arsenic from £14 17s. to £15 per ton, copper sulphate from £22.15 to £24.10 per ton. Exceptional firmness characterizes the market for coal-tar products, carbolic acid crystals having risen ½ penny per pound to 6d. and 6¼d. for 34-35 per cent and 39-40 per cent respectively. Crude carbolic acid has risen 2d. per gallon to 1s. 10d. Naphtha has also risen; crude 20 per cent at 120° C. fetching 4d. per gallon, and naphtha solvent 90 per cent at 160° 1s. per gallon. The lead derivatives maintain their prices, and flake litharge fetches £17 5s., powder litharge £17.15, red lead £16.15, an increase of 5s. per ton in each case. White lead has risen 10s. to £18 per ton. The potash, soda and sulphur compounds are unchanged.

The metal trades generally are in a very prosperous condition, the rise in regard to copper is a phenomenal one. The price on Oct. 31 was £71.5 per ton. On Nov. 30 the price had risen to £78. A heavy demand is reported from all quarters. Tin has risen from £140 15s. to £150 10s. per ton, but a small decline is anticipated. Lead has risen from £15 5s. to £16 2s. Hematite pig iron fell from 70s. at the beginning of the month to 68s. 6d. by the 8th, and recovered to 70s. 6d. by the 10th, and closed at 69s. 6d. Cleveland pig iron has fluctuated slightly, the closing price, 52s. 8d.

The all-round general activity is exceedingly agreeable, and one of the symptoms of improving trade has been [illegible] from some leading accumulator manufacturers notifying a rise in price.

LONDON, Dec. 6, 1905.

Fixation of Atmospheric Nitrogen.

The problem of fixation of atmospheric nitrogen is [illegible] attracting very considerable attention in Europe.

The "Societa di produtti azotati," in Rome, has finally acquired all patents of Dr. Frank, of Charlottenburg, for the manufacture of [illegible] by the action of [illegible] gen [illegible] calcium carbide. See our Vol. I, p. 448 and Vol. III, p. 70. The process consists in passing [illegible] powdered calcium carbide, preliminarily heated to [illegible]. When the reaction is started, it continues [illegible] small consumption of energy. The [illegible] thus formed contains about [illegible] as fertilizer.

The first plant, which will soon start operation, is located at Biano del Orte. Its capacity for manufacture of calcium carbide will eventually be 10,000 hp, of which 3,000 hp will first be installed.

It is also reported that a Norwegian company will take over the installation of the Birkeland-Eyde process, at Arundal, for the production of nitric acid from air. (See our Vol. II., p. 399 and 507, and Vol. III., p. 33.) This plant is at present in operation with a capacity of 3,000 hp, which will be enlarged to 30,000 hp.; it produces calcium nitrate.

Report of the Karlsruhe Meeting of the German Bunsen Society.

By H. Danneel and J. K. Clement.

This concludes the report, the first parts of which were published in Vol. II., pp. 327, 466.

Electrode Potentials.

Ostwald has suggested that the electrolytic potential of the mercury drop electrode (Tropf Elektrode) be taken as zero; the proposition was based on Helmholtz' theory of electrocapillarity. Ostwald also introduced the calomel standard electrode as a practical reference electrode, and gave it the value 0.56 volt.

Nernst showed later on that the Helmholtz theory is somewhat uncertain, and suggested that the potential of the hydrogen standard electrode be taken arbitrarily as zero. He emphasized the uncertainty of our information regarding the absolute zero of potential and the clear chemical meaning which the metal potentials have when they are measured against the hydrogen electrode.

Since then there has been an unfortunate pell-mell in the German chemical literature on the subject. At the instigation of the Bunsen Society's committee on units and measurements, two reports were presented at the meeting on the advantages and disadvantages of both methods of measuring electrode potentials.

Prof. R. Luther, Leipsig, represented the Ostwald school. Which electrode is to be selected for reference electrode? Both the calomel and the hydrogen electrodes are easy to reproduce. The hydrogen electrode possesses the advantage that it can be used in any solution of which the alkaline liter is known. The advantages of the calomel electrode are its greater simplicity, its constant readiness for use, the smaller influence of pressure and temperature, and the equal velocity of migration of the ions present.

What numerical value is to be attached to the calomel normal electrode? For exact data the question becomes irrelevant, and the most rational value would be zero. As a matter of fact there are two values for the potential of the calomel electrode in use: — 0.560 (Ostwald), and — 0.283 volt (Nernst). Prof. Luther pointed out that the hydrogen electrode is also hypothetical. In a solution, in which the concentration of hydrogen ions = 1, the gas laws no longer apply. The speaker discussed the sign of the potentials. It is most expedient to give the positive sign to all electrodes which are positive when compared with the reference electrode.

Dr. F. Kruger, of Gottingen, represented the Nernst side of the controversy. The theoretical zero of potential is of no importance so long as no relation between the solution tension of metals and other properties are known. The absolute zero point is of no more use for a table than any other zero point. The potential drop between different solutions is in most cases negligible compared with the errors of observation. An advantage of the calomel electrode is that the ionic velocities of the electrolyte are equal. But there is an appreciable error when it is used in acid solutions.

The calomel electrode is, therefore, suitable for comparisons in neutral solutions, the hydrogen electrode in acid and alkaline solutions, and especially where an oxidizing or reducing action takes place. The principal advantage of the hydrogen electrode is that it introduces no foreign ions into the solution.

The hydrogen electrode is of great didactic importance in the consideration of oxidation and reduction cells. After considerable discussion the decision was left to the committee on units and measures.

Explosive Antimony.

Prof. E. Cohen, of Utrecht, presented a paper on this subject. By the electrolysis of solutions of $SbCl_3$, or of other antimony haloids, a metal is obtained which explodes when roughly handled; it becomes hot and emits a white vapor. The latter consists of gaseous antimony haloids, which are always deposited by the electrolysis, and form with the metal a solid solution. The heat is produced by the transformation of the metal into another allotropic modification. The explosive metal is obtained only under definite conditions of temperature and concentration.

Crystalline Liquids.

Dr. R. Schenk, of Marburg, spoke on the nature of crystalline liquids. A number of liquids exhibit the phenomena of double refraction and are optically active. Up to the present time twenty-one of them have been found. They belong to the most varied groups of organic compounds, and one of them is inorganic, potassium iodide. On cooling slowly, needle-shaped, uniaxial crystals separate from the melt. On account of the surface tension, the corners of the liquid crystals are rounded. In a magnetic field the drops arrange themselves in certain definite directions. On heating, the crystalline liquids change to the ordinary liquid state. The point of transformation is quite like a melting point; *e. g.*, it is depressed by the addition of a foreign substance. Tamman and Quincke maintain that the liquid crystals are two-phase formations (suspension of crystals or emulsions). The author considers that they belong to a separate state.

Spectrometry for Determinations of Concentration.

Dr. F. Löwe, Jena, presented a paper on new applications of spectometry. The speaker exhibited a new refractometer, made by Zeiss & Co., which permits determinations of the concentration of alcohol with as great accuracy and in much less time than with the pycnometer. Printed descriptions can be obtained from Zeiss & Co., Jena, Germany.

Atomic Weight of Bismuth.

Dr. Gutbier, Erlangen, discussed the atomic weight of bismuth. He found by the oxidation of pure bismuth 208.02, by the reduction of Bi_2O_3 by means of NH_3 and Ag 208.03, and by the determination of the Cl in $BiCl_3$ as AgCl, 208.05.

Ozone.

Prof. Luther, Leipsig, read a paper on ozone. It acts in most cases a divalent oxidizing agent; one molecule O is consumed in oxidation and O_2 is liberated. In a few cases only are all three O molecules active, viz.: with substances which are oxidizable by O_2.

The valence in any given reaction can be determined with the aid of Nernst's formula, by measuring the e. m. f. of an electrode of inactive material in ozone solutions of varying concentrations. For a ten-fold change in the concentration of ozone the e. m. f. will change 0.058 volts when the ozone is univalent, 0.029 volts when it is divalent. The author found that a ten fold increase of concentration produced an increase in potential of 0.054 volt on Pt electrodes. Ozone is, therefore, univalent in this reaction. The author does not accept Gräfenberg's view that ozone is 6-valent. Gräfenberg attempts to explain his view by assuming that ozone is 6-valent in gas form only, not in solution. If this be true, Henry's law of absorption does not apply, doubling the gas pressure would produce a 64 fold increase of the concentration in the solution. The author determined the absorption of ozone and found that Henry's law holds.

The author found that the oxidizing potential of ozone in equal concentrations is 0.2 volt lower on iridium electrodes than on smooth platinum electrodes, and the variation of e. m. f. corresponding to a ten-fold change in concentration is 0.027 volt, *i. e.*, ozone is in this case divalent:

$$O_3 + 2(-) = O_2 + O''$$

or

$$O_3 + 2(-) + 2H = O_2 + H_2O$$

Where (—) means a negative charge. We might distinguish between a univalent, active and a divalent, passive, ozone; and as is the case with chromium, the reaction and not the substance acted upon may determine the e. m. f.

We should be cautious in interpreting the potential of O_2. This amounts to 1.07 volts on iridium electrodes, and reaches 1.15 volts on platinized platinum.

In the discussion, Prof. Haber said that according to his calculations the e. m. f. of the oxyhydrogen gas battery is probably greater than 1.2 volts. Nernst stated that he had calculated the e. m. f. of the gas battery from the dissociation of CO_2 of water vapor, and of HCl. Its value can be safely placed at 1.2207.

Oxidation of Palladium.

A paper on this subject was presented by Dr. L. Wöhler. Palladium sinters at 875° and then oxidizes rapidly. Palladium foil becomes green at 830°. The author has determined the dissociation constants and the heat of formation of the palladium oxidation. The heat of formation is 24 calories. According to his experiments, palladium suboxide does not exist. Platinum oxidizes at temperatures above 400° only. Pt — oxide decomposes at 560°. For this reason the oxidation of platinum has often been overlooked. Oxidized Pt contains 70 per cent Pt — monoxide and 30 per cent Pt — dioxide. After repeated oxidation and decomposition the Pt deteriorates: its capacity for O decreases, *i. e.*, the decomposition of the oxide is more rapid than the formation.

Production by Light of a Reversible Reaction.

Prof. F. Haber spoke on the precipitation of ferrous ions in aqueous solutions of potassium ferrocyanide, describing a typical case of the production by light of a reversible reaction in a homogeneous system. The formation of potassium ferricyanide from potassium ferrocyanide and the green coloring of the ferrous salt, under the influence of light, is non-reversible. On the other hand, the action of light on alkaline solutions of potassium ferrocyanide increases the concentration of ferrous ions to such an extent that they may be detected by reactions which do not take place when light is excluded The dissociation of potassium ferrocyanide increases in the light and decreases in the dark. In the light, iron can be precipitated from it as iron sulphide; or, when O_2 is passed through the solution, as ferric hydroxide. The precipitation does not take place in the dark. The author exhibited an apparatus containing an "Uviol lamp," in which ferrous ions were precipitated by the usual reagents. The more alkaline the solution the easier it is for the reaction to take place.

Haber's theory is that the light rays are unequally absorbed by the various [illegible] the undissociated, complex [illegible] CN' ions combined. The absorbed light produces further dissociation until the amount of energy absorbed by complex and by simple ions is the same.

On account of the disturbance of the equilibrium the simple ions reunite with evolution of heat, light separates them again, and so on; *i. e.*, the solution is a machine for transforming radiant energy into heat. The amount of energy supplied by sunlight in 1 second, to a surface of 1 sq. cm. area, is the product of the pressure of light [illegible] gr. per sq. cm. times the velocity of light (3×10^{10} cm. per sec.), or 1.2×10^5 gr. cm. per cubic cm.

A surface of 1 sq. cm. area would receive 12,000 calories per hour. Only a small part of this is used, as the amount of dissociation necessary to give a reaction of ferr[illegible]

Magnetic Compounds from Non-Magnetic Elements.

Dr. E. Wedekind spoke on magnetic compounds from non-magnetic elements. The author found that the manganese borides, MnB and MnB_2, which he had prepared by the Goldschmidt process, are magnetic, both in compact form and in the form of a black crystalline powder. Arsenide of manganese is non-magnetic. Manganese bismuthide is magnetic, although bismuth is known to be strongly diamagnetic. That these are compounds and not alloys is shown by the fact that the excess of either component can be removed in a stream of chlorine without the compound decomposing. The magnetism is permanent. A glass tube, filled with the magnetized powder and suspended by a thread, comes to rest with its axis in the magnetic meridian. Shaking produces demagnetization.

Conduction in Metals.

Dr. Max Reinganum spoke on the electrochemical [illegible] and the electric conduction in metals. It is interesting to find an indication that the Faraday principle of electrolysis applies also to metallic conduction.

The relation of electric conductivity k to heat conductivity h is expressed by the equation

$$\frac{k}{h} = \frac{1}{2} \frac{mu^2}{eT}$$

where u is the mean molecular velocity, e the charge and T the absolute temperature. The term $\frac{mu^2}{e}$ can be calculated directly for electrolytes, and for metallic conductors it can be calculated from $k \div h$. The two values are identical. For the particles which carry the current in metallic conduction, the relation of kinetic energy to charge is the same as for a univalent ion. If the kinetic energy of both is the same, the charge of a particle in metallic conduction is the same as the charge of a univalent ion. The author calculated that the charge of an atom of copper is equal to the charge of two free ions.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

Industrial Electrochemistry.

Tin Refining.—In "Elektrochem. Zeitschrift" of 1905, pages 136 to 140 and 161 to 164, Dr. Hans Mennicke further discusses the electrolytic refining of tin, basing his experiments and calculations on Betts' process for the electro-[illegible]type refining of lead. It seems possible that he has [illegible] looked one thing in applying the methods of Betts' process to tin refining, and that is that the [illegible] and the lead refining [illegible] because it has [illegible] other strong acids, except the sulphuric, on the lead salt;

Very probably the author would have gotten practically the same results with tin sulphate that he has with tin fluosilicate, and would have had a more convenient solution to prepare.

The most troublesome impurities in tin are antimony, copper, bismuth, arsenic and silver. The treatment of electrolytic tin slime should be cheaper than the treatment of lead slime, as a mere treatment with sulphuric or nitric acid will dissolve out most of the impurity. Theoretically the electrolytic refining of tin should succeed, but the author questions whether small-scale experiments would be capable of demonstrating this. Experiment shows that the loss of tin in the anode slime is greater than the corresponding loss of lead. The cathode tin is also found to contain lead, unless the anodes contain very little lead. The author does not fear that cobalt, nickel, iron and zinc will interfere with commercial success. "Work-tin" usually contains iron, lead, copper, bismuth, silver, antimony, arsenic, beside zinc and tungsten. Almost all bismuth, silver, copper, antimony, arsenic and tungsten remain in the slime. Success will depend not only on the process but on the grade of raw tin, which varies widely.

Usually the author used solutions containing about 10 per cent tin and 10 per cent free fluosilisic acid, with an e. m. f. averaging 0.4 volt, sometimes as high as 3 volts. The current density averaged 1 amp. per square decimeter (9 amps. per square foot). No gelatine or albumen was ever added. The temperature was about 20° C. The separation of electrodes was usually 5 cm., sometimes 7.5 cm. Large quantities of iron, zinc, nickel and cobalt made no trouble in the solutions. Tin oxide was found to be insoluble in fluosilisic acid. Freshly precipitated stannous hydrate was used to prepare the solution. Better results were obtained with raw tin than with tin alloys. The cathode deposits were extremely solid and tough. The lower the voltage the finer the tin crystals. With a cold electrolyte the author successfully detinned tin cuttings, producing a solid cathode deposit, which would not, however, be a practicable process for commercial work.

The author experimented with tin-lead alloys as anode with the same conditions of current, etc., as above. The anode was strongly but evenly attacked, and the cathode metal, especially on the start, was good. The electrolyte, however, clouded, and whitish deposits of lead fluoride formed on the anodes, showing the presence of HF in the electrolyte. The cathode deposit gradually became spongy, but consisted of absolutely pure tin. The author states as a conclusion that the presence of H_2SiF_6 in the solution will make a solid tin deposit if lead is absent. The current yield was found to be about 90 per cent, but grows less as the process is carried further. Perhaps the author's solutions also contained tin difluoride (from an excess of HF in the H_2SiF_6 used in making the solution), as appears from the observed formation of lead fluoride on anodes of solder. Tin difluoride should be an excellent salt for electrolytic refining, as basic salts would not tend to separate.

Electrolytic Nickel.—In our first volume a great amount of information was given concerning electrolytic nickel. Two processes were formerly used. The Browne process, at Cleveland, for the treatment of copper-nickel matte of the Canadian Copper Co., and the Thum process, at the Balbach works in Newark, for refining nickel shipped from the Orford Copper Co. The former process was discontinued when the works of the Canadian Copper Co. were bought by the "Nickel Trust," but this process will be put again into operation when the new electric power transmission of the Canadian Copper Co. to Copper Cliff is completed. The Balbach works stopped operation of the Thum process when the orders of the Orford Copper Co. for refining nickel stopped coming in. According to the November issue of the "Brass World" electrolytic nickel refining has now been taken up by the Orford Copper Co. "The nickel is deposited by means of the electric current upon a sheet of nickel which has previously been coated with graphite, to prevent the adhesion of the deposit to the cathode surface. When the requisite thickness has been obtained the deposit is removed from the cathode plate and is ready for use. The cathode plates of nickel, as now produced, have the dimensions of 3 x 4 feet and about ¼ inch in diameter. The average purity is 99.5 per cent. It is absolutely free from carbon. Electrolytic nickel is in special demand by manufacturers of german silver. It is mentioned that the electrolytic nickel plates are so tough that they must be cut by means of power shears; this is a disadvantage. The surface is "wart-like." "While the process is a secret one, we understand that a solution of chloride of nickel is used as electrotlyte. By this method it is possible to prevent the contamination of electrolytic nickel with sulphur, as would be the case were a sulphate solution used as the electrolyte." A chloride solution is also used in the Browne process.

Production of Copper Salts.—In "L' Industrie Electro-Chimique," of July and August, the following process of E. Campagne for the production of copper salts is described. If a solution of sodium sulphate is electrolyzed between copper electrodes, copper goes into the solution from the anode as copper sulphate, and sodium hydroxide is formed at the cathode with evolution of hydrogen. The former two react together and produce cupric oxide and sodium sulphate. The cupric oxide sinks to the bottom in form of a powder, and, being soluble in acids, it may be used as a very convenient starting material for the production of various copper salts. Some precautions which must be taken are mentioned, and the industrial application of the process for the production of copper sulphate and copper acetate are described.

Metallic Calcium.—For some time metallic calcium has been on the market in Germany, being made by the Elektrochemische Werke, Bitterfeld, according to the patents of Rathenau. The principal feature of this patent was the gradual rise of the metallic calcium cathode during electrolysis, so as to prevent the calcium that is being deposited from being redissolved in the electrolyte. Essentially the same method (although with differently designed apparatus) was used by J. H. Goodwin (our Vol. III., p. 80), whose complete paper may be found in the November issue of the "Journal of the American Chemical Society." Another experimental investigation of the same process was made by P. Woehler, who describes his results in the "Zeit. f. Elektrochemie," Sept. 8. The author used an electrolyte consisting of 100 parts of calcium chloride and 17 parts of calcium fluoride. The melting point is 660° C. (while the melting point of pure calcium chloride is 780° C.). The melting point of metallic calcium is 800° C. The author maintained the electrolyte at a temperature between 665 and 680° C., the highest temperature being used at the cathode, so that when the calcium rod is raised out of the fused bath the fused electrolyte adhering to it is of sufficient fluidity to flow down into the bath. Only in this way a pure metallic calcium rod can be obtained. The author used 40 amps. at 38 volts with a cathode current density of 100 amps. per square centimeter, and thus produced calcium rods of any length at will. The process runs well and with a high efficiency as long as one uses a fresh neutral electrolyte. But the longer the electrolyte has been in use, the poorer becomes the output. The cathodic current density is of little influence. The operation was equally favorable when at a constant current of 40 amps. The current density was gradually raised from 50 to 250 amps. per square centimeter (by diminishing the

cathode diameter from 10 to 45 mm.) The anodic current density must not be too low, since otherwise the electrolyte freezes around the anode and the chlorine gas accumulates through the whole electrolyte; neither must the anodic current density be too high (above 5.6 amps. per square centimeter), since otherwise a great many small arcs are formed between anode and electrolyte, with a large increase of e. m. f. With a fresh electrolyte the ampere-hour efficiency was found to be 82 per cent. "If the kw-hour is charged at 1.25 cents, the cost of electrical energy for the production of 1 kilogram of metallic calcium—namely, 62.12 kw-hours—becomes $0.78. The market price is $4.50 per kilogram." The specific weight was determined as 1.51.

Aluminium.—A statistical review of the aluminium review in Europe is given by R. Pitaval in the "Journal de l' Electrolyse" of Oct. 15 and Nov. 1. Although the Heroult patents have now run out, new producers have not entered the field; but the old companies have increased their plants and are dominating the bauxite supply. There are four large European companies producing aluminium: the Societe Electrometallurgique Francaise, producing 1,000 tons per year (Froges, La Praz, St. Michel); Societe des Produits Chimiques d' Alais, 600 tons; the Aluminium Industrie A. G., 3,500 tons (Neuhausen, Rheinfelden, Gastein); the British Aluminium Co., 600 tons.

Reduction of Metallic Oxides by Aluminium Carbide.—The only commercially successful method of producing aluminium is by electrolysis, and all attempts of making aluminium in the electric furnace by reduction of aluminium with carbon have been unsuccessful. Although it is possible to prepare alloys of aluminium by reduction of the oxide by carbon in presence of such metals as copper and iron, yet when it is attempted to obtain the pure metal by direct reduction, the product is almost exclusively aluminium carbide. The behavior of aluminium carbide when heated in contact with metals and metallic oxides is, therefore, of considerable importance to the metallurgy of aluminium, and a series of experiments made on this problem by J. N. Pring, at Manchester University, and published in the "Transactions of the (Brit.) Chemical Society," 1905, Vol. LXXXVII., pages 1530 to 1540, contain much that is of interest. To prepare aluminium carbide the author used the method of Moissan, which consists in heating aluminium contained in a carbon crucible in his ordinary arc type of furnace. Some figures are given on power consumption and yield. It appears that the aluminium carbide formed results chiefly from the action of carbon monoxide on aluminium. The author studied the reactions of aluminium carbide with oxides up to about 1,200° in a wind furnace, first without a flux and then in presence of cryolite as a flux. A further series of experiments was made at higher temperatures in an electric resistance furnace.

The chief results are as follows: The work here recorded seems satisfactorily to explain the reactions which occur between aluminium carbide and metallic oxides and metals. Up to about 1,400° the carbide behaves as a strong reducing agent, but both aluminium and carbon are simultaneously oxidized even when the carbide is in excess. Thus, under these conditions no separation of aluminium or of carbon can be detected. For instance, aluminium carbide interacting with copper oxide produces metallic copper and carbon dioxide, together with small quantities of carbon monoxide. A similar behavior is noticed when the reactions are brought about in the mass of a flux so as to facilitate the agglomeration of the reduced metal. At higher temperatures, however, selective reduction begins to be apparent, the reduction being more and more brought about by the carbon of the carbide, with a result that alloys of aluminium and the reduced metal are obtained, the percentage of aluminium increasing the higher the temperature is raised.

In the case of copper, the percentage of aluminium in the alloy seems to be [illegible] aluminium. Copper alloys were obtained containing up to 28 per cent of aluminium, whereas the equation

$$3CuO + Al_4C_3 = 4Al + 3CO$$

demands an alloy containing 36 per cent of aluminium.

In the case of iron, an alloy was obtained containing 40.7 per cent of aluminium, corresponding (the [illegible] free carbon, 49.6 per cent of aluminium. This corresponds to the equation

$$FeO + Al_4C_3 = 2Fe + 4Al + 3CO$$

which demands an alloy containing 49.2 per cent of aluminium. Any excess of oxide used in the reaction resulted in oxidation of the aluminium in the alloy. When the reaction between aluminium carbide and [illegible] oxide was brought about in presence of a bath of molten iron at a high temperature, alloys were also obtained in which more than 90 per cent of the aluminium [illegible] had been set free and taken up by the iron.

Calcium carbide exhibits, in a smaller degree, a behavior similar to that of aluminium carbide in the relative affinities of the calcium and the carbon for oxygen at high temperatures. Reactions between lead oxide and excess of molten calcium carbide gave an alloy containing 2.6 per cent of calcium.

These results are explained by the fact that at high temperatures alumina can be reduced by carbon, and the following equation, which at comparatively low temperatures is known to go in the direction from left to right, would, at high temperatures, apparently be reversed and occur in the opposite direction:

$$6Al + 3CO = Al_4C_3 + Al_2O_3$$

Owing to the evolution of heat in these reactions between oxides and aluminium carbide it was not possible to fix or estimate the temperature at which reaction actually took place. Aluminium carbide reacts with copper at a temperature below the melting point of platinum, and with iron just above this temperature, to form an aluminium alloy with liberation of free carbon. At higher temperatures the reaction takes place with violence and is complete.

Calcium Carbide.—In the "Journal de l' Electrolyse," R. Pitaval gives a summary of progress in calcium carbide manufacture in France during recent years. Instead of arc furnaces resistance furnaces are now used. While one hesitated formerly to use a higher furnace capacity than 300 hp., now the resistance furnaces of the Clavaux plant have a capacity of 500 hp. each, and in the Dario works (Froges de Veltri) furnaces with two electrodes and of 1,200-hp. capacity will be installed. Formerly the usual voltage was 60, now the tendency is towards 40 or 30 volts. The mean consumption of carbon electrodes is now 50 to 60 kilograms per ton of carbide, but with the latest types of furnaces this may be reduced to 20 kilograms. The greatest care is now taken to use pure and dried raw materials, anthracite and lime, at least 95 per cent pure.

Calcium Carbide and Acetylene.—An interesting paper, which has not yet been noticed in these columns, may be found in the "American Gas Light Journal" of July 10, 1905. The author, J. B. Marshall, [illegible] historical sketch especially of the experimental [illegible] the Wilson Aluminium Co., in North Carolina. Their object was to make aluminium by reducing alumina by means of metallic calcium, and they attempted to make the metallic calcium by reducing lime with [illegible] Lime and coal-tar were mixed and put in an electric furnace. After

a run, which lasted several hours, on this mixture a sample was taken out of the furnace, and in accordance with the usual custom was placed in water in order to cool it for an analysis. Large volumes of gas were given off, and at first it was thought that the substance was metallic calcium and that the gas given off was hydrogen, but upon lighting it it was found to burn with a heavy yellow flame and to give off quantities of soot, which proved to be some rich hydrocarbon gas and not hydrogen. Subsequent analysis of the substance proved it to be calcium carbide, and the gas was pronounced acetylene. This first experiment was made on May 4, 1892. The author then gave a review of the production and properties of commercial acetylene. The principal reasons for the success of acetylene in the lighting field are the low first cost of the plant, the simplicity of production, the small cost of maintenance and repairs, and the economy of the light furnished. The candle-power of acetylene light is so high that only one-tenth of the volume of acetylene is required to produce a given candle-power, as compared with the volume necessary when other gases are used. This means that in a town where ordinary illuminating gas would require a holder of 10,000 cubic feet capacity to supply one day's requirements, a holder of only 1,000 cubic feet is necessary if acetylene is used. The same reasoning applies to the size of the mains. (Acetylene is generally sold by 100 cubic feet, not by the 1,000, and the price ranges from $1 to $1.50 per 100.) The cost of producing acetylene does not depend upon the quantity of the gas generated, as no fuel or materials other than carbide and water are required. Acetylene may be produced in small quantities at quite as low a cost as it can be made in large volumes. The number of town lighting plants in this country was 4 in 1898, 8 in 1899, 30 in 1900, 39 in 1901, 60 in 1902 and 202 in 1904. In addition there are now in operation a very large number of isolated plants.

Welding by Means of Acetylene.—In the "Jour. f. Gasbeleuchtung," of Dec. 2, S. Traubel advocates the use of the acetylene-oxygen flame for autogenous welding of iron and steel. It is claimed that the cost is only one-third that of other autogenous welding methods and that the joints are always clean. He gives some data on the Fouche system, in which the pressure of the acetylene supplied to the burner is not more than 20 mm. column of water. One cubic meter of acetylene, weighing 1.165 kg., gives 12,900 calories (while 1 cubic meter of hydrogen gives only 2,580 calories in the oxyhydrogen flame). Theoretically, 2½ parts of oxygen are required for 1 part of acetylene, but in practice an excess of acetylene is used; the usual figures are 1 part of acetylene for 1.7 part of oxygen, parts meaning parts by volume. The acetylene-oxygen flame has in its center a small blue cone, the point of which has a temperature of about 3,000°, while around the cone there is an atmosphere of carbon dioxide and hydrogen, so that the iron or steel is welded in a reducing atmosphere. The acetylene-oxygen flame is smaller, more quiet and hotter than the oxyhydrogen flame. Some data are given on the cost of welding sheets of thicknesses of 1 to 9 mm. It is stated that the process is considerably used in France and Belgium. In connection with this subject the following notes from an address of R. Threlfall (London "Electrician," Dec. 8) are interesting: "At the recent exhibition at Liege there was a display of oxy-acetylene blow-pipes of large size, intended to compete with thermit for repairing wrought iron. It is very probable that with silica tubes and rods (made inside carbon tubes from sand) and suitable oxyacetylene blow-pipes it will be possible, even easy, to make any kind of apparatus, large or small, out of opaque silica. The method should also be capable of generalization for the fusion of those oxides which are reduced with difficulty by carbon; **i. e.,** zirconia, magnesia and the alkaline earths and the oxides of vanadium and titanium, provided they are presented in the right state of fineness."

Electrolytic Zinc Plating versus the Hot Dip.—"Dingler's Polyt. Jour." of Nov. 25 contains an account by I. Szirmay of comparative tests of iron and steel ware, zinc plated by electrolysis and by the hot dip. Both mechanical and chemical tests were made. The arrangement for the chemical tests is shown in Fig. 1. The zinc-plated iron plates and wires **e** are suspended within a vessel, the walls of which can be filled with hot or cold water so as to vary the temperature. The interior of the vessel was filled with air containing different proportions of sulphurous acid, carbon dioxide, moisture and steam. By alternately filling the walls of the vessel with hot and cold water the temperature was varied between the 6° and 45° C., resulting in considerable condensation on the test plates and wires. He found that during the exposition the plates covered with zinc by the hot dip became black, and that the hot dip does protect iron and steel ware against atmospheric influences. While in the plating of articles by the hot dip relatively large quantities of zinc are consumed on account of the impurity of zinc, if sufficient protection against rusting shall be obtained, much less zinc is required in the electrolytic method on account of the purity of the deposit. The large quantity of zinc and the non-uniformity of the deposit with the hot dip result in poor adherence. When sheets, tubes and wires undergo mechanical wear and tear, the zinc coating suffers, and premature rusting occurs.

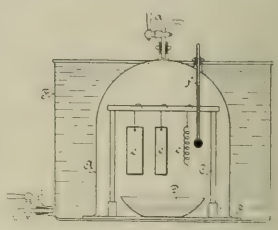


FIG. 1.—TESTING ZINC COATINGS.

THEORETICAL AND EXPERIMENTAL.

Electrolytic Regeneration of Chromic Acid.—E. Mueller and M. Soller ("Zeit. f. Elektrochemie," Dec. 1) has studied the influence of the anode material on the electrolytic production of chromic acid from sulphate. With a solution of pure chrome alum in 1-normal sulphuric acid there is practically no formation of chromic acid at a polished platinum anode. But the least traces of a dissolved lead salt accelerate the reaction extremely on account of the deposition of lead peroxide on the anode. Small quantities of chlorine ions also favor the reaction. With a lead peroxide anode the formation of chromic acid goes on very satisfactorily, while on a platinized platinum anode the ampere-hour efficiency is about one-third that at a lead peroxide anode. The good results obtained with lead peroxide are believed to be due to catalytic action, and are thought to be related to the fact that PbO_2 can react chemically with the chromium ions.

Passive State.—In the "Zeit. f. Elektrochemie," Dec. 1, C. Fredenhagen gives some points in favor of his view that the passive state of metals may be explained by a "charge" of the surface of the metal with a gas—whether the gas is assumed to be dissolved in the metal at the surface or whether there is an adhering surface film of gas or whether there is a combination of the two. The author believes that all facts which may be explained by the assumption of an oxide film on the surface are also explainable by his assumption, but that his assumption applies to more cases.

Generator Gas Cell.—A paper by F. Haber and A. Moser

in the "Zeit. f. Elektrochemie," Sept. 8, gives an interesting account of a theoretical and experimental investigation of what is commonly called the carbon cell. At a temperature of about 500° C. the e. m. f. of the carbon-oxygen cell and also of the carbon monoxide-oxygen cell (or, as the authors say, the generator gas cell), is near 1 volt, according to the theory (i. e., calculated from the free energy of the reaction). It is necessary to use an elevated temperature since at ordinary temperature the speed of the reaction between carbon and atmospheric oxygen is very low. The choice of a proper electrolyte at 500° C. would offer some difficulties, but since the authors were not after a commercial carbon cell giving strong currents, but only intended to measure the e. m. f. of such cells under varying conditions for the sake of comparison with the theory, they could use heated solid glass as electrolyte, which proved very satisfactory. (Concerning solid electrolytes see Haber and Tolloczko, our Vol. III., p. 99.) The construction of the apparatus, the theory of the subject, and the results obtained in the tests are given in detail. The results of the theory were fully confirmed at temperatures somewhat above and below 500° C. By using concentrated carbon monoxide (instead of a mixture of monoxide and dioxide) the reaction is changed from that of the generator gas cell into that of the carbon cell. The application of hot glass as electrolyte permits the realization of the oxygen-hydrogen gas cell at high temperatures.

Iron and Steel.

Carbon Deposition.—Dr. F. Zimmerman describes in "Stahl und Eisen" for July 1 an investigation on the action of carbon monoxide upon metallic oxides and metals. He brought the gas into contact with these materials at the temperatures of boiling diphenylamine (320° C.), mercury (360° C.), sulphur (445° C.) and phosphorus sulphide (508°). Neither NiO, CoO, FeO or Fe^2O^3 acted as inciters or catalysors to produce deposition of carbon at any of these temperatures. Metallic nickel was very active, the spongy metal deposited on pumice causing very rapid deposition of carbon, with evolution of heat, without the nickel being oxidized. Cobalt behaved similarly. Equilibrium was attained in about 7 hours. Pumice impregnated with metallic iron behaved very differently. Up to 400° the reaction was similar to nickel and cobalt, but much slower; at 445° and 508° the reaction was very rapid, but in the sense of the equation

$$Fe + CO = FeO + C$$

the reaction proceeding so far that gas pressures as low as 50 millimeters were produced in the apparatus. The author believes that the iron really first promotes the reaction

$$2CO + Fe = CO^2 + C + Fe$$

by catalytic action, and then is oxydized by the CO^2:

$$Fe + CO^2 = CO + FeO,$$

the CO thus formed repeating the cycle, until almost all gas has disappeared. The facts thus disclosed should have a considerable influence in determining the nature of the reactions in the upper part of a blast furnace.

High-Speed Steel.—Ledebur contributes a short article to "Stahl und Eisen," July 1, bringing forward the views of one of his students, Otto A. Böhler, on the question of the nature of air-hardening high-tungsten steel. The gist of the view is, that when this steel is cooling the Martensite which is present at a high temperature does not split up into cementite and ferrite (the mixture called perlite), but the presence of the tungsten lowers the critical temperature at which such change takes place below the ordinary temperature, so that the carbon all remains behind as hardening carbon. It is to be regretted that Ledebur published such rather crude [illegible] "crux experiments" [illegible] hardened steel to the [illegible] (like the magnetic reversability of high nickel steel [illegible] change spoken of as being [illegible] at a very low temperature. [illegible] would know whether Böhler's views were worthy [illegible] ceptance or not.

Blast Furnace Practice.—Fr. Schraml, of Pribram, discusses in "Stahl und Eisen," May 15 [illegible] the charges in the upper part of a blast furnace. He calls attention chiefly to a point usually overlooked; viz.: that since the charges are warmed by the gases, the former must always be colder than the latter, and that the difference may be very considerable. At the throat, for instance, the gases are at 250° to 450° C., while the charges are nearly at outside temperature; below the throat, where the charges are, say, at 600°, the gases must be at some 450° higher temperature than at the throat in order to be able to warm the charges to 600°, resulting in a gas temperature at that point of 750° C. From these considerations, the author draws the moral that the gases issuing from the blast furnace should be cooled down by being brought into proper contact with cold charge until their temperature is 100°, so as to utilize better their sensible heat in the furnace; but omits to point out the best way of accomplishing this end, which is to use hotter blast and dried blast, so as to run the furnace with less air blown in per ton of product, thus reducing the amount of gas passing upwards per unit of charge, and thus most effectively lowering its temperature.

Reduction of Ores.—Oskar Simmersbach, of Dusseldorf, makes some proposals in "Stahl und Eisen" of Oct. 15, which are more interesting than practical. He calls attention to the fact that iron ore is used on a large scale in open-hearth furnaces in connection with pig iron, and that, although only about three-quarters of the iron of the ore used is reduced to the metallic state, yet the iron as ore costs only two-thirds as much as iron in pig iron, and there is, therefore, pecuniary advantage in using as much ore as possible. The reduction of the ore to metal is performed by the action of silicon, phosphorus, manganese and carbon in the pig iron used, and the more highly oxidized the ore used (FeO, Fe^3O^4, Fe^2O^3) the less iron is reduced into the bath. The gist of Simmersbach's proposal is to previously partly, or nearly entirely, reduce the ore used to the metallic state, at least, let us say, to partly deoxidize it, so that when subsequently added to the pig iron bath in the steel furnace a much greater amount of iron is reduced into the bath. The reduction is to be done in a cylinder, in which the ore is first heated by burning blast-furnace gas in contact with it, and then deoxidized by passing blast-furnace gas alone through the heated ore, which is afterwards cooled out of contact with the air, and then used in the open-hearth furnace. The operation proposed would require much gas and entail extra labor charges, so as to probably eat up all the margin between cost of iron in iron ore and in pig iron. A more practical method of accomplishing the same end might be to impregnate a porous form of ore with carbon, such as coal dust, or to briquette it with coal dust, or impregnate it with waste tar or pitch, and so that form add [illegible] the pig iron bath. The hot carbon would then [illegible] absorb some oxygen from the ore, and so increase the amount of iron reduced into the bath.

Miscellaneous.

Gas Power.—As was noticed in this [illegible] Vol. III., p. 52, the New York Section of the Society of Chemical Industry devoted one evening last year to [illegible]

gas as a source of power. The complete report of this very interesting and successful discussion is now available in Vol. XXIV. of the "Journal" of the Society of Chemical Industry. No. 11. It contains the following papers: Willard L. Case, on generation of producer, Mond and blast furnace gas; Oskar Nagel, on the utilization of gas from suction producers; C. G. Atwater, on coke-oven gas; J. D. Pennock, on the Mond producer, with some shorter remarks by F. Schniewind, Franz Meyer and E. A. Uehling. This set of papers contains a very large amount of useful and reliable information, based on actual practice, and should be consulted by anyone interested in the subject from an industrial point of view. The theoretical side of the subject is presently discussed in J. W. Richards' Metallurgical Calculations, published in our columns.

Liquid Air.—At the recent meeting of the Association of German Naturalists, Pictet presented a paper in which he mainly attacked the theory on which Linde's method of liquefying air is assumed to be based. An abstract of the paper is given in "Zeit. f. Elektrochemie," Nov. 3. "Liquid air may find a useful application in practice, since nitrogen first evaporates from it and oxygen follows later. It is possible to obtain oxygen of any desired purity from liquid air. A 50 per cent oxygen, the so-called industrial oxygen, may find useful applications in many industries."

Cleaning Blast Furnace Gases.—Krull describes in "Glueckauf," of Oct. 28, the new cleaning and cooling apparatus for blast furnace gases devised by Bian and exhibited at Liége. The gas is purified so as to contain about 0.5 gram of dust per cubic meter, and the temperature is reduced to about 40° C.

Refractory Materials.

Magnesite.—E. Daumann, in a recent number of the appendix to "Jernkontoret's Annaler," describes the new magnesite industry of Norway. At the end of 1904 the "Norska Magnesit-Akteebolaget," with a capital of $50,000, commenced to exploit the deposits near to Snarum in Southern Norway. These deposits cover considerable ground, at a thickness of 4 meters. Works, with a yearly output of 2,500 tons of burned product have been erected, at which the rock is first crushed, then burned at a low, white heat; then ground, pressed into bricks under 200 tons pressure, and then burned at high white heat in regenerative furnaces. Analysis of a magnesia brick from this source shows 83.6 per cent magnesia, 4.6 ferric oxide, 2.0 alumina, 0.05 manganese monoxide, 9.3 silica, 0.046 phosphorus pentoxide, 0.003 sulphur, 0.50 loss on ignition. The bricks contain no lime, but are higher in alumina and silica than Styrian magnesia bricks. Dr. Bischof, of Weisbaden, testifies that the Norwegian bricks "are decidedly more refractory than the Styrian." It is interesting to know that increased supplies, of improved quality, of this very useful refractory are obtainable.

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

Electric Furnaces.

Steel Mixer.—P. L. T. Héroult, 807,027, Dec. 12, 1905. Application filed April 19, 1905.

The inventor describes a mixer for steel of considerable capacity, to secure uniformity in the product of an entire works to secure a certain desulphuration and recarburation and conserve the oxidizable additions—such as silicon, manganese and the like—in order to reduce the quantities of these additions employed to a minimum. By reason of the metal being at a high temperature and in a very fluid condition, the oxides of silicon and manganese rise to the surface, leaving a clear product. Heat from the electric current has the advantage that it is non oxidizing; moreover, the inventor maintains in the mixer a non-oxidizing atmosphere by the introduction, for example, of producer gas. The slag is thereby maintained deoxidized so as to favor the desulphuration of the metal. The mixer is closed. The aperture through which the electrodes pass is preferably made tight by a water-cooled pressure-gasket. The atmosphere within the furnace is maintained under pressure by the gas introduced from the producer.

"The largest charge ever smelted in an electric furnace, as far as I am aware, has been about 4½ tons. It is proposed to make this mixer large enough to carry a charge of 300 or 400 tons, to be fed by five or six Bessemer or Siemens-Martin furnaces." It is, therefore, of greatest importance to make the radiation losses as small as possible by providing a minimum surface. With a pig-iron mixer the temperature of liquefaction is about 1,050° C., and it is only necessary to pour in fresh iron from the blast furnace at sufficient intervals in order to maintain the mass in a liquid state. With steel the temperature of liquefaction is 1,600° to 1,800° C., and it is necessary to apply additional heat.

An apparatus, "having a capacity of 100 tons, will require in order to maintain its temperature at that of the fusion of steel about 1,500 hp. If 2,500 hp. is furnished to the apparatus, there will remain 1,000 hp., producing 635 calories per horse-power per hour, which places at the disposal of the bath of steel 635,000 calories per hour. The specific heat of steel being 0.25, said quantity of heat is capable of reheating through 100° for an hour about 25 tons of steel, which are supposed to have been furnished to the mixer at a temperature already sufficient, or nearly sufficient, for tapping."

A 100-ton furnace, capable of reheating through 100° 25 tons for an hour, or of reheating through 50° 50 tons for an hour fed by a 2,500-hp. generator, should preferably have about the following dimensions: Surface of the bath, 26 square meters; depth, 1 meter. The electrodes, using a single-phase current, may be about 95 centimeters square in cross-section by 5 meters high, so as to assure their lasting six or seven weeks. With a three-phase current, using three electrodes, the cross-section should be proportionally reduced.

Referring to Fig. 1, a crucible a is closed by a cover b, through which pass three electrodes $c^1c^2c^3$. Each of these electrodes is carried by an arm, such as d^1 (or d^2 or d^3), of which the support e^1 is in engagement with, and may be raised or lowered by means of a system of pinions and an endless worm controlled by an electric motor f. Each vertical support e^1 is guided by rolls g between U-shaped members fixed upon the head of the crucible with interposed insulating material. Each carbon or electrode is embraced by a collar of metal sheets which is compressed or slackened by means of a screw h, operated by a wheel i through a suitable intermediate system of gearing. The crucible a carries a spout j, through which the metal may be poured out when the furnace is inclined. There are three doors $k\,k^2k^3$, corresponding, respectively, in position with the electrodes. The slag is ordinarily poured out of the rear doors k^2 and k^3, and the metal out of the front door k^1. The electrodes are surrounded by cooled pres-

sure-gaskets l. The crucible a is also provided with a passage m in communication with a gas producer n, from which the gases are led into the crucible a, which in operation is entirely closed. The metal and slag ordinarily occupy the positions indicated in Fig. 1.

The crucible a carries curved tracks, which rest upon a suitable pedestal o, and has its other point of support on

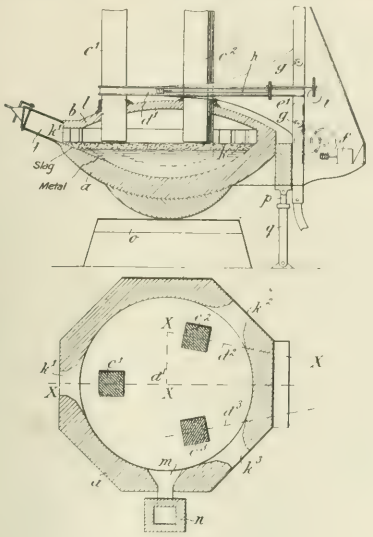


FIG. 1.—STEEL MIXER.

the articulated piston of a hydraulic cylinder q, which is also articulated and which serves to determine the angle at which the crucible is tipped.

Converting Cast Iron into Steel.—P. L. T. Héroult, 807,026, Dec. 12, 1905. Application filed May 19, 1903.

This is a combination patent for using an ordinary metallurgical steel process with subsequent refining in the electric furnace. Cast iron is first refined by oxidation, for instance, in the Bessemer converter, until the content of the molten metal in carbon has been reduced to between 0.02 and 0.05 per cent. When phosphoric pig is treated the oxidation is pushed further, until the phosphorus has been practically completely oxidized and has passed into the basic slag. The metal, instead of being further treated by addition of carburizing material, more or less rich in manganese, silicon, etc., in the converter itself, is then passed into the electric furnace, where it is kept in a state of fusion and is changed into steel of the composition wanted by the addition of carbon (in the form of "carburite") and of any desired ferro-alloys. The following table shows for a given percentage of carbon the minimum content of oxygen which may be obtained with the Bessemer converter and the electric furnace respectively (assuming an absence of manganese and silicon and the highest temperatures obtainable in the two apparatus:

	Bessemer.	Electric.
C = 0.05 or less.....	O = 0.75	O = 0.30
C = 0.10.....	O = 0.50	O = 0.01
C = 0.20.....	O = 0.30	O = 0.00
C = 1.60.....	O = 0.01	O = 0.00

In the Bessemer and open-hearth processes it is impossible to carry the oxidation sufficiently far to completely eliminate the impurities initially present, without leaving a considerable quantity of oxygen; but superoxidized metal, thus produced and formerly useless, can [illegible] factorily treated in the [illegible] gives the following table illustrating the [illegible] prises a good basic Bessemer [illegible] has own process.

	Bessemer.	Heroult
Carbon	0.10	0.005 [illegible]
Silicon	0.01	0.000 [illegible]
Manganese	0.50	0.000 [illegible]
Phosphorus....	0.07	0.005
Sulphur.....	0.07	0.005

Some of the eight claims seem very [illegible] portance for the industrial development [illegible] metallurgy of steel. The first claim reads: "A process for converting cast iron or the like into high grade or 'crucible' steel, which consists in first treating the metal in a Bessemer converter, then transfering [illegible] furnace, in which it is finally treated to deoxidize and carburize it." The third claim reads: "A process for converting cast iron or the like into high grade or 'crucible' steel, which consists in first refining it by oxidation and then removing the slag and transfering the remainder to an electric furnace, and there deoxidizing and transforming it into the desired product."

Resistance Furnace.—J. S. Dorian, 805,783, Nov. 28, 1905. Application filed March 27, 1905.

The construction is shown in Fig. 2. The carbon lining D of the furnace is connected with one pole of the electric supply, while the other pole is connected with the hollow graphite tube G; H being an insulating material. Inside of the graphite tube G, the resistor proper I is provided, which is brought to incandescence by the electric current. The temperature is regulated by raising or lowering the graphite tube G, so as to bring a greater or smaller portion of the resistor I to incandescence. If the exposed part of the resistor rod I burns off, or becomes broken, the same is replaced by sliding it downwardly in the graphite tube G, until it again makes contact with the bottom D, so that operation does not need to be discontinued.

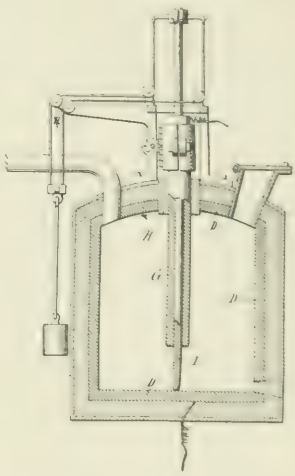


FIG. 2.—RESISTANCE FURNACE.

Resistance Furnace.—R. W. Meyers, 806,173, Dec. 5, 1905. Application filed Feb. 14, 1905.

The invention uses [illegible] etc., which have a very high resistance at ordinary temperatures, but become conductors when heated. These he calls "pyro-electrolytes." They are formed into bars or rods and supported by graphite or metallic blocks. To start the furnace, the "pyro-electrolytic" [illegible] heated by a blow pipe, etc. To prevent [illegible] rent from passing through the furnace [illegible] graphite or metallic supports, the [illegible] outside shunt, the resistance of which [illegible] creasing temperature.

Metals and Alloys Free from Carbon.—F. von Kuegelen and G. O. Seward, 807,034, Dec. 12, 1905. Application filed Jan. 16, 1904. (Assigned to Willson Aluminium Co.)

In an electric furnace with carbon electrodes it is difficult to obtain chromium, ferrochrome, ferrotungsten, etc., free from carbon. The inventors cast the high-carbon metal or alloy, obtained in ordinary electric furnace practice, in form of pencils, and employ two such pencils in an electric refining furnace in which the carbon is eliminated by the decarburizing action of the slag or lining. A suitable slag for refining ferrochrome is a mixture of lime and chrome ore with suitable fluxes. Claim 1 reads: "Decarburizing a metalliferous substance by fusing it as an electrode in presence of a substance having a high affinity for carbon."

Electric Furnace for Bisulphide of Carbon.—E. R. Taylor, 805,501 and 805,502, Nov. 28, 1905. Application filed April 4, 1901, and June 2, 1902.

The inventor's continuous electric-furnace process for the manufacture of bisulphide of carbon has already been described and discussed on pages 60, 63 and 76 of our Vol. I. Patent 805,501 relates to the use of self-renewing electrodes adapted to perpetuate themselves in the continuously-working furnace and to feed themselves by gravity, as described and illustrated on page 76, Vol. I. Patent 805,502 refers to the treatment of the residue which would accumulate in the bottom of the furnace; the residue is fused by the electric current and is drawn off.

Electric Ring Furnace.—K. Meiser, 806,690, Dec. 5, 1905. Application filed April 26, 1904.

Several electric furnace chambers are arranged in a ring. Of the chambers but one is always in operation, while the other chambers are being cooled down or subjected to a preliminary heating by circulating air through the ring. The furnace is adapted for cases where close heating is required, as in the manufacture of incandescent-lamp filaments, and also for preheating materials which are non-conductors at ordinary temperature. Fig. 3 shows a plan view in the upper diagram and a section on line AB in the lower diagram. The electrodes p and n, which convey the current, are arranged at the top and bottom of each chamber, 1, 2, 3, etc. The bottom electrodes remain always in position, while a top electrode is introduced only into that compartment which is just at work, i. e., in the drawings into chamber 2. Chamber 2 is heated by electricity, chambers 8 and 1 are cooled down, chambers 3, 4, 5 are subjected to a preliminary heating, chamber 6 is recharged, and chamber 7 is emptied. The heat from chamber 8, 1, 2 is conducted into chambers 3, 4, 5 by the circulating air, which enters into chamber 8, flows through chamber 1, 2, 3, 4, 5, and when slide a is open escapes from chamber 5 into conduit b. The furnace allows the air to be drawn off from the last chamber at as high a temperature as 500°, and even at higher temperatures, the entire heat which the air contains being passed into the first chamber without waste. The circulation is obtained through a ventilator provided at e, which when slide a is open draws off the air from chamber 5 into conduit b, and forces it from here into conduit f.

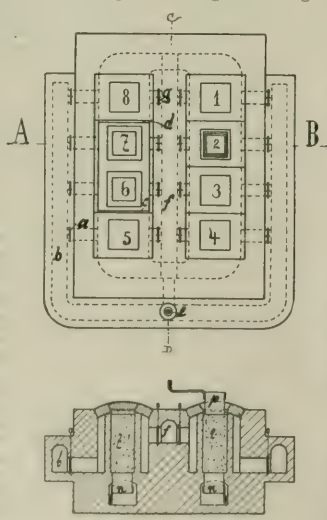


FIG. 3.—RING FURNACE.

Electrolytic Processes.

Separating Nickel and Copper by Electrolysis.—N. V. Hybinette, 805,969, Nov. 28, 1905. Application filed Nov. 25, 1904.

See the abstract under Recent Metallurgical Patents.

Production of Calcium.—O. Ruff and W. Plato, 806,006, Nov. 28, 1905. Application filed Jan. 20, 1903.

The inventors produce metallic calcium by electrolysis of fused calcium chloride with the addition of such calcium salts as are capable of lowering the melting point of calcium chloride. Suitable calcium salts of this kind are calcium sulphate, iodide, bromide and fluoride. The inventors preferably use the fluoride. Pure calcium chloride melts at about 780° C., while the melting point of a mixture of 1 kg. of calcium chloride and 165 grams fluoride is about 655° C. The density of the latter molten mixture is stated to be 2.5 (that of pure calcium at 29.2° C. is given by Goodwin as 1.5446). The melting point of metallic calcium is stated to be 780° C. The inventors carry out the electrolysis at about 760° C., with a cathodic current density of 3 to 6 amps. per sq. mm. The negative electrode is made of iron (the lower part of which may be heated to red heat, but below bright white heat); the positive electrode is made of carbon or nickel. The two electrodes are separated by means of a partition of suitable material, for instance, iron, which is immersed into the molten mass.

Bipolar Electrode for Electrolytic Production of Bleaching Liquors.—R. Kother, 806,413, Dec. 5, 1905. Application filed Sept. 7, 1905.

The electrode, shown in Fig. 4 in vertical and horizontal cross-section, consists of a thin layer a of platinum foil (of a thickness as low as one-two hundreths of a millimeter), covering one side of an insulating plate b of glass or india rubber. This side acts as anode. The platinum foil is

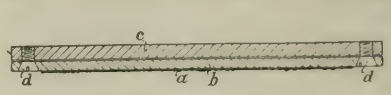


FIG. 4.—BIPOLAR ELECTRODE.

folded just round the edges of the insulating plate, and makes thus contact with the graphite plate c which acts as cathode. The bipolar electrode is held together by the screws d. Instead of platinum foil a thin network of platinum may be used.

Electroplating Apparatus.—L. Potthoff, 806,835, Dec. 12, 1905. Application filed Dec. 1, 1904.

The inventor describes details of construction of a tumbling barrel for electroplating small articles in quantities. The barrel is made of skeleton construction, consisting of annular end pieces connected by longitudinal bars, on which is stretched on the inside a lining of heavy cocoa matting. This is sufficiently strong and durable to support the weight of the articles in the barrel and to withstand the abrasion due to the work and the rotation of the barrel; this lining is also sufficiently porous to permit the electrolyte to pass through, so that the anode may be disposed both within and without the barrel, thus securing a practically unlimited amount of anode surface. In one side of the barrel a longtiudinal opening is provided, and

interiorly over this is mounted a shield or hood, which, when the barrel rotates in one direction, prevents the articles to be plated from passing out, but which becomes effective when the barrel is rotated in the opposite direction, so as to discharge the articles out through the opening.

Purifying, Sterilizing and Ageing Liquors.—R. C. Turner, 12,421, Dec. 12, 1905. Application filed April 21, 1903. July 28, 1905.

The inventor provides a reservoir containing the liquid to be purified and a series of funnels and purifying vessels. The liquid passes from the reservoir into the first funnel, from there into the first purifying vessel, from which it flows over into the second funnel, then to the second purifying vessel and so on. All the funnels are connected to one pole of the supply circuit, while each of the purifying vessels contains an electrode at the bottom, all these electrodes being connected to the other pole of the supply circuit. When the liquid flows from one funnel on to the electrode plate on the bottom of the purifying vessel below, the electric circuit is closed through the liquid, and the liquid is treated by electrolysis, and is simultaneously acted upon by the oxygen of the air while it flows downwards.

ELECTRIC DISCHARGES.

Treating Solid Materials.—K. Birkeland and S. Eyde, 802,620, Oct. 24, 1905. Application filed Sept. 1, 1904.

Pulverized solid materials are treated by the effect of an electric arc, which is made to fill the interior of a furnace in a disc shape in the same manner as was described at length on page 399 of our Vol. II., in the same inventor's process of winning nitric acid from the air. The present patent refers to the reduction of ores, whereby the ores are in a powdered state and mixed with powdered coal. This mixture is passed downward through a vertical furnace chamber consisting of two walls 1 and 2 in Fig. 5, placed near each other. These two walls enclose a narrow but long and lofty furnace chamber. Through the end walls (not shown) the adjustable electrodes (7 being the cross-section of one of them) reach into the said chamber, so as to leave between their points a short adjustable space lying in a line connecting the magnet poles 8 and 9 of the vertically-arranged electromagnet 10, 11 and 12. The pole pieces 8 and 9 are provided with a number of radially arranged iron projections or arms 13, 14, for spreading the magnetic lines of force as far as possible out from the middle of the furnace. The electromagnet 10, 11, 12, is of large dimensions, but the electrical energy required by it may be small; for a 30 to 40-kw. furnace the energy of the magnetizing current is 0.4 kw.; 15 is a feeding device, by aid of which the pulverized material is fed into the furnace in a thin, even stream. It passes along or through the arc spread in the furnace for the whole extension of the same, and hereby will eventually be wholly or partly evaporized. The electrodes may eventually be made hollow and used for introducing gases—for instance, hydrogen, illuminating gas or other reducti[illegible] particles having been treated, [illegible] while 17 is the exit tube [illegible] furnace.

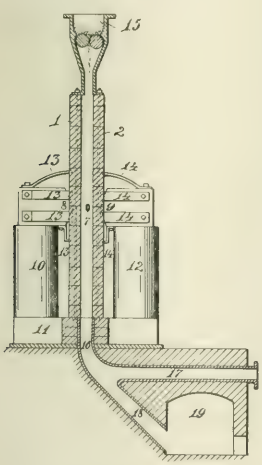


FIG. 5.—ARC DISCHARGE TREATMENT.

Production of Ozone.—A. C. Wood, 804,291, Nov. 14. Application filed Feb. 17.

Details of construction of an electrical apparat[illegible] production of ozone by electrical discharges through oxygen. It consists essentially of a tube [illegible] trode composed of sheet metal bent along its length intermediate of its edges to provide a plurality [illegible] and insulating supports for holding the electrode in the circuit.

STORAGE BATTERIES.

To explain the absence of this division in our last and in our present issues, it should be stated that no storage battery patents have been issued during the last months, the last number of this department being No. 8[illegible] Oct. 3, 1905, abstracted on page 433 of our Vol. III.

RECENT METALLURGICAL PATENTS.

NICKEL.

Separating Copper and Nickel.—Two patents have recently been granted to Mr. N. V. Hybinette, formerly with the Orford Copper Co., for the treatment of copper-nickel matte. The first patent (805,555, Nov. 28) relates to the treatment of "concentrated matte" (containing about 25 per cent sulphur and [illegible] to 5 per cent iron, the remainder being copper and nickel in differing proportions, generally 35 to 40 per cent of each metal) by the following three steps:

The first step is to grind and roast the matte. The roasted material is leached with sulphuric acid until the material will no longer neutralize the acid. Only dilute acid is used for leaching, so that the amount of iron [illegible] in the solution will at no time be more than 5 to 10 per cent. In this way it is possible to remove the bulk of the copper without dissolving much nickel; the solution will contain about one part of nickel to ten parts of copper. The solution so prepared is crystallized and boiled down, and again crystallized, and at each repeated crystallization crystals of commercially pure copper sulphate are obtained. The mother liquor contains copper and nickel in about equal parts; it is concentrated and boiled down to a solid mass of copper-nickel sulphates.

The material that has been leached contains about 55 to 60 per cent Ni and 12 to 18 per cent Cu, and is subjected to the second step of the process. It is mixed with sulphuric acid of about 60 per cent free acid, in such quantity that there will be sufficient acid to form sulphate of copper with all the copper present. The mass is then slowly brought up to a low red heat, and roasted at that temperature for a short time. Sulphates of copper and nickel are formed, but preferably the former. When the material reaches a temperature of about 500° C., the sulphates begin to decompose, but the nickel and copper sulphates are not affected in the same way. After completed [illegible] the material is drawn from the furnace and leached [illegible] weak sulphuric acid, whereby again a solution [illegible] about ten parts of Cu to one of Ni is obtained, which [illegible] crystallization, yields commercially pure copper sulphate. The mother liquor is boiled down, giving [illegible] equal parts of copper and nickel. This [illegible] phuric acid and leaching is repeated. The [illegible] will then contain about 70 per cent Ni and 3 to 5 per cent Cu.

The third step is to mix the [illegible] acid or a mixture of common [illegible] to heat again to a low red heat. The [illegible]

continued until a maximum of copper and a maximum of nickel are present in form of chlorides. The material is then leached with water and weak acid, whereby a residue of practically pure nickel oxide is obtained with about ½ per cent copper still present. The nickel oxide is melted to metallic nickel anodes, which are electrolytically refined. This last leaching of chlorides leaves a solution from which no pure salts can be separated.

To sum up again, the first step yields copper sulphate crystals and a residue containing 55 to 60 per cent Ni and 12 to 18 per cent Cu. The second step treats this residue, and yields a residue containing 70 per cent Ni and 3 to 5 per cent Cu. This residue, when treated in the third step, yields metallic nickel.

The mother liquors, left in all three steps, and consisting of mixed copper and nickel salts in solution, are heated to a strong red heat, whereby all the nickel sulphate and chloride are changed into oxide. This residue is leached with water and weak sulphuric acid, whereby copper sulphate is again obtained and the residue of the leaching is nickel oxide.

The essential novel features are the second step and the combination of the three steps in one process. There are many electrolytic copper refineries which treat nickel-bearing copper. These plants make as a by-product a mixed sulphate of about equal portions of nickel and copper. This may be handled as the mother liquor from crystallization of copper.

Electrolytic Process for Separating Nickel and Copper.—The second patent of Mr. N. V. Hybinette (805,969, Nov. 28) refers to an interesting electrolytic process, the appa-

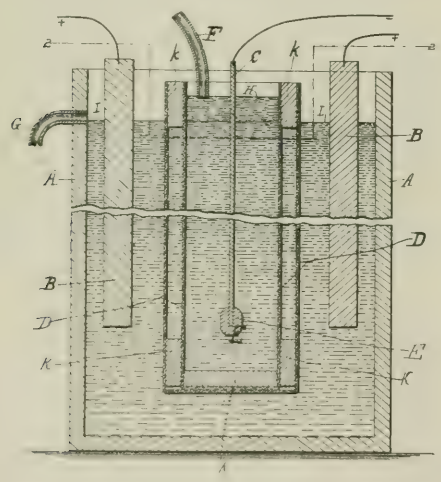


FIG. 1.—ELECTROLYTIC CELL FOR SEPARATING COPPER AND NICKEL.

ratus being shown in Fig. 1. A is the tank, C the cathode, and B the anodes. The cathode is surrounded by a porous diaphragm D, constructed of a wooden frame K and two thicknesses of cotton cloth D, at a distance of about ¾ inch from each other. F is the inflow for the electrolyte, G the overflow; they are so adjusted that the level H of the electrolyte in the cathode compartment inside the porous bag is about 1 inch above the level L in the anode compartment. The electrolyte is thus caused to flow in a regular stream from the cathode to the anode, in a direction perpendicular to both plates. The cotton cloth D is very porous, and the diaphragm bag is, in other words, only a convenient mechanical device for directing the circulation of the liquid in a direction from cathode to anode.

The cathode starting sheet C is made of copper (greased so that the nickel deposit may be easily removed), and is prevented from warping by the wooden rip E. The anode plates are cast from the nickel-copper alloy which is to be separated; they should contain less than 1 per cent of sulphur and less than 3 or 4 per cent of iron. The electrolyte consists of a dilute solution of pure sulphate of nickel with a small proportion of a weak acid—such as phosphoric acid, boric acid, etc. During the operation nickel is deposited on the cathode, while an equivalent amount of both nickel and copper, with some iron, goes into the solution from the anode. Thus the solution in the anode compartment becomes gradually richer in copper and iron, but it is prevented from passing over into the cathode compartment by the flow of the liquid through the diaphragm in opposite direction. The solution in the cathode compartment is relied upon to remain pure nickel sulphate. After the porous bag has been in use for some time the pores become filled up with slimes of boric salts. Any platinum, palladium, gold, etc., contained in the anode, together with considerable copper, especially copper sulphide, pass into the anode slime. The anodic solution of mixed nickel and copper sulphates passes continuously off through G, and is regenerated outside of the cell as follows:

The copper is removed from the solution by cementation upon nickel or a nickel-copper alloy, which must be free from sulphur, silicon and carbon. Under these conditions the cementation is quick, if the solution is kept at a boiling temperature. The solution flowing off from G is, therefore, first passed through a tank, in which are suspended slabs of a nickel and copper alloy, containing, say, 30 per cent of copper, and then through a second tank containing pure nickel slabs. Any copper in the solution is thereby cemented out. To remove the iron, anodic oxidation (with insoluble anodes) and precipitation with nickel carbonate are employed. All the copper and iron are thus removed from the solution, while the nickel content is simultaneously raised again to its original value. The solution is now ready to be supplied again through F to the electrolytic cell. The electrolyte thus passes through a complete cyclic process.

Zinc.

Zinc and Lead from Complex Sulphide Ores.—A process of A. H. Imbert (807,271, Dec. 12) is based on the property of metallic copper to act as a very energetic desulphurizer toward most of the ordinary metals. In particular, if granulated copper be mixed with crushed zinc sulphide, or with galena in the proper proportions, the reactions will be with galena at 800° C.:

$$PbS + 2Cu = Cu_2S + Pb,$$

and with zinc blende at 1,100°:

$$ZnS + 2Cu = Cu_2S + Zn,$$

the zinc escaping from the molten mass under violent ebullition and being condensed outside. With a complex sulphide ore the same two reactions as above will take place successively, but at temperatures varying in accordance with the predominence of one sulphide or the other. If care be taken to protect the lead which accumulates at the bottom of the crucible from exposure to the greater temperature necessary for vaporizing the zinc, the contamination of the zinc vapors with those of the boiling lead is prevented. Two forms of apparatus are described. The copper sulphide is reduced again to copper by the Bessemer process, and the copper is used again with new zinc and lead sulphides.

Zinc Furnace.—E. Dor Delattre (806,121, Dec. 5) patents details of construction of regenerative gas furnaces, to obtain a constant and regular temperature in every part, and

to prevent the fouling of the openings for the entrance of the air and gas. The principal feature is the arrangement of the openings for the entrance of the air and gas, so as to be easily accessible and adjustable. The burner gaps are provided with movable blocks or slabs, whereby the air and gas currents can be altered and regulated at the moment of entering the furnace. The openings which connect the two furnace chambers are so shaped that they compel the proper mixing of the air and the part of the gas which has not been burned in the first chamber, and to insure complete combustion of the whole of the gas.

Zinc, Gold, Silver from Ores.—J. Nicholas (805,577, Nov. 28) treats ores, matte, slimes, etc., containing zinc, cadmium, lead and precious metals as follows: The material is pulverized, mixed with water to a pasty condition and sodium chloride, and is then heated in an oven under exclusion of air, whereby the zinc and cadmium compounds are changed into chlorides. These are leached out, the cadmium is precipitated by sulphuretted hydrogen, and the zinc by milk of lime. The residue of the leached mass contains lead, silver and gold, is fluxed in a furnace, and the metal so obtained is remelted, and when in a molten condition aluminium is introduced into it and the whole kept in agitation until the aluminium melts and becomes alloyed with the gold and silver. On cooling, the aluminium separates from the lead and rises as a crust to the surface, carrying with it the gold and silver.

BOOK REVIEWS.

High Temperature Measurements. By H. Le Chatelier and O. Boudouard. Translated by G. K. Burgess. New York: John Wiley & Sons. Price $3.00.

The second edition of Dr. Burgess' translation of "High Temperature Measurements" is of especial interest to those engaged in electrometallurgical work, because of the additional work done since the appearance of the first edition of the book. In his preface to the second edition Dr. Burgess says: "The greatest advances have been made in optical pyrometry, and the chapter on this subject has been greatly extended and preceded by one on the laws of radiation."

In dealing with the highest temperatures, such as those used in the electric furnace, it is not possible to use thermo-electric or resistance pyrometers, the temperatures reached being altogether too high. Thus it is to the optical pyrometer that we must turn when it is desired to study electric furnace temperatures, and there can be no doubt that when the problems involved in the application of optical pyrometry are solved, great improvements in electric furnace practice will be effected.

It might be thought that a serious difficulty that will be experienced in the application of the optical pyrometer lies in the formation of vapors that will obstruct a view of the interior of the furnace. Suppose, for example, that a tube furnace is in use, and that the heating is obtained by passing a current through the tube, which is made of carbon. As the temperature rises the tube will rapidly become filled with dense vapors, probably due to the vaporization of impurities, such as silica and iron in the carbon, and these will certainly cut off a large amount of light. Even if the carbon composing the tube is pure, vapors will be obtained from the surrounding material, whatever that may be, since carbon becomes very porous to vapors when at a high temperature. It would be interesting to get a reply to this objection from optical pyrometer experts.

As a concise statement of what has already been accomplished in the methods of measuring high temperatures, the book will be found useful by all metallurgical engineers who wish to get an exact record of the course of temperature during all stages of operation.

Monographien [illegible] XVII. Halle a. S.: W. Knapp.

During the past year our German cousins have been indefatigably turning out the monographs on applied electrochemistry, from the press of Wilhelm Knapp in Halle.

Among the works which have recently appeared we note the following:

(Vol. XI.). Die Galvanoplastik. By Dr. W. Pfanhauser. 130 pages. Price 4 marks.

This is a very practical and timely work, written by a technician well posted on the subject. He gives extensive details of the forming of the moulds, making the conducting, composition of the depositing baths, [illegible] and production of various specialties, such as cliches, grammophone plates, engravings, etc. It is a skillful presentation of the subject brought up to date.

(Vol. XII.). Die Elektrochemische Industrie Deutschlands. By P. Ferchland. 64 pages. Price 2.50 marks.

In the narrow limits of sixty-four pages one could hardly expect a very thorough presentation of this subject, but, nevertheless, one is entitled to expect more than the author has gotten into them. The treatment of the subject is unsatisfactory, and the monograph a disappointment.

(Vol. XIII.). Carborundum. By F. A. J. FitzGerald. 44 pages. Price 2 marks.

The author has been intimately connected with the carborundum industry, and his work is eminently satisfactory, not only from the standpoint of the description of the details of manufacture of this distinctively electrochemical product, but also satisfactory from the standpoint of general information concerning the erection and running of large incandescent electric furnaces.

(Vol. XIV.). Elektrolytisches Verfahren zur Herstellung parabolischer Spiegel. By Sherard Cowper-Coles. 30 pages. Price 1 mark.

A short paper, which should have appeared as a contribution to some journal or scientific society. The author describes only his own process, and undoubtedly the publication of this fragment as one of the "Monographs" has lowered the dignity of the series.

(Vol. XV.). Artificial Graphite. By F. A. J. FitzGerald. 60 pages. Price 3 marks.

This is one of the strongest monographs of the series, in that it describes in detail, and at first hand, the actual manufacture and properties of this interesting product. Some foreign scientists contest very vigorously the theory of its production, as explained by Mr. Acheson, the inventor, and described in detail by Mr. FitzGerald, but a few years will settle that dispute, leaving untouched the practical details so closely and minutely described.

(Vol. XVI.). Die Darstellung des Zinks auf elektrolytischem Wege. By Dr. Ing. Emil Guenther. 224 pages. Price 10 marks.

This work is a welcome contribution to [illegible]. The author was a student under Dr. Borchers, and had the benefit of the advice of this redoubtable electrometallurgist, as well as having in part of Germany where the electrometallurgy of zinc first took root. The processes are very systematically grouped into the treatment of aqueous solutions and of fused salts, and the former systematically subdivided. It is to be regretted, however, that the author did not include the electric furnace processes within the scope of his [illegible].

(Vol. XVII.). Hydrometrie und Elektrische Energie. Theoretical part. By Dr. Emil Voit. 118 pages. Price 4.50 marks.

While this monograph is a very interesting and readable one, and contains some information concerning [illegible] and

wherefore which will be of value to practical electrochemists (together with a great deal of theoretical speculation which will never be of use to anyone), yet its appearance in this series of monographs on "*Applied* Electrochemistry" is an anomaly, and gives the impression of padding the series. Our advice would be to the publishers to put such works as this into a new series of monographs on "Theoretical Electrochemistry."

California Water-Power Plant.

Our most up-to-date metallurgical plants are now covered with networks of electrical circuits, supplying electrical energy for lighting, traction, and especially for all possible power purposes. This is in accordance with the prevailing tendency in our industries of pushing the output of a plant to the maximum limit by substituting machines and automatic devices for human labor, wherever possible; and there is no energy form so flexible and transportable as electrical energy. Naturally, electric power is now also seriously entering the mining field, but while the progress is steady, yet it is relatively slow. The principal problem is here the power problem: how to generate the electrical energy in the cheapest way.

This problem has repeatedly been discussed in these columns, and it has been pointed out that there is not one and the same solution of the problem in all locations. According to the local conditions, water power, gas power, or steam power may be preferable. For mining purposes, water powers are often available and easy to develop. In this respect a new water-power plant at the Pacific Coast—that of the North Mountain Power Co.—contains much that is of interest. This plant has really been erected rather with an eye to the future than for the present need of the communities which it serves.

This plant is located in the central part of Trinity County, California, 2 miles below the town of Junction City, where Canon Creek, from which the water used for power is obtained, flows into the Trinity River.

Weaverville, the county seat, is 12 miles distant by road. The nearest railroad point is Redding, on the "Shasta Route" of the Southern Pacific Railroad; Humboldt Bay, on the Pacific Ocean, with Eureka, the chief coast city of Northern California, lies almost due west, distant 50 miles in a straight line.

The altitude of the plant is about 1,400 feet. All material, cement and machinery were hauled in over 60 miles of the severest mountain roads, across three distinct divides or summits, viz.: Trinity Mt., Brown's Mt., and Oregon Mt. It required eighteen to twenty animals to pull each of the larger pieces, weighing 18,000 pounds up the grades, and when mud was encountered it was necessary to hitch eighteen animals to the fall of a block and tackle fitted with steel cables. Despite these difficulties, however, no mishap occurred to any of the machinery.

Watershed

The water used at the plant is diverted from Canon Creek, which has a drainage area of 52 square miles above the diverting dam. The upper part of the basin is a rugged, glaciated granite country, extending up to an altitude of from 9,000 to 10,000 feet above sea level, where some snow remains throughout the year. The higher parts of this basin are in the nature of an immense amphitheater, lying on a flank of Thompson Peak, which is a characteristic feature of all elevated glaciated topography. And, as is so often the case, the converging and descending glaciers have scooped out immense holes with almost perpendicular sides, which now form the beds of two lakes. These have been utilized as storage reservoirs.

Ditch System

The dam is small, and serves merely for diverting the water. It is of the usual rock-filled crib form, which has been the type used for many years in mining operations on the Pacific Coast.

The ditch system consists of alternating sections of ditch, flumes, and one tunnel. Part of the ditch is cut in solid rock,

Fig. 1.—18-Horse team hauling the lower half of stator of a 750-kw. alternator over 60 miles of mountain road.

but the most of it is dug in the side-hill soil which stands well and puddles well.

The flumes are nineteen in number, and vary in length from 30 to 1,200 feet. Those immediately below the dam are laid upon a solid bed-rock bench, and equipped with adequate spillways, waste-gates and sand-boxes. The total length of the flumes is 5,250 feet.

A tunnel 1,821 feet long has been driven to replace a temporary flume which was unavoidably built upon treacherous sliding ground. This tunnel is heavily timbered throughout its entire length, and tightly lagged except where the rock is unsusally hard. The dimensions of the main tunnel are 4 feet 9 inches by 5 feet 10 inches in the clear. The total length of the ditch, flumes and tunnel is 7¼ miles.

The average grade of all is 0.00184 or 9.73 feet per mile.

The Penstocks.

The penstocks, two in number, are each 1,105 feet long. Under a total head of 604 feet there is an effective head of 600 feet, or working pressure of 260 pounds per square inch.

There is one complete independent penstock for each of the two units.

Buildings

The plant proper consists of (1) the power house, (2) the transformer house, and (3) the high tension switch house. In addition there are (4) the shop, (5) the operator's dwelling, and (6) the stable. At the forebay is the forebay tender's cabin, and at the dam is a house for the tender of the head gates.

Fig. 2.—Trinity River plant of North Mountain Power Co.

Power House

Each of the two hydraulic units consists of a pair of 14 inch Pelton wheels under one sheet steel housing, provided with ring oiling, self aligning bearings, coupled to the generator

through flexible leather link couplings. The nozzles are of the deflecting type. With the largest tips in service the wheels are capable of driving the generators at 25 per cent overload.

The wheels are controlled by type "I" Lombard governors using oil. The pressure and vacuum are maintained by oil pumps belt-driven from the wheel shafts.

The governors are not fitted with any form of switchboard speed control, but a single operator has no difficulty in synchronizing a generator under the load conditions which exist in the plant.

FIG. 3.—OIL-FILLED, WATER-COOLED, 300-K. V. A. TRANSFORMERS IN TRANSFORMER HOUSE.

The tail-race is 6½ feet wide, and excavated for 280 feet through bed rock to the Trinity River.

The generators, two in number, are of the Bullock type, furnished by Allis-Chalmers Co., of Milwaukee, being three-phase, 750 kw., 500 r. p. m., 2,200 volts, 25 cycles, rev. field, 6-pole.

Two Bullock exciters 125 volts, 45 kw., 900 r. p. m., are driven by belts from the generators. The switchboards are of marble and have Wagner instruments.

TRANSFORMER HOUSE.

The transformer house, like the power house, is built of concrete. It contains seven step-up transformers, viz.: two banks of three each, and one in reserve. They are of Bullock make, 300 kilovolt-amperes, water-cooled, oil-insulated, 2200/15050 volts, 25-cycle. Each bank is located in a separate room, and the seventh, or spare transformer is in a third room. They stand upon rollers, which in turn rest upon T-rails set into concrete piers. The spaces between the piers are filled with gravel and provided with sub-drainage so as to allow any oil to rapidly escape in case of leakage or accident. The water piping is connected up with unions, and so arranged that every transformer is interchangeable with every other one, as far as water and electric connections are concerned.

A track runs the entire length of the building, and at such an elevation that the floor of a car is at the same height as the rails which are imbedded in the concrete piers. Thus any transformer can be quickly rolled out upon the car, removed, and another reconnected in its place in case of accident.

The penstocks are each tapped on the power and after passing through cut-off and throttling valves under the control of the operator, water is delivered into a cylindrical steel tank, set upon a little knoll about 50 feet above the level of the transformers. From this tank the water circulates through the cooling coils of the transformers by gravity.

The transformers are connected in delta on the low-voltage side, and in Y on the high-voltage side, and in Y on the [illegible] neutral.

THE HIGH-TENSION SWITCHBOARD.

In the high-tension switchboard are [illegible] single-throw air-brake switches, and [illegible] rent multiplex lightning arresters connected to [illegible] phase circuit.

POLE LINE.

The pole line extends almost due west from the power house to the sub-station in Eureka. The length of [illegible] 55 miles are over a [illegible] rugged mountainous country. The altitude of the plant is only 1,480 feet, and Eureka [illegible] level, but the line passes over several summits ranging from 4,500 to 5,500 feet in altitude. Fifty miles of its length lie in a heavily timbered country, requiring a tremendous amount of clearing, the trees ranging from 2 feet [illegible] in diameter. It was necessary to construct a trail nearly the entire length of the line.

The route deviates from a straight line only slightly and only where the topography made it unavoidable.

It is a "thorough" transmission, so to speak, there being no taps on the line anywhere between the plant and the sub-station at Eureka.

The line is a single circuit, three-phase, averaging thirty-five poles to the mile.

The poles are redwood at the western end; in the inaccessible stretches the native red fir was the only wood available.

Insulators are Locke two-piece single petticoat porcelain, 8½ inches in diameter. In the fog-belt a larger insulator of similar design is used.

Pins are sun-dried eucaluptus, oiled, 1½ in diameter.

The spread of wires on the standard length spans is 4 inches; wire is No. 4 copper, except on some long spans where stranded cable is used.

The length of spans varies from 50 to 1,400 feet, according to the conditions and topography of the country.

The potential at present used on line is 30,000 volts.

SUB-STATION AT EUREKA.

The sub-station at Eureka includes an auxiliary steam plant as follows:

Two (2) B. & W. water-tube boilers fitted with Peabody patent oil burning furnaces duplicate oil pumping system;

FIG. 4.—VIEW IN POWER HOUSE, SHOWING WATER-WHEEL [illegible] AND SWITCHGEAR.

Gilbert [illegible] feedwater heater. Leavy Wheeler [illegible] surface condenser with [illegible] circulating pumps, cooling water [illegible] Bay, and a McIntosh & [illegible] of manual [illegible] running at 500 r. p. m. is connected to eng[illegible]

A Bullock rotary converter, 500 kw., 500 r. p. m., 6-pole, 25-cycle, 550-volt, is arranged for direct connection to this jackshaft by a jaw clutch and so driven by the engine. This permits of carrying the load by steam when necessary to shut down the transmission line for repairs.

The engine is fitted with a switchboard speed-control device. The clutch has a synchronism indicator in the nature of a lamp, so that the engine may be connected to the rotary while it is running at full speed on the power transmitted from the Trinity River plant.

For the rotary converter there are three Bullock transformers, 190 kw., water-cooled, 25-cycle, 30,000/352 volts. For stepping down for the local distributing system are three G. E. transformers, 400 kw., water-cooled.

For furnishing power to the 60-cycle incandescent and arc

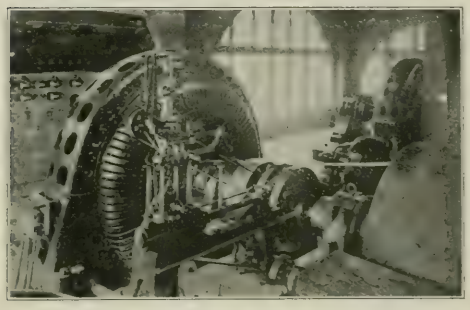


FIG. 5.—750-KW. ALTERNATOR AND WATER-WHEEL GOVERNOR.

lighting circuits of the city of Eureka, a three-phase, 60-cycle generator is driven by the rotary, acting as a synchronous motor.

The sub-station is fitted with switch boards and h. t. switches suitable for handling this equipment.

A fuel-oil tank 54 feet in diameter by 25 feet deep, holding 10,000 barrels, has been built near the substation, and is connected to a dock on Humboldt Bay by a pipe line.

The load at present consists chiefly of lights in the city of Eureka. Some motors are already connected to the circuits, and the motor load is being rapidly developed.

Vaporization of Jacket Water.

The amount of water needed for cooling water jackets in the usual way is quite considerable, and in places such as Leadville, Denver and other towns, where some of the smelting companies buy the cooling water from the city water companies, the expense for this apparently small item is very high. In such cases a considerable saving may be accomplished by the system of vaporization of jacket water, patented by the Colorado Iron Works Co. This system is discussed at length in the book recently issued by this company, entitled "Some Details as to Smelting Practice and Equipments."

The method is based on the well known fact that water boiling at 212° F., when changed into steam of the same temperature, absorbs a certain amount of heat from the surroundings. This heat represents the energy required for the change from the liquid to the gaseous state, and because it disappears with the formation of the steam it is spoken of as the latent heat of steam. The change of 1 pound of water at 212° F. to steam at the same temperature absorbs 966 B. T. U.

If the jacket water is simply heated to a higher temperature and discharged as water, the heat absorbed is simply the so-called specific heat. If it is discharged as steam, the heat absorbed is the sum of specific heat and latent heat. Let us assume that the water is supplied to the jackets at 62° and discharged as water at, or just below the boiling point, 212°, then the water is raised in temperature 150° F., and since 1 pound of water needs 1 B. T. U. to be raised by 1° F., every pound of cooling water absorbs under those circumstances 150 B. T. U. If the water is not discharged as liquid water of 212°, but as steam of the same temperature, each pound absorbs not only those 150 B. T. U. but 966 additional heat units (latent heat), that is, in the whole, 1,116 B. T. U. In other words, the heat absorbed, or the cooling effect, is in the latter case seven and one-half times greater if steam is discharged instead of water. Or to produce a certain cooling effect, seven and one-half times less water is required. Of every 7½ pounds of water used when water is discharged, 6½ pounds thus appear to be wasted. This explains why, under a great many conditions of practice, a very considerable saving may be accomplished by means of vaporization of jacket water.

Some slight change from the usual forms has to be made in the necks and backs of the jackets, as rounding corners and enlarging outlets, to avoid spasmodic ebullition and adapt them to the conditions involved.

Steam may be discharged directly from the several water-jacket sections, but a better method is to connect all the sections by means of pipes to a horizontal drum on each side of the furnace. The jackets and pipes are kept full of water. The level is maintained somewhere near the centers of the drums, and the separation of steam takes place in the drums and escape pipe leading the steam away to the open.

The vaporizing system prevents deposit of lime and other impurities on the inside of the jackets to some extent, though not wholly. There is by far less water used in this system, and to that extent there is less deposit of sediment in the jackets. Water is not vaporized from the jackets but from the drums or tanks to which the jackets are connected. Hot water in the jackets rises to the top and thence up through the dis-

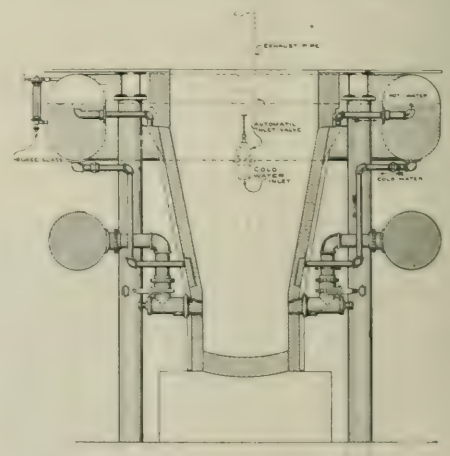


FIG. 1.—VAPORIZATION OF JACKET-WATER.

charge pipes to the surface of the hot water in the drum, where it bursts into steam. Hot water cannot go to steam below when the way is open for it to rise, but steams at the surface. As all the steaming is done at the surface of the water in the drums so all, or most, of the combined impurities in the water are deposited in the drums, which are very easily cleaned out and cannot be damaged by such deposits.

This system is perfectly adapted to vaporizing, as also to overflowing jacket water, no change whatever being necessary other than to open a valve in the apparatus when changing over from the one to the other. As the water supply runs

short if overflowing, vaporization at once begins automatically and continues to compensate for whatever shortage there may be in overflow supply until the supply dwindles away down to one-eighth the ordinary supply necessary for overflow when discharging boiling water, and at that point all is evaporated and the supply must not run lower. There is no disadvantage whatever in evaporating the whole necessary, while there is much advantage in the system, as pointed out above.

It is sometimes the practice among smelter people to run so much water through the jackets that it is discharged at a temperature much below boiling. This is done under the impression that the jackets are more durable when kept at the lower temperature. This, however, is a misapprehension, for as a matter of fact cast iron as well as wrought iron has greater strength and a greater elastic limit at 400° F. than it has below that temperature. The heat of jackets will not be variable when vaporizing, or in any case where this system is used, as it is by the common method of discharging water when the supply turned on them is varied.

There are more cast-iron jackets broken and more plate-steel jackets damaged by running an excess of cold water

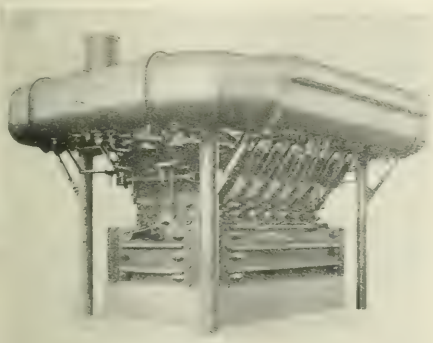


FIG. 2.—LEAD FURNACE WITH VAPORIZER EQUIPMENT.

through them than in any other way. The hotter jackets can be kept up to 400° F. the more durable they will be.

The construction is shown in Fig. 1. Water enters the vaporizer at the bottom of the back end section of the tank, through the automatic inlet valve, which admits only just enough water to take the place of that passing off as steam. Its action is constant, it has no wearing parts, and hence is everlasting.

In Fig. 1 this end section of the tank, with the inlet valve and its chamber and float and the exhaust pipe, are shown in dotted lines, all being at the opposite end of the furnace from the observer and seen through, as it were.

There is no possible contingency to prevent its working, and it always works just as long as the float floats, and it carries the valve with it, admitting more or less water to maintain the level constant, in the tank.

No water overflows from the tank or drum, or is lost in any way, except just what is vaporized and passes off through the exhaust pipe as steam.

From the bottom of the tank water passes down through pipes into each jacket respectively.

Two of these pipe connections, one on each side of the furnace, are shown in the cut, extending from the bottom of the tank downward and into the jackets above the tuyeres. Each jacket has its independent down-pipes from the bottom of the tank, in like manner as those shown.

The water entering the bottom of the tank cold, so remains cold at the bottom, only rising as that at the surface vaporizes off and is continuously supplemented from below with cold water. The cold water entering the jackets by the supply pipes [illegible] and rises in the jackets, through suitable pipes [illegible] to the steam drum, where it [illegible] and passes out to the atmosphere through the exhaust pipe.

Fig. 2 shows a lead furnace equipped with this vaporizer appliance.

Portable Moving-Coil Galvanometer.

Patents have recently been granted to the Leeds & Northrup Co., of Philadelphia, for a portable moving-coil galvanometer. The distinguishing novel feature is the method of suspending and winding the moving system.

The suspension consists of upper and lower filaments, and in this particular it differs from the pivot and jewel as ordinarily used in instruments of the portable type. These filaments are fully protected against breakage in a manner to be described later. This arrangement of the filament suspension permits high sensibility and avoids the initial friction to be overcome when jewel bearings are adopted as a means of suspension. This is of importance, especially when the galvanometer is used for zero methods, and by reason of this fact it can be substituted for instruments of the mirror type in potentiometer work for accuracies of 1-5 per cent and in all Wheatstone and slide-wire bridge measurements to commercial accuracies. Its high sensibility makes it available for use in many cases where it was ordinarily supposed that it is necessary to use a non-portable suspended coil galvanometer. The entire construction of the instrument is one of extreme simplicity and compactness, while at the same time preserving the sensitiveness and general efficiency of other forms which it has hitherto been considered impracticable to make so simple and compact.

The magnetic field is produced by a U-shaped cast-iron permanent magnet, Figs. 2 and 3, having inwardly directed poles at one end, so as to have north polarity only on one side of the air gap, separating the poles, and south polarity only on the other side of the air gap. The magnet is slotted so as to accommodate the moving system as shown. The air gap between the poles being very short the field is uniform and intense.

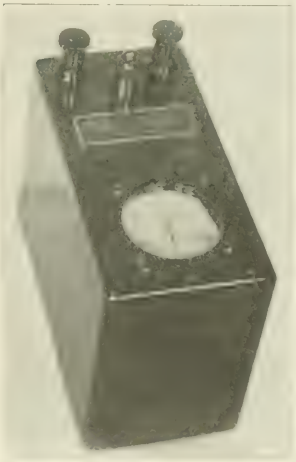


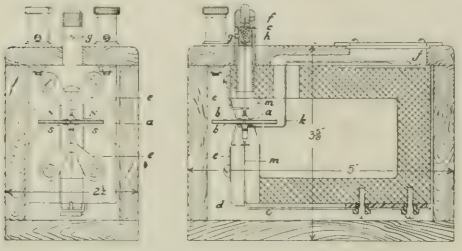
FIG. 3.—THE PORTABLE GALVANOMETER.

The moving system consists of a coil of fine wire wound alternately in opposite directions, so as to produce alternate north and south poles when viewed from one side, as shown in diagram in Fig. 4.

[illegible] manner that normally [illegible] coil extends [illegible] are equidistant [illegible] about one-half of the coil surface is [illegible] the coil, as Fig. 4 [illegible]

swing around to inclose the greatest number of lines of force, while the other two coils of like polarity will be repelled. The aluminium discs b also serve the purpose of damping the system and rendering it dead-beat. The galvanometer can be given a high, medium or low resistance by connecting the coils in series, series-parallel or parallel.

The suspensions consist of three wire filaments of a high



FIGS. 2 AND 3.—DIAGRAMS OF CONSTRUCTION.

tensile strength, which are non-corrosive. To give the entire system resiliency in the event of sudden jars, the upper suspension terminates in a spiral spring c, and the lower suspension is attached to a long, flat phosphor-bronze spring d, as shown in Fig. 3. An additional protection against excessive strains on the moving system is provided by means of two guards e. These guards have clearance holes to provide for the rotation of the system around its axis of suspension, but to limit the sidewise movement or play of the system, thereby arresting the movement of the coil due to any sudden shock. The movement of the system vertically is limited by the pole faces of the magnet, which also prevents breaking of the wire filaments.

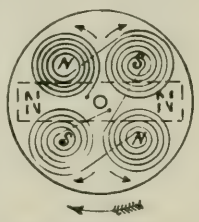


FIG. 4.—MOVING SYSTEM.

The torsion head f provides a delicate zero adjustment, and by means of the screw g, operating in an annular groove h, engages the end of the groove and thus acts as a stop to limit the extent to which the torsion head may be turned, and therefore prevents the suspension filament from being twisted off by turning the adjusting head too far.

The magnet with its complete moving system and scale is mounted in a highly polished mahogany case, as shown in Fig. 1. A circular opening in the top, protected by a glass window, provides for the visibility of the scale. The binding posts are securely mounted as indicated, and are arranged to clamp large or small wires. The outside dimensions of the case are 5 inches x 2½ inches x 3½ inches high.

Industrial Notes.

Siloxicon Brick.—In our last volume, page 445, we referred to the formation of the Acheson Siloxicon Articles Co. According to a recent report in "The Cataract Journal," of Niagara Falls, the new plant of this company, on Buffalo Avenue, near the city, is now nearing completion. The Niagara Falls Power Co. has already obtained permission from the Common Council to string the new line to supply power to the Acheson Siloxicon Articles Co. Mr. E. G. Acheson expects to inaugurate at once "experiments at the new plant in the making of brick by the new process. Just how many hands will be employed is conjectural at this time. Early in spring the plant will be opened full-blast, and will give employment to a large number." Concerning the manufacture and the properties of siloxicon see our Vol. I., pages 287 and 373, and Vol. II., page 442.

Carbon Exhibit.—The National Carbon Co., of Cleveland, Ohio, had an interesting exhibit at the recent Electrical Show in New York, covering the complete line of primary batteries and carbon products. The new Columbia igniter cell for automobiles, the Stand By, Globe, Laclede, New Century, Hercules, Solar, Ravenna and Columbia signal dry cells, and the Trogan and National Fuller wet cells were on exhibition, as well as flash lights, carbon brushes, arc-light carbons, carbon electrodes, specialties, etc. The exhibit was in charge of Mr. A. E. Carrier and Mr. A. C. Henry.

Balances.—We have received from Messrs. Voland & Sons, New Rochelle, N. Y., their illustrated catalogue E on balances and weights of precision. The catalogue gives illustrated descriptions of assay balances, analytical balances, bullion balances, balances for scientific use, diamond balances, jewelers' balances, prescription balances, pulp balances, riders and all different kinds of weights. The new factory of Messrs. Voland & Sons, in New Rochelle, has been equipped in the best and modern way.

Bulletins No. 1046 and 1047 of the **Allis-Chalmers Co.**, electrical department, contain illustrated descriptions of Bullock multipolar motors and generators and of Bullock oil-insulated transformers.

Thermit.—Mr. E. Stütz' recent Franklin Institute paper on thermit practice in America, which was published in abstract on page 430 of our last issue, has now appeared in full in the December issue of the "Journal of the Franklin Institute," and has also been issued in nicely illustrated pamphlet form by the Goldschmidt Thermit Co., of New York.

Electric Ovens.—A recent invention of Prof. Morse and Dr. Frazer, of Johns Hopkins University—a graphite electric stove—furnishes an excellent heater for drying and constant-temperature ovens and baths. A particularly valuable feature of their method is that the graphite mixture may be applied to any desired shape and adapted to any safe voltage, and so the necessary heat for the oven is generated and used efficiently. Prof. Morse finds that several electric ovens in nearly continuous use in his laboratory are operated at a cost actually less than the same would cost with gas heating. The International Instrument Co., of Cambridge, Mass., manufacture them.

Crucible Melting Furnace.—The Monarch Engineering & Mfg. Co., of Baltimore, Md., have lately increased their plant facilities, in order to be able to fill promptly the rapidly increasing orders from foundrymen in this country and abroad for the Steele-Harvey crucible melting furnace, using fuel oil and air by either compressor or low-pressure blower. The crucible is not removed in the pouring of the metal, thereby extending the life of the same. There are no ashes or loss of metal in the pit. The furnace is specially used for melting aluminium, platinum, phosphor bronze, copper, tin and brass foundry mixtures, and strongly appeals to the brass foundryman, since the old style crucible is still used, but under modern methods. The capacities range from 120 to 1,200 pounds per heat.

Electric Traction in Mines.—The Westinghouse Electric & Mfg. Co. are doing a large business in equipping mines with electric locomotives, to replace the older forms of haulage, whether animal or mechanical. Electric mine haulage, considered from either the points of efficiency or economy, has so many advantages, as compared to the older practice, that the time is not far distant when any

other method of mine haulage will be the exception. A recent contract closed by the Westinghouse Co. is one with the Newport Mining Co., who have decided to equip their mines, at Ironton, Mich., with both surface and underground electric haulage. They will use electric locomotives the year around in the various levels underground for bringing the ore to the bottom of the shaft, and after the transportation season has closed will use electric locomotives on the surface for hauling ore from the top of the shaft to the various stock piles for storage. For these purposes they have ordered six 4-ton Westinghouse mine locomotives. Electrical apparatus for the equipment of the necessary power station will also be provided by the Westinghouse Co., consisting of a 150-kw., 250-volt generator, direct connected to a Corliss engine of 130 r. p. m., and a three-panel switchboard, besides other auxiliary apparatus.

Steam Turbines.—The success of the steam turbine makes it clear that with its aid steam power will be able to hold its own satisfactorily in many localities in competition with gas and water-power. The recent starting up of a steam turbine at the Washington Street power house of the Utica Gas & Electric Co., in Utica, N. Y., calls attention to the fact that this is the first turbine to be put into operation by the Allis-Chalmers Co., who have recently entered the steam turbine field. In doing so, the Allis-Chalmers Co. has effected an alliance with the Turbine Advisory Syndicate, of England, has made more recently an agreement which Hon. Charles A. Parsons for interchange of data, thereby giving to the Allis-Chalmers Co. the benefit of the vast experience of Mr. Parsons, the original inventor of this type of turbine, and to whose engineering ability and indomitable energy the evolution and present state of perfection of the successful steam turbine are principally due. We will publish a description of the Allis-Chalmers steam turbines in one of our next issues.

Steam Turbines as Reserve Power in Water Transmission Systems.—The Missouri River Power Transmission Co., operating in the vicinity of Helena, Mont., and notable as being one of the highest voltage transmissions in the world, has begun the erection of a steam turbine power plant, to operate as a reserve to their present water-power plant. A large part of the output of the Missouri River system is transmitted to the city of Butte and the Anaconda district mining properties, and the service has grown to such an extent as to warrant the erection of a reserve plant as a protection against serious fluctuations in the available water power. Turbines of the Westinghouse-Parsons type will be installed, the initial equipment consisting of two units of 2,000 kw. capacity each, operating at 1,200 r. p. m., with 150 pounds steam pressure, 28-inch vacuum, 100° superheat. They will be capable of delivering continuous overload of at least 50 per cent condensing, or full load without the assistance of a condenser. The generators will be of the revolving field type, and completely enclosed, with forced ventilation. This feature will entirely eliminate the peculiar hum of the high-speed turbine generator of the open type. The generators will operate at 1,200 r. p. m., and deliver 60-cycle, three-phase current at 2,400 volts. The motor-generator exciting and complete switchboard equipment accompanies the turbine plant. The Westinghouse Machine Co., builders of the turbines, will also equip the boiler plant with their mechanical stokers.

Gas Engines.—The De La Vergne Machine Co., of New York, which has recently completed its contract for 40,000 hp. of Koerting two-cycle, double-acting gas engines, of which 32,000 hp. is employed for driving blowing engines, and 8,000 hp. for driving direct current and polyphase alternating-current generators, have recently been given a contract for three 500-hp. Koerting gas engines to be direct connected to 325-kw., 550-volt direct-current Crocker-Wheeler Generators for the Boston E[illegible] Company. These engines will be put in operation about Jan. 1, 1906. Owing to the rapidly increasing Southern business of the past year the De La Vergne Machine Co. has established a branch agency at Atlanta, Ga. This agency is to cover the States of North Carolina, South Carolina, Alabama, Florida and Georgia, and will handle business connected with the three lines of machinery manufactured by the De La Vergne Machine C[illegible] refrigerating and ice-making machinery, "Hornsby-Akroyd" oil engines and Koerting gas engines. Their representative will be Mr. W. M. Hargreaves, and the office will be located at 510 Candler Building.

Electricity in the Cement Industry.—Cement is divided into two classes—slag and natural cement. In the manufacture of natural cement the rock is taken from the quarry or mines and burnt in a kiln. It is then run through a conical crusher, or between rolls and ground into a powder, after which it is conveyed to screens, which separate the cement which is fine enough to be packed. The coarser particles then go to fine grinding machines. Considerable power is required in these cement plants, and electricity is coming into general use. Either the alternating or direct-current system is adapted to this work, the choice depending on local conditions. The plant of the Iola Portland Cement Works, at Dallas, Tex., is equipped with Westinghouse type-S direct-current motors, and a contract just closed with the Santa Cruz Portland Cement Co., San Francisco, Cal., gives a good idea of the size of some of these plants. This order calls for one 800-hp. type-C motor, ten 250-hp., fifteen 150-hp., one 75-hp. and one 30-hp. type-CCL motors. These are all alternating-current motors of the induction type, and aggregate a total of 5,655 hp.

Storage Battery Exhibit.—At the recent Madison Square Garden Electrical Show, in New York City, the General Storage Battery Co. had an interesting exhibit of Bijur "high-duty" batteries and accessories of storage battery practice. Of special interest among the exhibits of this company were a charging booster and a constant-current booster. The charging booster consists of a constant-speed motor, driving a separately excited generator. When charging a storage battery, the armature of the separately excited generator is connected in series with the station bus-bars, and the combined voltage of the two sends current into the battery. As the battery charge proceeds its voltage increases, the excitation of the separately excited machine is increased accordingly, to maintain the charging current to the battery. The constant-current booster also consists of a constant-speed motor driving a separately excited generator. The excitation of this generator is automatically varied, in order to permit only a constant current to pass from the armature. The armature of the generator is connected to the terminals of a single storage cell, and at all times passes a current of constant value to this cell, regardless of the work being done in the circuit from the battery. The field winding of the generator consists of a single standard 110-volt winding, the exciting current in this winding being automatically varied by means of the Bijur regulator. The operation of fluctuating loads, such as in the operation of elevators, [illegible] was neatly illustrated on a miniature scale by some exhibits, in which violently fluctuating currents, produced by closing and opening an electric switch, were [illegible] by a storage battery, operating in conjunction with a [illegible] which fed a practically steady current to the [illegible] of an average value equal to the amount of the vi[illegible] tions. The Bijur "high-duty" battery, which is built by

the General Storage Battery Co., was described at length in our Vol. III., page 203. The exhibit was in charge of the office force of the General Storage Battery Co., presided over by General Sales Manager J. B. Cowen. The General Storage Battery Co., whose factory is located at Boonton, N. J., has recently opened branch offices at the following addresses: Chicago, 111, 135 Adams Street; Cleveland, Ohio, 1234 Williamson Building; Washington, D. C., Commercial Bank Building.

Personal.

Mr. Isedore S. Prenner, E. E., attorney and counsellor has taken new offices at 1108 Betts Building, Philadelphia, where he will continue his practice as consulting engineer and electrochemist.

Mr. R. J. Truax has returned to Deadwood, S. D., from Cleveland, Ohio, where he attended the stockholders' meeting of the American Tungsten Co. A hoisting plant and camp buildings are now being erected on the tungsten property near Deadwood.

Mr. Edward Taylor, the distinguished inventor of a continuous electric furnace process for the manufacture of carbon bisulphide, sailed for Europe on Dec. 9. Mr. Taylor will attend the Congress of Applied Chemistry in Rome.

Dr. N. Monroe Hopkins has been appointed chief electrical engineer of the Bureau of Docks and Yards of the Navy Department in Washington. The office is a new one, which became necessary by reason of the ever-growing use of electricity in the navy yards and stations. Dr. Hopkins is a member of the firm of Monroe, Hall & Hopkins, chemists, assayers and electrical engineers, of Washington, and has held for several years the chair of electrochemistry at George Washington University. A handsome treatise from his pen on experimental electrochemistry has recently been issued (see our Vol., III., p. 477). Dr. Hopkins has a host of friends among electrochemists and electrical engineers, and many members of the American Electrochemical Society remember with pleasure the good services he rendered them as the very cordial and obliging chairman of the committee on entertainments at the society's Washington meeting in 1904.

Digest of U. S. Patents.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

Treatment of Carbon and Production of Graphite.

No. 572,472. Dec. 1, 1896. Hamilton Y. Castner. London, England.

Produces a species of graphitic carbon by subjecting gas-retort or other carbon to the intense heating action attained by the passage of an electric current through it; *e. g.*, makes an electrode 18 inches long and 1 inch square, attaches to each end a carbon terminal of much larger diameter, covers the whole with powdered charcoal and passes a current of 500 amps. for a few minutes. The product has decreased density and increased conductivity, and is suitable for electrolytic anodes.

No. 617,979. Jan. 17, 1899. E. G. Acheson, Buffalo, N. Y.

Produces shaped articles consisting in whole or in part of graphite, for example, electric generator and motor brushes, pencils and crayons, crucibles and stove polish, by mixing powdered amorphous carbon, such as coke, charcoal or lampblack, with an elemental substance capable of combining with carbon, or with an oxide or salt of such substance, reducible by carbon, such as iron sulphate or peroxide, iron filings or silica, adding a binder, such as sugar, molasses, tar or pitch, shaping and electrically heating to a high temperature. The molded articles are preferably heated by placing them in a cylindrical or oblong bed of fine carbon, surrounded by a thick layer of granular, amorphous carborundum, and passing an electric current through the bed, serving as a resistance conductor. With a furnace having electrodes 16 feet apart and a carbon bed 20 inches in diameter, the starting current may be 300 amps. at 150 volts, increasing to 7,000 amps. at 100 volts. For motor brushes, the charge may consist of a mixture of powdered coke or charcoal 99 parts, and iron oxide 3 parts. The production of graphite is considered to be due to the successive production and decomposition of a carbide, the added base first combining with carbon and then being driven off as the temperature rises, to recombine with other carbon, until it finally escapes. It may be advisable to leave a portion of the amorphous carbon unconverted, to make the article stronger.

No. 618,704, Jan. 31, 1899, Hiram S. Maxim, London, England.

Produces a hard crystalline material resembling diamond, by heating a mixture of liquid or solid carbon dioxide and gasoline in an electric arc. The carbon in contact with or contiguous to the electrodes is converted into a species of diamond scale. It may be necessary to continue the high temperature for a long time, to allow the carbon to crystallize. By having only enough carbon to convert the carbon dioxide into carbon monoxide, with a little free hydrocarbon remaining, "the crystallization may take place from the residuum of gases." The product is used for the manufacture of lamp filaments. The electric furnace is a massive vessel of steel, lined with silica or magnesia brick, or with a metal tube having a spiral cooling pipe and closed at the ends by screw stoppers, through which pass the water-cooled holders of the carbon-rod electrodes.

No. 645,285, March 13, 1900, E. G. Acheson, Buffalo, N. Y.

Produces graphite by electrically heating a carbonaceous material, naturally containing distributed impurities which will progressively convert the carbon into a carbide: for example, anthracite coal containing 5.783 per cent of ash, consisting largely of silica, alumina and iron oxide, or lignite, peat, non-caking coals and some kinds of wood or charcoal. The charge is placed around the core of a resistance furnace, such as that used for the manufacture of carborundum, preferably a core of solid or loose graphite of reduced diameter. As the material is heated carbide, or carbides, for example, of silicon or iron, are progressively formed and decomposed, and the carbon is thus gradually converted into graphite before the final volatilization and escape of the impurities.

No. 702,758, June 17, 1902, E. G. Acheson, Buffalo, N. Y.

Produces graphite electrodes, motor and generator brushes, battery carbons, etc., by molding the articles of a mixture of carbon and impurities which will convert it into graphite, and placing the articles in a resistance furnace, transverse to the path of the current. The furnace may have a length of 30 feet and a width, for articles 30 inches long, of 52 inches. The base and walls are made of firebrick, the bottom being covered with a layer of silicon carbide. The articles are placed in a bed of finely-granular coke, and a thick layer of a mixture of ground coke and sand is placed between the bed and furnace walls and over the bed. The terminals, of amorphous carbon, may have a cross-section of 400 square inches. The articles, if plates, are arranged in spaced piles. When the articles are cylindrical they may be arranged transversely with their surfaces in contact, without interposed packing. The starting current may be 1,400 amps. at 210 volts, increasing for 5 hours to 3,600 amps. at 210 volts. The voltage is then lowered, the amperes increasing to 9,000 at 80 volts. The furnace is then cooled and the articles removed.

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J. W. RICHARDS, PH. D.....*President*
E. F. ROEBER, PH. D.....*Editor*

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Refractory Uses of Bauxite.

We heartily commend to the attention of our readers a recent article by Mr. A. J. Aubrey, published in this issue, on the refractory uses of bauxite, especially in open hearth steel furnaces, in rotary Portland cement kilns and in lead refining furnaces. The problem of providing an efficient furnace lining became a burning question with the advent of extremely high temperatures in the electric furnace. The results obtained in this field are now bound to react on the practice of metallurgical furnace work with comparatively lower temperatures. It is not accidental that, in this development now going on, this journal has been the leading journalistic exponent. It is true that metallurgists have long recognized the importance of the problems involved; among others, the Thomas basic Bessemer converter stands a monument of the splendid work of the old metallurgy in this field. But it is equally true that with very few exceptions refractories have not been made up in the past with any regard as to the conditions which they will meet in the special furnace in which they are to be used, and if attention has been given to this matter, recognized authorities agree that it has been "simply guess work or sort of a hit or miss business." That this situation is now changing, Mr. Aubrey's article bears witness. More than once we have expressed our opinion on the necessity of research laboratories for our large manufacturers, and it is with great pleasure, therefore, that we say that Mr. Aubrey's article begins with the first important achievement of the new research laboratory of the Laclede Fire Brick Mfg. Co. There is an immense work still to be done and there is room for all, but much will depend on the coöperation of manufacturer and user, of the refractory maker and the metallurgical engineer. Even those metallurgical managers who still consider research laboratories to be a fad of modern times, will become interested if they learn that by properly paying attention to the matter they may prolong the life of their furnace linings six or eight times. Mr. Aubrey's article marks undoubtedly an excellent beginning.

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The Role of Current Density in Electrolytic and in Electric Furnace Processes.

In published accounts of many electrochemical processes the importance of using a certain current density, as expressed, the amperes per square centimeter must be maintained between certain distinct limits in order to secure the desired effect or to get maximum efficiency. On the other hand, if we compare the practice of different plants using in principle the same process and each adhering to a certain current density considered the most profitable one under the circumstances, we find that the figures for the best current density may differ very greatly. Thus in one electrolytic copper refinery current densities ranging between 0.15 and 0.2 ampere per square decimeter, or 10 to 30 amps per square foot, are found

ferent plants in this country. The reason for this somewhat surprising discrepancy is well known. Any industrial plant is a unit, and its most efficient operation depends on a great many different but interconnected factors. The current density is only one such item in an electrochemical plant, and is recognized by copper refiners to be closely interconnected—from an industrial point of view—for instance, with the cost of power, the amount of wages and the composition of the anodes. Yet in this labyrinth of industrial considerations we need an Ariadne line of thread. It is a narrow point of view if one looks on a subject only from one side, yet a search-light, which concentrates the rays to even one single point, may, nevertheless, give useful information, as long as the observer remains unbiased and does not imagine that he sees the whole thing. Though restricted, therefore, in its scope, a brief and necessarily incomplete discussion of the rôle of current density from the single view point of electrochemistry may nevertheless be useful. To be as broad as possible within our restricted limits of discussion, the consideration shall be based on the energy principle. But we may first mention a single relation which follows directly from Ohm's law and is sometimes useful in calculations. This relation states that the current density in amperes per square centimeter of cross-section equals the voltage drop per centimeter of line of current, multiplied by the electric conductivity. This rule is valid for any cylinder, the walls of which are formed by lines of current as long as the current density is the same for all cross-sections. The rule is, of course, always correct if expressed in the terms of the differential calculus.

* * *

Let us first consider the most elementary case of an electric furnace process in which the function of the electric current is simply to heat the charge to a certain temperature and to maintain it at that temperature for a certain time. An example is the production of artificial emery by fusing calcined bauxite. When the charge is heated, a stationary condition will be reached, and if the ultimate process is simply fusion the stationary condition will correspond to the fusion temperature. In this stationary condition the amount of electrical energy will be exactly equal to the heat absorbed by the fusion plus the heat given off from the furnace to the outside by radiation and conduction. This loss of heat depends essentially on the surface design of the furnace and the heat-insulating qualities of its walls, but for a given temperature and a given furnace the watts lost per square centimeter of surface will have a certain value. The total loss of heat to the outside must be made up by the electrical energy. Thus, with a given charge it is the watts per cubic centimeter which are the important energy factor. This will also be the case if the process does not consist in simply heating the charge with or without fusion, but is accompanied by a chemical reaction, as is the case in the conversion of anthracite coal into graphite and in the production of carborundum and calcium carbide; electrolytic effects, however, must be excluded for the present. In all these cases the electrical energy supply must equal the energy needed for the chemical and physical reactions taking place at the temperature, plus the loss of heat by radiation and conduction. If the chemical reaction is exothermic, its energy is not counted to be taken as negative. This brings up the question: If in electric furnace processes the watts per cubic centimeter are the fundamental item, what is the relation of this item to the current density? From the rule given at the end of the preceding paragraph it follows at once that the watts per cubic centimeter equal the square of the current density, multiplied by electric resistivity. This result also follows at once from the well-known formula of the Joulean heat. It means that the current density determines directly the watts per cubic centimeter, but it also emphasizes another point which, although almost elementary, is often overlooked. It is by no means always true that the resistivity is constant. With a granular charge the resistivity may, for instance, be varied within considerable limits by compressing the charge more or less tightly, and in one and the same furnace the pressure will undoubtedly vary to some extent from point to point. Moreover, the resistivity may change enormously with the temperature, and the temperature is not the same at all points throughout the furnace. For instance, the problem may be to so design a furnace as to have a gradually higher and higher temperature while approaching the tap-hole, with the highest temperature just at that point. This can, of course, be done by properly adjusting the cross-section of the furnace chamber. But the design should not be based on the amperes per square centimeter but on the watts per cubic centimeter. This would, indeed, appear to be the safest proceeding in making the calculation in all such cases.

* * *

Let us now consider the case of an electrolytic process. The energy which is supplied to an electrolytic cell will be partly changed into chemical energy, represented by the oxidation and reduction processes at the anode and cathode respectively, and partly into heat. In order to simplify the discussion, we will first get rid of the heat—which is easier on paper than sometimes in practice. We have the Joulean heat, which is covered by the discussion in the preceding paragraph, and we have also another heating effect which is thermo-electric. Even if we use a cell so large in cross-section and with the electrodes so near together that the Joulean heat would be negligible, the current will tend either to raise or decrease the temperature of the cell, and this temperature effect may be different at the two electrodes. If this latter heat effect is zero, in a special case, then, aside from the Joulean heat, the total electrical energy consumed will equal the chemical energy produced, which is Thomson's rule. If this heat effect is not zero, Helmholtz's formula is to be substituted for Thomson's rule. Now let us consider the electrical energy. This is given in watt-hours, and represents the "free energy" of the electrochemical reaction. The watt-hours are the product of ampere-hours and volts, and the ampere-hours determine directly the number of gramequivalents oxidized and reduced at the electrodes, according to Faraday's law. Thus in electroplating, every 96,540 coulombs or ampere-seconds will deposit 1 gram atom of the metal on the cathode from a univalent salt solution. Now, if an electrolytic process goes on, the reaction, in order to proceed, requires a certain free energy, in order to get so and so many grams of a certain product, a distinct number of watt-seconds, or volt ampere seconds, is required. But this same weight of product also requires a distinct number of ampere-seconds according to Faraday's law. Hence, it

will be seen that the volts are also fixed. When we have an electrolyte which can give a number of different reactions—for instance, a mixture of metallic salt solutions from which the different metals may be deposited on the cathode—each reaction requires a certain number of watt-seconds, say x watt-seconds, to produce 1 gramatom of the product, but each gramatom of the product requires 96,540 n ampere-seconds where n is the change of valency during the reaction. If we have to do with metallic deposition, or with evolution of gas, n is the equal to the valency in the solution. Hence, to get a certain product the voltage must have a distinct value $0.00001036\ x \div n$, and for our metallic salt mixture we have a voltage series of such a kind that if we gradually raise the e. m. f. at the terminals, we can deposit successively one metal after the other. Each metal has a critical voltage in this mixture; with any e. m. f. below this critical voltage the metal cannot be deposited. If we have a mixture of univalent metallic salt solutions, then the decomposition of 1 gram-equivalent of each salt solution—or the deposition of 1 gram-equivalent of each metal on the cathode—will require a certain number of watt-seconds, say, successively, $x_1\ x_2\ x_3\ \ldots$ watt-seconds; *i. e.*, to deposit one metal after the other in this series, we must supply successively more and more energy; if we multiply every figure in the above watt-second series with 0.00001036, we get for univalent ions directly the voltage series; *i. e.*, to deposit one metal after the other in this series we must successively raise the voltage. With a change of valency not equal to unity, the figure 0.00001036 is simply to be divided by the valency change. In any case by a simple multiplication with a conversion factor we can "change" the free energy of the reaction—whether given in calories or watt-seconds or any other energy unit—directly into volts; and each of these conversion factors depends only on the valency change, and is otherwise entirely independent of the chemical nature of the reaction.

* * *

This is after all the quintessence of the principles governing electrolytic action. But, since this whole note is written to discuss the rôle of current density, the question suggests itself, where does the current density come in? Within the electrolyte itself its effects will be evident when looked upon from the point of view of the ionic theory. The transport of electricity through the solution occurs by the migration of ions, and the velocity of migration is—at least for dilute solutions—equal to the ionic mobility (which is a characteristic constant of each kind of ion) multiplied by the voltage drop per unit length of line of current. Now this latter quantity is, according to the rule given above at the end of the first paragraph, directly proportional to the current density. However, the chief interest in electrolytic action lies in the reactions at the electrodes, and, as we saw before, these depend essentially on the voltage. We may go further and consider the anodic and the cathodic reactions separately. Then we can say that the anodic potential drop and the cathodic potential drop determine what can and will happen at the anode and cathode respectively. If we gradually raise the voltage new reactions set in, but under the conditions how such experiments are generally carried out, the current density increases together with the voltage. Instead of a critical voltage series we could, therefore, give a critical current-density series. But the voltage series is independent of the arrangement of the experiments, while the current density depends materially on the latter. However, in all that precedes we have always assumed that we have a given electrolytic system, and all above considerations are strictly correct, as long as this system remains unchanged. But for all practical applications it is of the greatest importance to take the fact into account that the current itself tends to continually change the system by introducing new products at the electrodes into the system; and it is evident that here the current density is of decisive importance. The greater the current the larger is the amount of new products formed at the electrodes; further, the smaller the electrode surface the less is the quantity of electrolyte which is in contact with the electrodes, and the greater is the liability of the nearly-formed products to displace the original electrolyte. This explains, for instance, why in copper refining the best current density depends on the composition of the anode. The conditions can be clearly seen in storage battery operation. The capacity of a given lead accumulator is the smaller the shorter the time of discharge or, what is the same, the higher the rate of discharge. This is distinctly the effect of current density. The higher the current density the greater the dilution of the electrolyte in the pores of the accumulator plates and the lower the voltage, and, therefore, the efficiency. As Mr. D. H. Browne once wrote in our columns: "Close experiment will almost always reveal the proper conditions under which certain work can be done, but the continuance of these conditions is attainable only by continual vigilance, which is as much the price of liberty as it is the price of success." A necessary part of this vigilance in electrolytic processes must consist in watching the current density and adjusting it to the needs of the moment.

Loss of Heat by Radiation and Conduction.

In any metallurgical furnace the energy required for its operation is in part usefully spent for the production of the necessary physical and chemical reactions, and is partly wasted by loss of heat due to radiation and conduction. Some day metallurgical engineers will reach the same status which has already been reached by electrical engineers, who, in dynamo and transformer design, can predetermine on paper the losses, and therefore the efficiency. It should not be difficult for metallurgists to get practical rules for determining the radiation loss. As a first approximation the outside radiating surface of a furnace is thermodynamically a black body, and the radiation loss is then given by the Stefan-Boltzmann law. The necessity of maintaining the radiating surface on as low a temperature as possible is manifest from this law, since the loss by radiation increases with the fourth power of the temperature. On the other hand, to get rules for the predetermination of the loss of heat by conduction, much more experimental work is to be done. But we note with pleasure that various experimenters are now working in this direction. The recent work of Hutton and Beard (our Vol. III., p. 241) and the experiments of Bied, mentioned in the Synopsis of the present issue, are interesting examples of useful work in this field.

The Outcome of the Detinning Suit.

The long legal battle between the Vulcan Detinning Co. as complainant and the American Can Co. as defendant, has now been decided in favor of the latter. The decision is specially interesting in its references to the history of the electrolytic detinning industry, and also with respect to the protection of secret processes by law.

The decision was rendered by Vice-Chancellor Bergen, in the Court of Chancery of New Jersey, Newark, N. J., and reads in part as follows:

"The evidence in this cause shows that the firm of Th. Goldschmidt, of Essen, Germany, had, prior to 1894, as the result of experiments pursued for that purpose, so perfected a process by which well-known chemical and mechanical principles could be so applied, in the separation and recovery of different metals, as to render the detinning of tin scrap profitable to a degree not theretofore attained.

"The successful combination was not patented, but its inventors sought to protect themselves against infringement by keeping their discovery a secret, only imparting it to such of their employees as the carrying on of the business required, and then only upon a promise not to divulge."

Dr. Hans Goldschmidt is mentioned as the member of the firm most active in making these experiments.

It is stated that "the growth of the business at Essen required the purchase of large quantities of tin scrap, some of which was bought in England and some in New York, the purchases in the latter city being made largely, if not entirely, from A. Kern & Co., a firm engaged in importing and exporting goods.

"The scrap from England appears to have been shipped prior to 1895, through the Zealand Steamship Co., in the employment of which company were M. Laernoes, R. J. Brakema and H. L. Herman, who, through their connection with the Zealand Co., it is fair to presume, acquired some knowledge of the character and extent of the Essen business, for about this time they built a detinning plant at Vlissingen, Holland, under the name of the 'Electro-Tinfabriek,' and through newspaper advertisements, followed by personal interviews, induced two of the confidential employees of the Essen factory * * * to leave their employment, in Essen, and enter the service of the Holland company.

"I have no hesitation in finding that the processes used by Laernoes and his confederates in the Holland factory were, for all practical purposes, the same as that used by the Goldschmidts in their Essen factory. * * *

"The possibility of erecting a detinning plant in this country attracted the attention of Mr. Adolph Kern, the senior member of the firm of A. Kern & Co., as early as 1892, at which time he opened negotiations with the Messrs. Goldschmidt, looking to the establishment of such a plant in America, to be operated according to the processes in use in Essen. The negotiations, carried on principally by correspondence until 1897, produced no satisfactory result, and in December of that year a party of gentlemen, seven in number, of whom the defendant Assman was one, met with Mr. Kern at an office in New York City, to consider the desirability of making further efforts towards the founding of a detinning factory in the United States. * * *

"In furtherance of this conclusion, Mr. Kern first visited the Holland factory, and then went to Essen, where he saw the Messrs. Goldschmidt, and tried to induce them to join in the enterprise, but they refused, giving as a reason that in their opinion the cost of wages and materials in the United States would prevent the successful application of the process in that country. From Essen Mr. Kern returned to Holland, and entered into negotiations with Laernoes, who acted for the Electro-Tinfabriek, of Vlissingen, which resulted in an optional contract, by the terms of which, in consideration of one-third of the capital stock of a company to be organized for the carrying on of the detinning business in the United States, the Holland company were to furnish the processes, etc.

"On his return to this country Mr. Kern submitted to his associates the result of the negotiations he had made in their behalf, and he was directed to accept the option and to procure a formal contract based on its terms. This was done, and Laernoes, with Zeyen, came to this country to supervise the erection of the factory and instruct the purchasers how to operate it. On the arrival of Laernoes the formal contract was prepared and executed between the 'Electro-Tinfabriek' and A. Kern & Co., Laernoes acting and signing as the representative of the Holland firm; thereupon a company was formed under the name of the 'Vulcan Metal Refining Co.'"

Factories were erected in 1898 at Sewarren, N. J., and in 1899 at Streator, Ill. "In 1901, Mr. Assman, having then became a director and officer of the American Can Co., resigned his position with the Vulcan Co., and sold his stock to Mr. Kern, giving as a reason for dissolving his connection with the Vulcan Co., that as the American Can Co. was a large seller of tin scrap to the Vulcan Co., of the selling of which he had charge, he could not consistently continue to occupy the position of buyer and seller. Shortly after Mr. Assman dissolved his relations with the Vulcan Co., the American Can Co. began the erection of two detinning plants, one at Paulsboro, N. J., and the other at Joliet, Ill."

"No reasonable doubt can exist under the evidence in this cause that the methods and processes in use by the defendants, the complainant, Laernoes and his partners, under the name of the Electro-Tinfabriek, and by the Messrs. Goldschmidt, at Essen, are for all practical purposes identical, the changes in mechanical construction being so small and unimportant as to hardly merit consideration.

"The bill of complaint prays that the American Can Co. and the defendants Assman, Baumann, Schmaal and Egbert may be enjoined from operating the detinning plants at Paulsboro and Joliet, or constructing or operating any other factories in imitation of those of the complainant. * * *

"When stripped of all refinements the situation presented is this: Dr. Goldschmidt discovers a secret process for detinning; Laernoes entices Zeyen, a trusted employee, to betray his master, whereby he became possessed of a secret process belonging to another; that secret the complainant purchases under conditions which charge it with knowledge of the wrong committed against Dr. Goldschmidt, thereby helping Laernoes and Zeyen to market their stolen property. It is not to be conceived that a court of equity will stain its hands by contact with such a disreputable proceeding."

"Since the bill of complaint was filed in this cause, the defendants have obtained from the Goldschmidt's a license, executed in due form, permitting them to use their secret in the United States forever, and in Canada for ten years following the date of such license, which was Oct. 17, 1904. This assignment conveys to the American Can Co. a title to this secret process from its discoverer and is superior to any rights which the complainant has."

CORRESPONDENCE.

Optical Pyrometry.

To the Editor of Electrochemical and Metallurgical Industry:

SIR:—The reviewer of the book of Le Chatelier and Boudouard on *High Temperature Measurements* (page 35 of your January issue) brings up the question as to the effect of vapors on the indications of optical pyrometers.

This question has already been dealt with by Prof. Ch. Féry in the use of one of his pyrometric telescopes, in studying the boiling and distillation of zinc, copper and brass in a Moissan furnace (*Annales de Chimie et de Physique*, Vol. XXVIII.,

p. 428, 1903.) In the case of zinc the superheated vapor caused the radiation pyrometer to read about 120° C too high (*i. e.*, 1,040° C. instead of 920° C.); while in the case of copper the vapor present apparently lowered the boiling point by 100° C. at 2,200° C.

That such vapors *may* influence the readings of an instrument sighted upon a tube furnace is unquestionable; but there is room for much more work in determining the limitations of this effect. George K. Burgess.

Washington, D. C.

Specific Resistance of Molten Iron and Steel.

To the Editor of Electrochemical and Metallurgical Industry:

Sir:—I desire to call your attention to a grave error in the figure for the resistivity (that is, specific resistance) of molten iron, recently published in your journal, which is evidently a million times too small. As this is perhaps the only figure published of this constant, it seems most unfortunate that it is incorrect.

The figure referred to is given on page 384, first column, of your issue of October, 1905, in the form of an abstract of a note by Mr. Gustave Gin, read before the American Electrochemical Society, on the electric resistivity of iron and steel at high tempertaures. He there gives the results of a determination for cast iron containing about 93 per cent iron and about 3 per cent each of carbon and manganese, and gives as the resistivity "0.00016 microhm-centimeters for temperatures between 1,280° C. and 1,340° C." This figure should evidently be one million times as great, that is, it should either read 160 microhm-centimeters or else 0.00016 ohm-centimeters, or more correctly ohm, cubic centimeter units.

It will appear evident that the figure, as given, is much too small, from the fact that the resistivity of pure copper in those units (namely, microhm-centimeters) is 1.667, and that, therefore, this molten iron would have a resistivity which is ten thousand times less than copper, which, of course, would be too remarkable to be true.

To further demonstrate its incorrectness, I have made a comparison with some other determinations of the specific resistance of iron and steel at very high temperatures, nearly, though not quite, in the molten state. These determinations were made by W. Kohlrausch, and are given in the well-known tables of Landolt & Boernstein, page 468. As they may be of interest to your readers, I reproduce them here. They are there reduced to conductivity in terms of mercury at 0° C. In this unit the figure for electrolytic iron at room temperature is 8.405; at a red heat, 0.8913; at a yellow heat, 0.8196. For cast steel, the composition of which is not given, the figures are as follows: For room temperature, 5.154; at red heat, 1.096; at yellow heat, 0.901; at nearly a white heat, 0.826. It will be noticed from these figures that the conductivity of electrolytic iron diminishes to about a tenth when heated to a yellow heat, and that of cast steel diminishes to about a sixth at a white heat, which means that the resistance of the former increases about ten times and the latter about six times. The resistivity of iron at ordinary temperatures is, roughly, 0.1 in ohm, meter, square millimeter units. Ten times this would be just about the specific resistance of mercury, which in these units is, roughly, 1, hence unless there is a phenomenal and very marked change in the specific resistance between white heat and the melting point, which is not likely, it is safe to assume that the resistivity of molten iron is, roughly, a little more than that of mercury.

Taking the Kohlrausch figures, roughly, as 0.82 for a yellow or white heat, which corresponds to temperatures of 1,200° to 1,300° C., and, therefore, almost exactly those given in the paper by Gin, I find that when reduced to specific resistance it is about equal to 1.15 in ohms, meter, square millimeter units; that is, a column of molten iron 1 meter in length and 1 square millimeter in section, would have a resistance of 1.15 ohms at about those temperatures. If the resistivity given by Gin is increased a million times it would become 1.6 in ohms, meter, square millimeter units, and, therefore, would correspond well with the figure of Kohlrausch for the solid state, but nearly at the melting point; it is, therefore, fair to assume that the figure of Gin should be a million times as great, and that the error apparently consisted in writing "microhms" instead of "ohms." The paper gives the analyses down to five figures, and, therefore, with very great accuracy, and discusses small errors of 1 per cent, and yet in the final result he has made an error of one million.

I have compared your abstract with the original copy of Mr Gin's paper, and find that the error is in the original paper and not in your abstract of it.

I take this opportunity to express my regret that Mr. Gin did not at the same time deduce another important figure which is so necessary in designing resistance furnaces, namely, the watts per cubic centimeter which are necessary to melt and to keep molten the iron. While, of course, these figures depend largely upon the particular construction of the furnace, yet in all well-constructed furnaces in which the heat radiation can be supposed to have been reduced to a minimum, there would probably be no very great variation in these constants; in any case, however, an approximate figure would be very welcome to designers of such furnaces, as such constants are not yet found in reference books, thus compelling the designer to guess at it. The nature of Mr. Gin's test was such that he could readily have determined these constants without any further tests, as he knew the cross-section, the current and the time.

I hope that the above-mentioned correction will prevent others from being misled by Gin's error, as I was at first, until I noticed the absurdity of the figure. Carl Hering.

Philadelphia, Pa.

A Co-Operative Analysis of a Copper Slag.

To the Editor of Electrochemical and Metallurgical Industry:

Sir:—In an article by Mr. Thorn Smith, having the above caption and appearing in the November and December, 1905, issues of this journal, the results obtained by different chemists are tabulated and criticised.

Being particularly interested in copper and zinc determinations, I noted the results and criticisms relative to them more especially. As regards copper, the only statement to which I would call attention is the following: "The precipitation on aluminium is tedious, and a trace or more of copper is almost invariably found in the filtrate." Mr. Smith does not stand alone in this supposition, as I have frequently seen the same observation made by others. In my laboratory the time consumed by this precipitation is from 5 to 7 minutes, and the filtrates show absolutely no copper with hydrogen sulphide. The only reasons I can conceive of why anyone should fail to achieve this result are lack of proper conditions for the precipitation, or oxidation and re-solution of some of the precipitate during the washing. Having no trouble whatever myself, I am naturally forced to the above conclusions. I always add a little hydrogen sulphide water after the boiling and previous to the filtration, and wash the precipitate with hydrogen sulphide water. The added hydrogen sulphide water is to insure complete precipitation and prevent reoxidation. In the great majority of cases it produces no discoloration due to copper, whatever. When a slight discoloration results it is due to some inattention to details. In any event it is of no consequence, since the copper sulphide is retained on the filter.

Referring to the zinc results, I find the following severe strictures in regard to my method: "The cause of the lack of agreement seems to be in the use of the Volhard method. Fair accurate work with this method is quite possible, but less, and even for satisfactory results it is thought that the

method so serves the majority of chemists. A perusal of the results reported to the committee on methods of zinc analysis will soon convince the closest friend of Von Schulz and Low's method what may be expected in everyday work."

Later in the article, Mr. Smith inconsistently remarks: "Not enough results are at hand to criticize the Von Schulz and Low method." This is quite true.

As I am the sole author of the method referred to and have made thousands of determinations by it, the above statements would seem to call for some reply on my part.

To begin with, comparatively few chemists appear to be using my method precisely as I describe it. There seems to be a disinclination on the part of technical chemists to follow any method not their own just as its author dictates. Even my own assistants have to be watched in this regard. Each operator thinks he sees short-cuts or improvements, or some modification "just as good," with the result that the straight method is rarely employed. Thus I have found in a number of instances that chemists claiming to use my zinc method are not, properly speaking, using it at all, and yet the method is forced to suffer the opprobrium due to their inferior work. Whenever my methods are employed in my own laboratory it is because I know of no better technical ones, and I certainly stand ready to adopt something different as soon as it is actually demonstrated that my zinc method is "worse than useless." My assistants can easily make an ordinary zinc determination in 20 minutes, and the repeated assay of prepared mixtures of known composition, containing all the usual constituents of ores, has amply demonstrated the accuracy of their work. Referring to my records I find that the average difference between duplicates in our last ten umpire zinc assays is 0.047 per cent, and the greatest difference is 0.10 per cent. This is "everyday" work. The ores ran from 30 to 60 per cent zinc. Has anyone anything better to offer?

Mr. Smith states, in regard to the zinc in the slag analysis: "Of the ten results reported Nos. 3, 10 and 13 are the best." I observe that No. 10 was the Von Schulz and Low method, and No. 13 "the same as No. 10, probably." I further find that my method was used in only two other instances, Nos. 4 and 14. Referring to No. 4, Mr. Smith states: "In view of the fact that this chemist had difficulty with practically the whole of his work it would hardly be fair to lay the blame for poor work on the method used." Result No. 4 is classed as "fair, but hardly satisfactory."

On the basis of the above results it would seem that Mr. Smith is not justified in the severe criticisms quoted above.

After reading Mr. Smith's article I wrote to him requesting a portion of the slag in question. On its receipt I handed the sample to one of my assistants and directed him to assay it for zinc and copper, giving no other instructions than to simply make one determination in each case.

The following are his results and those of Mr. Smith. In regard to the latter, Mr. Smith states: "Each constituent was determined several times, and in most instances by different methods." My assistant's determinations were made on 0.5 gram of material in each case:

	Copper.	*Zinc.*
Burnett	0.85%	2.91%
Smith	0.81	2.94

In view of the above, and my own extensive experience with both my copper and zinc methods, I feel that I am amply justified in calling Mr. Smith's criticisms in question.

Denver, Col. A. H. Low.

To the Editor of Electrochemical and Metallurgical Industry:

Sir—I have read Mr Low's criticism of portions of my report on "A Co-operative Analysis of a Copper Slag," with much interest. Mr. Low's difficulty, or misunderstanding, in regard to the precipitation of copper on aluminium is a common one, and one which all of us are prone to. We are apt to assume that if a particular material is amenable to simple treatment in one locality, that no difficulty will be experienced in a region widely remote, where the nature of the material may be widely different.

Mr. Low further assumes that my statement in regard to the presence of a trace of copper in the filtrate is a "supposition." Neither Mr. Low nor any one else can precipitate the copper from a Ducktown slag in from 5 to 7 minutes, providing a reasonable amount of slag be taken for analysis. Our low mattes, averaging about 20 per cent, cannot be treated in any such time; in fact, 15 or 20 minutes are required.

Like Mr. Low I add hydrogen sulphide as a precautionary measure, and often get no precipitation of copper sulphide. So far as I can see the difficulty lies in the fact that our slag is practically an iron slag, and the large amount of ferric iron prevents rapid precipitation.

Our neighbors, the Tennessee Copper Co., experience the same difficulty. On rich mattes, assaying up to 50 per cent of copper, the time of precipitation is much shortened, perhaps to the time mentioned by Mr. Low. As I seldom have occasion to run these high mattes by this scheme, I am unable to state positively. In all of my statements regarding the slag I assume that a reasonable amount of material is taken for analysis, and I do not consider one-half gram sufficient.

In regard to Mr. Low's remarks on zinc I am somewhat at a loss for a reply. In the issue of the *Engineering and Mining Journal* of Jan. 20, 1906, Mr. C. E. Rueger states: "It is generally conceded by experienced chemists that the ferrocyanide method does not give very concordant results on small quantities of zinc. Mr. Furman, for instance, places the minimum at about 4 per cent zinc, using 1 gram of material."

In my report I did say, as Mr. Low states, "A perusal of the results reported to the committee on methods of zinc analysis will soon convince the closest friend of Von Schulz and Low's method what can be expected in everyday work." But I did not "inconsistently" remark that not enough results are at hand to criticize the Von Schulz and Low method. Even the casual reader can see that my first statement referred to the "zinc committee's" work, while the second applied to my own. I endeavored to do the method justice, so far as the slag was concerned.

Wherever there was any doubt as to whether Von Schulz and Low's method was used I gave them the benefit of it. As Mr. Low states, "each operator sees short cuts, or improvements, or some modification just as good." This is true in all methods of analysis, and accounts, in part, for the many discrepancies in co-operative work. Probably what is known as Von Schulz and Low's method, and commonly used, is not that at all in most cases.

Mr. Low further refers to his records, and states that the average difference between duplicates in ten umpire zinc assays is .047 per cent. That this does not prove the infallibility of his method goes without saying, no more than do duplicate results in determining alumina by difference or iron by titration.

If Mr. Low will carefully read my discussion, or at least that part referring to zinc, he will discover that I did not classify No. 4 as "fair but hardly satisfactory." Reference to the tabulated statement of analyses will show that No. 4 found 4.91 per cent of zinc by Von Schulz and Low's method. In my article I stated that "it would hardly be fair to lay the blame for poor work on the method used." I did, however, classify No. 14 as "fair but hardly satisfactory."

There are several methods of analysis which give first-class results in the hands of the originator, but only fair to poor results in the hands of others. That slight changes in manipulation and neglect of vital points, cause the errors is undoubtably true. I have recently attached my name to a method for the preliminary separation of copper from slag, using a large weight of material, and in view of the fact that but few follow it as instructed and then complain of difficulty, it now seems to me to be a mistake for a chemist to so name a method, however original it may be. Thorn Smith.

Isabella, Tenn.

Ore Dressing by Flotation.

The flotation process, which has achieved a great industrial importance in recent years, especially by its successful application to the treatment of Broken Hill tailings in Australia, is the subject of two recent very interesting Faraday Society papers. The joint authors of one of them are Mr. James Swinburne and Dr. G. Rudorf; the author of the second paper is Prof. A. K. Huntington. We give in the following an account of both papers together.

Practice of the Flotation Process.

Practice in Broken Hill District.—The operation of flotation processes is concisely stated by Prof. Huntington as follows: "Flotation processes [of the Potter-Delprat type] have been successfully applied in practice in further concentrating the metalliferous portion of certain tailings, *i. e.*, residues which had previously resisted concentration, produced in the treatment of the sulphide ores of the Broken Hill mining district.

"The Broken Hill ores are composed of an intimate mixture of sulphides of lead and zinc carrying silver, with a gangue or non-metalliferous admixture mainly composed of garnets, quartz and rhodonite (silicate of manganese), with small quantities of rhodocrosite (carbonate of manganese), siderite (carbonate of iron) and calcite.

"The following analyses given by the Broken Hill Proprietary Co. in an inquiry in the Supreme Court, Victoria, show the composition of the tailings from the Broken Hill district:

	Broken Hill Proprietary No. 1.	*Block 10 Mine.*	*Central Mine.*
PbS	6.62	5.31	1.18
ZnS	25.37	34.86	37.89
Cu_2S	0.18	....	0.12
FeS_2	2.88	2.58	2.88
FeS (in blende)	2.37	2.25	2.32
FeO	5.61	....	4.76
MnO	11.23	....	9.55
Al_2O_3	6.90	....	4.06
CaO	3.94	....	2.92
Co_2	0.44	1.50	1.40
SiO_2	32.70	....	24.30

"The flotation process consists in adding to the sulphide tailings either a solution of dilute sulphuric acid, say 2½ to 10 per cent in practice (Potter's process), or a solution of acid sulphate of soda (Delprat's process), at a temperature of about 65° C. Under these conditions the metallic sulphides of the ore are floated up by attached bubbles of gas, and form a coherent scum which can be removed, leaving the earthy matter. When carefully carried out the separation is practically a quantitative one."

Ores Suitable for Flotation.—A most important question is, of course, which ores can be treated by the flotation process, and in this respect Messrs. Swinburne and Rudorf give some interesting notes in their paper. While this information is somewhat interconnected with their argument on the physics of the process which will be given below, the results of some tests will be given here which they have made with various ores and sulphides in dilute acid or acid sulphates of soda. Of those tested "the following are most easily floated, in order of merit:

"Molybdenite.
"Stibnite.
"Galena.
"Mixed zinc-lead sulphides, such as the Broken Hill ores.
"Copper glance.
"Zinc blende.
"Iron pyrites.

"The last two scarcely float at all. Copper glance only floats if slightly weathered, and then only in very small amounts.

"With the Broken Hill ores the ratio of the Zn S in the concentrate differs very little from that in the ore, most the gangue being left behind. Galena is fairly easy to float if fine ground, but owing to its rather high specific gravity very fine pulverization is necessary to float it successfully. Stibnite and Molybdenite go quite easily, owing to their very great [illegible]

"We have made a good many experiments with various ores but do not think it is necessary to go fully into them here. We have, therefore, only given this short list as an example of the way in which things go.

"We also tried some of the gangue obtained from the Broken Hill ores by the Phœnix process due to one of us. This looks like river sand, and consists principally of rhodonites and garnets. Not the slightest flotation was obtained.

"It is interesting to note that ground malachite—copper carbonate—floats fairly well in water. It may be remembered that this mineral has a peculiar greasy feeling. Beyond this we have not found any other carbonates that float.

"This list is only roughly correct, however, and it is most probable, in fact almost certain, that various samples of sulphides differ considerably, and if a slight weathering is necessary to produce the small trace of carbonate needed to give flotation, the 'greasiness' in itself is not sufficient to give the separation. It is, in fact, quite impossible to tell beforehand which ores can be separated by flotation."

Nature and Origin of the Gas Causing Flotation.

The first question to be answered is: What are the gas bubbles which lift the sulphide particles and where do these gas particles come from?

Nature of the Gas.—As Messrs. Swinburne and Rudorf point out, "the explanation generally given is delightfully simple. It is said that the acid liberates hydrogen sulphide from the sulphides, and the gas then sticks to the particles and carries them up. As the acid does not act on the gangue, which in such cases is largely silicious material, the gangue is not carried up, and thus the separation takes place.

"This theory does not fit the facts. In the first place, the sulphides generally treated are not attacked by the acid solution, so that there is no hydrogen sulphide evolved; or, at most, it is given off in very small quantities, which are out of all proportion to the amount of material floated. The sulphides of zinc, iron and manganese alone are attacked at all by dilute acids. It cannot, therefore, be the hydrogen sulphide that carries up the particles of sulphide. On the other hand, there is a considerable evolution of gas, but it is carbon dioxide, and carbon dioxide is chiefly given off by the gangue, which generally contains calcite and other carbonates, the carbonates being mostly produced by the weathering of the sulphide particles. According to the theory just given, this action, contrary to what is actually found to be the case, ought to float up either the gangue as a whole, leaving the sulphides, or it should at least float up the carbonates. The next point is that in most cases flotation does not take place at all until the temperature approaches boiling point."

Origin of the Carbon Dioxide.—Prof. Huntington in his paper starts right away with the statement that it has been shown beyond question that the gas causing flotation is CO_2, and then discusses where it comes from.

"It was at first assumed by some that the CO_2 was derived from calcite, and by others that it came from carbonates, produced on the surfaces of the sulphides by 'weathering.' This latter supposition cannot be correct, because the Broken Hill ores at present mined are obtained from a depth below the water level, and on examination have been found to be free from 'weathering.' On going into the matter I was able to prove that the CO_2, which gives rise to the flotation, is liberated mainly from the rhodocrosite and siderite, and comes not from the calcite."

For this purpose Prof. Huntington made a [illegible] experiment with Broken Hill "Block 10" tailings, using a H_2SO_4 solution. So soon as the flotation had proceeded as far as "

could go (about 2 minutes) the solution was separated from the tailings and analyzed for the Fe, Mn, Ca, Zn and Pb which had passed into solution.

The following is the result calculated as carbonates:

	Per Cent.
$FeCO_3$	1.30
$MnCO_3$	1.85
$CaCO_3$	0.15
$ZnCO_3$	1.27

It will be observed that the quantity of $CaCO_3$ is very small, but, of course, that might be due to the slight solubility of $CaSO_4$. Another experiment was, therefore, made by Prof. Huntington in which HCl was used instead of H_2SO_4, with the following result:

	Per Cent.
$FeCO_3$	1.87
$MnCO_3$	1.61
$CaCO_3$	0.55
$PbCO_3$	1.20

Zn not determined.

Here, again, the $CaCO_3$ is very little, although $CaCl_2$ is very soluble. The merest trace of sulphuretted hydrogen, if even that, is given off.

"In a 5 per cent sulphuric acid native iron carbonate is not acted on until a temperature of about 65° C. is reached, which is the temperature at which flotation takes place with the sulphide ore. Carbonate of manganese behaves in a similar manner. If either be added to sulphide tailings, from which the CO_2 has been removed by previous exhaustion with acid and washing without drying, and acid or acid sulphate be added, as good a flotation is obtained as with the tailings from which the CO_2 had not been removed.*

"Carbonate of lead gave no flotation, the action of sulphuric acid on it being very feeble, owing to the formation of an insoluble coating of lead sulphate.

"Zinc carbonate, when added to sulphide ore, is readily acted on in the cold, and practically gives no flotation of sulphides with either a cold or a hot solution. Owing to the crushed native carbonate being light and spongy, it floats in the solution and generates the gas there rather than on the ore. So far as I could judge, if naturally present in the ore, it would give, if any, a very unsatisfactory flotation, which is characteristic of carbonates decomposed in the cold. Sodium carbonate gives no flotation, but separates the sulphides and the gangue into two layers, and calcium carbonate, although it gives a certain amount of flotation, does not give rise to a true scum, but only to a floating, spongy material, the remainder of the sulphide falling back on to the top of the gangue. A sample of yellow dolomite, when treated with H_2SO_4, gave off some CO_2 at about 30° C., and at about 40° C. the sulphides of the tailings (previously exhausted of CO_2 and washed with alcohol) were lifted, but no coherent scum was formed, the sulphides falling back.

"A copper ore veined with $CaCO_3$ on slightly warming (it was a very cold day) gave off abundance of CO_2, and the finer particles of sulphide floated without, however, forming a good and permanent scum. After the CO_2 had all been exhausted the whole of the sulphide was easily floated by adding $FeCO_3$.

"It has been stated that tailings can be exhausted of CO_2 by the action of acid in the cold. The following experiment was made, which does not bear out this statement, and goes to corroborate the other evidence that the CO_2 is not derived from calcium carbonate:

"Twenty grams Broken Hill tailings and 25 cc. H_2SO_4 (5 per cent) were placed in a flask fitted with a cork and delivery tube dipping into paraffin. No gas was evolved in 24 hours, and on decanting off the liquid and heating no gas was obtained. On adding fresh acid to the residue and heating a normal flotation occurred.

"It has also been suggested that the reason why the flotation does not take place until the temperature has been raised, is that the Co_2 is at first absorbed by the solution. I have made an experiment in which the solution was first saturated with CO_2, but it made no difference, the flotation did not begin until the usual temperature was reached.

"I have also shown experimentally that only acids which attack carbonates of iron and manganese are any use, and that this only takes place in a hot solution.

"I find, however, that sulphur or graphite mixed with calcite and treated with H_2SO_4 in the cold completely float, even when these bodies have been previously thoroughly wetted by grinding them with glycerol.

"These experiments, coupled with the fact that $CaCO_3$, does not give good flotation with tailings exhausted of CO_2, is enough to show conclusively that $CaCO_3$ is not the source of the CO_2, causing flotation of the Broken Hill sulphide tailings. Beyond that we have the fact that no carbonate which is decomposed easily in the cold gives good flotation, or indeed flotation at all, with the exception of $CaCO_3$, which gives a poor flotation. Crystalline $FeCO_3$ and $McCo_3$ are, in fact, the only carbonates which give flotation with a true coherent scum. $FeCO_3$ and $MnCO_3$ in the precipitated amorphous condition do not give any flotation. In that state they are easily decomposed by acids in the cold.

"Experiments were likewise made to obtain flotation with CO_2 by using a solution into which it had been compressed, the solution being finally warmed. Success did not attend the attempts to float sulphides in this way."

Prof. Huntington feels justified, therefore, in assuming that the effective CO_2 in the flotation process, as applied to Broken Hill tailings, is derived from minerals composed of crystallized carbonates of iron and manganese.

Theory of the Flotation Process.

Swinburne and Rudorf's Capillary Theory.—Messrs. Swinburne and Rudorf consider that the whole matter is mainly a question of capillarity. Two distinct capillary forces come into play at the boundary surface of the solid and the liquid. The first one is adhesion, that is attraction between the solid and the liquid particles; it is measured by the contact angle between the surfaces of the liquid and solid. The second is cohesion, that is the attraction between the liquid particles themselves, as measured by the surface tension. "Greasy" substances are those for which the cohesion of the liquid is greater than the adhesion of the liquid to the solid. Greasiness seems to be intimately connected with the formation of a gas film or air film on the surface. The degree of greasiness increases with time as the gas film gets thicker.

To apply these conceptions to the explanation of the flotation process it is necessary to explain with their aid why are the sulphides selected by the gas bells to be floated, and why does the temperature matter?

Let us consider a particle of ore and the gas bell. The surface tension of the liquid tends to make the surface of the liquid as small as possible consistently with its still containing the air or gas. If the particle has no adhesion the surface of the liquid surrounding it must be considered as part of the surface which has surface tension, so that this surface will arrange in such a way as to be the least possible while including both the gas and the particle. If the particle is large in proportion to the gas bubble the surface will surround it, the gas will fill up any inequalities and allow the surface to take the spherical form. If the particle is small the result will be a gas bell with a particle nearly inside it. This is the effect of cohesion.

* Cleveland oolitic carbonate and other specimens of clay ironstone give a very poor flotation with Broken Hill tailings. The sulphides distinctly separate from the gangue, but will not float to the surface. This is probably due to the presence of the clay. But graphite floats perfectly after being washed with alcohol, when Cleveland ore and acid are added.

On the other hand, adhesion tends to increase the contact surface between the solid particle and the liquid so that the particle is more and more immersed in the liquid until, with increasing adhesion, the gas bell is detached.

In order to make particles of ore float up, it is necessary to get gas bells that will stick and not detach. Most constituents of ores have too great adhesion for water or dilute acid to be floated, and to float them it is necessary to diminish the adhesion between solid and liquid. This can be done by increasing the temperature. Raising the temperature reduces both adhesion and cohesion, but while the surface tension vanishes only at the critical temperature, the adhesion appears to vanish about the boiling point. By raising the temperature near the boiling point, the adhesion is therefore sufficiently reduced for the surface tension to overcome it, and the attached gas bells then lift the particles. Even then particles with small adhesion will alone be raised. This explains the effect of temperature.

The phenomenon of adhesion seems to be intimately connected with that of the air-film. Surface tension produces an extra pressure on the gas in a bell, and if the bell is very small the pressure is very great, so that it is difficult for the bell to start. The air-film may give it a chance of starting.

The difference of adhesion of various minerals to dilute acid or water is considerable. Thus galena is more difficult to wet than silica, and so on; and those ores alone can be concentrated by flotation "whose valuable constituent is some greasy sulphide such as galena, or at most so automatically attached to the greasy sulphide that it comes off with it. Zinc sulphide is thus easily raised by galena when intimately associated, as in the Broken Hill and some of the Vieille Montaigne Co.'s ores.

"It is not enough to have the greasy particles; it is necessary to produce the gas bells and get them against the particles. In practical work the only gas that comes off is carbon dioxide, and this appears to come largely from calcite. But mixing calcite with an ore which has a greasy constituent will not necessarily float it; the gas is apt to come away from the particles of calcite without getting attached to the ore. It seems much more likely that the gas that really effects the flotation is generated by the action of acid on small quantities of carbonates produced by a slight weathering of the ores. It is most likely due to small amounts of carbonate of iron and manganese. These carbonates are also not attacked by dilute acid in the cold."

Huntington's Suggestions of Electrostatic Action.—Prof. Huntington points out that the flotation process must either be considered to be due to surface tension phenomena or to electrostatic attraction. While he does not maintain that the latter explanation would hold, yet he makes some interesting suggestions concerning it. He has found experimentally that the CO_2, as it rises, carries a positive electric charge with it while the solution gets negatively charged; the negative charge of the solution can be shown by an electrometer connected with the containing vessel only if the gas leaves the solution, taking the positive charge with it. Prof. Huntington also has established that the gas or the fog in the bubble gives up its charge very slowly, and cannot therefore be easily neutralized. He then made experiments to see whether it is possible to make the sulphides float by means of air or CO_2, passed through the sulphides with or without previous electrification. No perceptible selective flotation was obtained in either case, and he was unable to ascertain whether the surface tension of the liquid in contact with the sulphides was influenced by electrification. In other words, while he established that electrification occurs, yet he was unable to obtain any evidence that it plays any part in the flotation.

Huntington's Criticism of Swinburne's and Rudorf's Theory —Prof. Huntington criticises the assumption of Swinburne and Rudorf of the presence of an air-film on the surface of the sulphide particles; he found that the particles were floated perfectly after he had taken great care to remove the [illegible] film of gas by exhaustion, [illegible] treatment with air free distilled water and [illegible] the pump, etc. He remarks that flotation depending on the presence of an adhering film of air is made out of [illegible] suggested concentration processes, which however are different in principle from those under consideration. In these processes the ore is brought very carefully into the surface of the water, so that the gangue or easily wetted portion may sink and the sulphides which are not well wetted may remain floating. What is aimed at is to prevent the sulphides from sinking, which is a different thing to making them float up from the bottom.

It is a curious fact that specimens of zinc blende containing but little gangue, which do not form a scum on treatment with sulphuric acid and carbonate of iron, give an excellent flotation if previously mixed with sand. On the other hand, too great dilution with inert matter injures the flotation. The Broken Hill vanner tailings appear to be ideally suitable for the flotation process. Prof. Huntington thinks that "these experiments dispose of Messrs. Swinburne and Rudorf's contention that blende scarcely floats at all, and incidentally of the statement that zinc blende is thus easily raised by galena when intimately associated. Swinburne and Rudorf themselves admit that galena has to be very finely pulverized to float successfully; it is different, therefore, to see how it could be expected to float with attached inert blende. It appears more probable that the blende helps the galena to float. This view was strengthened by some experiments made on the wetting of sulphide surfaces.

The degree of wetting of a surface may be judged by measuring the angle which the liquid makes with it. Direct determination of this angle for different minerals by a goniometer method are described, the values obtained varying from 47° to 63° for different sulphides. Quartz and magnetite are completely wetted by water. Alcohol wets all sulphides, but on again placing in water the apparent "greasiness" is restored.

When a gas bubble comes into contact with a mineral particle, the extent of surface of the bubble which will be in contact with the solid depends on the angle of contact of the solid with the liquid. If the angle is such as to cause a relatively large surface of the gas to be in contact with the solid, and the bubble is sufficiently large, the particle will be floated. The less the wetting the greater will be the force required to detach the bubble. If the surface of the solid is wetted the bubble has no attachment at all.

Swinburne and Rudorf on the Action of Different Solutions —As was already mentioned, Potter originally used dilute acid, preferably sulphuric, of about 2 per cent strength, while a saturated salt-cake solution is employed by Delprat. Messrs. Swinburne and Rudorf have found that generally the latter solution works rather better than the former.

"With salt cake solution the bubbles get larger than with dilute acid and rise more slowly, owing to the greater density and viscosity of the former solution. Certainly a much more stable and compact scum is formed when salt-cake solution is used, and this is no doubt due to the viscosity of this solution, but more especially to the surface viscosity which is always greater than the ordinary viscosity or internal friction. It is well known that bodies on the surface of a liquid attract one another if all be wet or all be dry. If one particle be wet and another dry then there is repulsion between them. The compactness of the scum in the case of salt-cake solution is therefore, most likely due to the fact that the particles in consequence of the increased surface viscosity as compared with dilute acid tend to keep drier than with the dilute acid, and therefore attract one another, forming a compact scum. It must further be pointed out that this apparent attraction is really a surface tension phenomenon again. If particles of dry wood rest on water, each produces a little dent in the surface and increases its area. The increase of area of the water surface is most

when the particles are all together, making one big dent. This holds good—to a less extent—with greasy particles floating, and with completely wet particles. With wet particles the dents become prominences, but the surface action is otherwise the same."

Both papers were presented at the same meeting of the Faraday Society, and elicited considerable discussion, an account of which is given by our London correspondent in his monthly letter, published on another page of this issue.

Refractory Uses of Bauxite.

By A. J. Aubrey.

Bauxite is a hydrated oxide of aluminium corresponding to the chemical formula $Al_2O_3 \cdot 2H_2O$, although the proportion of water varies greatly and the aluminium is sometimes replaced by iron. Silica and titanium oxide occur as impurities. The silica varies widely in different deposits. Other oxides of aluminium containing water are diaspore ($Al_2O_3 \cdot H_2O$), and gibbsite ($Al_2O_3 \cdot 3H_2O$). These are found in small quantities, however. The andydrous oxide of aluminium, known as corundrum (Al_2O_3), is found in still smaller quantities than the above, so that bauxite is the only oxide of aluminium found in considerable quantities. There are three localities in the United States where bauxite has been found in commercial deposits, viz.: in Arkansas, New Mexico and the Georgia-Alabama district. The last-named district has probably been worked out by this time, and at present the chief sources of the ore are the Arkansas deposits.

The crude bauxite is washed at the mines to remove some of the free silica. It is next calcined at a temperature of 2,500° F., approximately Seger cone 12, and during this process it gives off its chemical water, amounting to about 30 per cent of the raw ore, and also undergoes great shrinkage. It has been observed that the bauxite shrinks very little until after Seger cone 9, approximately 2,390° F., is reached, and from Seger cones 9 and 12 the greatest amount of the shrinkage takes place. Accordingly the lowest temperature limit at which bauxite should be calcined is 2,500° F.

An analysis of washed calcined bauxite from Arkansas made by the writer is as follows:

	Per Cent.
Mechanical water	0.88
Silica	6.40
Fe_2O_3	1.43
Alumina	87.30
TiO_2	3.99

The calcined material can be bonded with fire-clay, sodium silicate or lime, and made into brick and tile. As little as 4 per cent plastic fire clay can be used for a bond for hand-made brick. When bonded with lime the brick becomes quite hard a few hours after making, so hard that they will not take the impression of the finger nail. This setting or hardening is probably due to the formation of a calcium silicate between the lime and the free silicia analogous to the setting of silica brick when bonded with lime. After a careful drying treatment the brick are burned in down-draft kilns at a high temperature, and when burned they are hard and tough and can be thrown to the pavement or batted vigorously against one another without breaking. A brick 9 inches x 2½ inches x 4½ inches weighs 7.5 pounds, and stands a crushing test of 10,000 pounds per square inch.

The problem of calcining the washed granular bauxite presents itself. It cannot be successfully calcined in a vertical shaft kiln, because it will pack and clog on account of its fineness, and so obstruct the passage of the kiln gases. On the other hand, it is not feasible to calcine it in the ordinary down-draft kiln on account of the perforated floor. The writer believes that to calcine it on a commercial scale would require a rotary kiln using oil, gas or powdered coal for fuel, preferably the former. Provided the cost of fuel is low at the mines these kilns should be installed there in order to save the cost of transportation, which is now paid on the 30 per cent of water that the crude bauxite contains.

Steel Furnaces.

For basic open-hearth steel furnaces a brick high in alumina and low in silica is required. A brick of this character is obtained in the following way: A pure white variety of the pisolitic bauxite, which has already been washed at the mines and a portion of its free silica removed, is selected and sieved. All the fine material passing through the sieve is rejected, as it contains the greater part of the silica. The pisolities, or pebbles about the size of peas, are retained, as they are higher in alumina and freer from silica than any other part of the bauxite. Now, by using lime selected for its freedom from silica, as a bond, a brick can be procured which will contain as low as 6 per cent or 8 per cent silica.

A high silica content has always been the chief objectionable feature of bauxite brick for basic open-hearth steel furnaces, because the silica is attacked by the basic slag. Authorities on open-hearth practice have generally declared that bauxite brick would be suitable for basic open-hearth furnaces, provided a brick could be made with less than 12 per cent silica content. And, indeed, recent tests made along this line seem to bear out this statement.

A test was made several months ago in one of the basic open-hearth furnaces of the Bethlehem Steel Works, in which a bauxite and a magnesite brick were placed side by side near the gas and air ports, and submitted to the highest temperatures attainable in the furnace. The magnesite brick bent and showed viscosity after a period of 7 minutes against a period of 15 minutes for the bauxite brick. Again, a magnesite brick and bauxite brick were bathed in the slag near the doors for a period of some time, after which they were withdrawn and examined when cold. The magnesite brick was incorporated with slag while the bauxite, when broken open, showed that the slag had not penetrated to its center but had remained as a coating over the outside. Both bricks were quite badly affected, but it was readily apparent that the bauxite withstood the action of the corrosive slag equally as well as the magnesite.

More recent tests with brick having a lower content of silica have shown up equally as favorable as the magnesite when exposed to the action of the basic slag. The comparison is hard to make in any case, because of the fact that either bauxite or a magnesite brick, if left in the hearth of the furnace for any length of time submerged in the slag, are so violently attacked that the differences in the portions that are taken out are not easily detected. Even a magnesite brick if thrown into a basic open-hearth furnace will be entirely eaten away after considerable length of time, so that the only real test of bauxite brick will be to build a hearth, or portion of hearth, out of these brick and test them, covering them with calcined bauxite and treating them in every way like the magnesite hearths are treated. Some steps are now being taken towards accomplishing tests of this character.

Sir William Siemens found that bauxite was a superior furnace lining, and even though he used an inferior bauxite, containing as much as 35 per cent oxide of iron, he claimed they lasted five or six times as long as the "Stourbridge First Brick." Siemens also says of bauxite: "It is important to observe that bauxite when exposed to intense heat is converted into a solid mass of emery, of such extreme hardness that it can hardly be touched by steel tools, and is capable of resisting mechanical as well as the calorific and chemical action to which it is exposed."

Bischof, in his "Feuerfeste Thone," says that bauxite "when not impure on account of the admixture of foreign substances,

especially of iron, which generally occurs in considerable quantities in compounds of aluminium, is extremely refractory." He also goes on to say that "the addition of varieties free from iron, or the white ones, to other refractory clays offers the only important means known of increasing their percentage of alumina, and at the same time their refractoriness."

It appears then that bauxite has been regarded as a refractory material for a long time, but that no definite use has been made of it, also that the inferior grades have been more generally tested than the purer ones.

Aside from its use for open-hearth furnaces, which, by the way, is being thoroughly exploited and investigated to make certain of its fitness before taking any risks, bauxite brick have shown up quite successfully in other lines.

Two recent applications are its uses: first, as a lining for rotary Portland cement kilns, and second, as a lining for lead refining furnaces.

Lining for Rotary Portland Cement Kilns.

As a lining for a rotary Portland cement kiln it has shown unusual durability and has given excellent service. As an experiment, the hot zone (about 10 to 12 feet) of a 60-foot rotary kiln fired with coal dust, was lined with a 6-inch bauxite lining, whereas the kiln had previously been lined with a 9-inch fire-clay block.

A block for lining rotary Portland cement kilns in the hot zone must possess the following qualities: It must neither be too hard nor too soft, for if too hard it will not allow the cement coating to stick to the surface, and if too soft it will not hold the coating, but the latter will pull off, bringing portions of the brick with it. The cement coating affords an excellent protection for the lining, and if brick do not hold this coating they soon burn out. The bauxite block made for this purpose has all of the aforesaid qualities.

After ten months of continuous service night and day this 6-inch lining is still doing the work. The loss of output in cement for every 24 hours that one of these rotary kilns is shut down is equivalent to the sum of $250. As the minimum time required for lining or patching a kiln in the hot zone is from 36 to 48 hours, one can easily estimate that the loss of output will be equivalent to a sum varying between $375 and $500. In this case it is quite apparent that a superior lining will be the cheaper in the long run. Again, it must be remembered that only the hot zone need be lined with a bauxite block.

Lead Refining Furnaces.

The second and most recent application of bauxite brick has been that of lining portions of lead refining furnaces.

Pig lead, containing copper, antimony, and occasionally arsenic and other metals in small quantities, is charged into the hearth of the refining smelter and melted by the reverberation of hot gases from above.

The hearth of the furnace, which is 10 by 13 feet in dimensions, is lined on the bottom by 9-inch square brick set on end. The sides of the furnace are lined with a single course of 9-inch brick laid with ends against the walls.

The temperature of the furnace is from 1,300° to 1,400° F., and outside of the part it plays in melting the fluxes bears no relation to the destruction of the brick, as they are sufficiently refractory to withstand much higher temperatures. It is purely a case of chemical action on the brick.

During the process of refining, a scum rises to the surface of the molten lead, and this scum contains most of the impurities which the refining is designed to remove. It consists largely of litharge, or yellow oxide of lead, PbO, copper oxide CuO, and antimony oxide Sb_2O_5, with possibly other oxides in much smaller amounts.

Wherever the brick are exposed to this scum, particularly around the doors, where there is a larger supply of oxygen, and accordingly litharge is more easily formed, and along the level of this scum, on the walls, they are badly eaten away, and have to be taken out after several weeks' service and generally have to be patched after a week has passed. Consequently the furnace loses much time for repairs which lessens its output considerably.

The cutting away of the brick is explained as follows: Multiple silicates are formed between the oxides of the metals in the scum and the silicates of alumina and free silica in the brick, and these are very fusible, because lead oxide, PbO, which is one of the most fusible of the oxides, comprises the greater part of the base or RO element of the silicate. But there are also several other oxides in the base of the silicate, and it is quite well understood that multiple are more fusible than single silicates.

Hence, as very fusible silicates are formed between the scum and the fire-brick the brick are very rapidly decomposed and eaten away. It was observed that a porous brick wore out faster than a dense brick, also that the scum penetrated along the joints for several inches beyond the exposed face of the brick. This was probably due to the fact that the raw fire-clay which was used in making the joints is much more soluble in the scum than the burnt clay. Accordingly great care should be taken in laying up the brick to see that the joints are as tight as possible.

Whereas, the scum was composed of highly basic oxides, it was considered reasonable to supplement a basic lining for the fire-brick lining, and accordingly a lining of bauxite was put in wherever the brick were exposed to the slag. The result was that the bauxite brick lasted from five to six times as long as the fire-brick lining.

With the above-mentioned uses and its possible advent into open-hearth practice elsewhere, the writer feels that bauxite brick will at some early future date comprise one of our most valuable and useful refractory materials.

Laclede Fire-Brick Mfg. Co., St. Louis, Mo.

The Carborundum Furnace.

By F. A. J. FitzGerald.

When a mixture of silica and carbon is heated to a sufficiently high temperature the silica is reduced, and if a proper amount of carbon is present silicon carbide is formed, according to the following equation:

$$SiO_2 + 3C = SiC + 2CO$$

At relatively low temperatures the silica-carbon mixture yields a greenish-colored amorphous substance, but as the temperature is increased this is converted into crystalline silicon carbide, which has been named "carborundum" by its discoverer, Mr. E. G. Acheson. If the temperature is increased considerably beyond that of its formation, carborundum is decomposed, the silicon being expelled as vapor and the carbon left behind in the form of graphite. Moreover, this decomposition takes place without fusion of the carborundum, and the graphite is left as a beautiful pseudomorph of the carborundum crystals.

The physical properties of carborundum suggest at once that a special form of furnace is necessary for its manufacture. Since it is infusible the furnace cannot be tapped as in the case of calcium carbide, and it is, therefore, difficult to devise any method by which the manufacture of carborundum could be made a continuous process. Moreover, since it is easily decomposed at very high temperatures it is necessary that there be some means of regulating the temperature of the furnace so that this is kept between the limiting temperatures of the formation and decomposition of silicon carbide. This condition indicates that an arc furnace would be unsuitable, for while it is undoubtedly possible to produce carborundum in a furnace heated by an arc, it would not be a practical or efficient method of manufacture. No doubt an arc may be used

in the production of substances that are vaporized or decomposed at the temperature of the arc itself, for as fast as these are formed they may be drawn off from the high-temperature zone; but, as we have seen, that is not feasible in the case of carborundum.

In the earliest form of carborundum furnace an arc was used, the furnace charge surrounding two carbons, between which an arc was drawn. This, however, was soon abandoned for a furnace in which the charge was heated by means of a suitable resistor.

For a fairly complete description of carborundum and its manufacture the reader is referred to *Carborundum-Monographien uber angewande Elektrochemie*, Band XIII., from which publication the following description is derived:

Fig. 1 is a diagrammatic sectional plan, and Fig. 2 a transverse section of a carborundum furnace having a capacity of

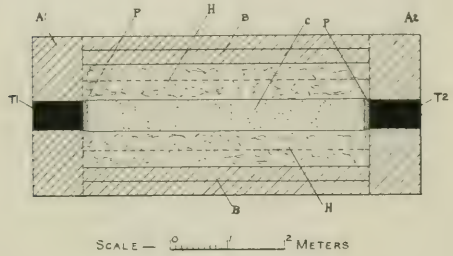


FIG. 1.—SECTION OF CARBORUNDUM FURNACE

750 kilowatts. A1 and A2 represent the end walls of the furnace, and carry the carbon terminals T1 and T2 respectively. The end walls are permanent structures, unlike the side-walls B, B, which are rebuilt every time the furnace is run. C is the resistor, or "core," composed of granular coke, and is electrically connected with the terminals T1 and T2 by the packing carbon P. The resistor is surrounded by the charge H. The broken lines in each figure indicate approximately the distance from the resistor of the carborundum formation.

The terminals of the furnace consist of twenty-five solid carbons, 860 mms. long, and with a cross-section of 100 x 100 mms. The carbons are set in five horizontal rows of five carbons each, and between each row is a copper plate, which serves to connect the terminal with the cables carrying the current. When a furnace is to be charged the side walls are first built up, and the mixture is then thrown into the furnace until it is rather more than half full. Pieces of sheet iron are set up in a vertical, transverse position a few centimeters from the ends of the terminal carbons, in order to keep the mixture from coming into contact with these. This is done because the space between the pieces of sheet iron and the carbons must be filled with finely powdered coke, which forms the "packing carbon" that makes electrical connection between the terminals and the resistor. Then a semi-circular trench, having the diameter desired for the resistor, is formed between the pieces of sheet iron at either end of the furnace. This trench is filled with granular coke, and then as much more of this material added, so that when rounded off on top the resistor has the form of a cylinder extending from one piece of sheet iron to the other. The packing carbon is put between the sheet iron and the carbon terminals and tightly rammed into place, after which the sheet iron is removed. More mixture is thrown into the furnace and it is ready for the current.

The theoretical considerations discussed in a former article[1] apply directly to the carborundum furnace, which consists essentially of a cylindrical resistor surrounded by the material to be heated. If the radius of the resistor is p, its length L and temperature Θ, and if the temperature of the inner surface of the cylinder of surrounding material is Θ_1, then we have for the quantity of heat C flowing from the resistor per second:

$$C = 2\pi pL\,(\Theta - \Theta_1)\,A \qquad (1)$$

where A is a constant, and the assumption is made that there is no flow of heat from the resistor to the terminals of the furnace. From equation (1) we have

$$\Theta_1 = \Theta - \frac{C}{2\pi pLA} \qquad (2)$$

If the furnace is to be used for a current of W watts then

$$C = JW \qquad (3)$$

where J is a constant, and substituting this value for C in equation (2) we have

$$\Theta_1 = \Theta - B\,\frac{W}{2\pi pL} \qquad (4)$$

Now, when it is considered that the temperatures of formation and decomposition of carborundum have a certain value it is plain that for a given value of W it is necessary to adjust the radiating surface ($2\pi pL$) of the resistor, so that at no time Θ_1 will be greater than the temperature of decomposition of carborundum. On the other hand, the value of $2\pi pL$ must not be so great that Θ_1 will be below the temperature of formation of carborundum.

After a cylinder of carborundum, having an external radius p_2, is formed, and if the temperature of the outer surface of this cylinder is Θ_2, the quantity of heat C_1 passing through the cylinder per second is

$$C_1 = \frac{2\pi LK\,(\Theta_1 - \Theta_2)}{\log \frac{p_1}{p}} \qquad (5)$$

where K is the heat conductivity of the carborundum cylinder. In the carborundum furnace the rate of generation of heat is kept constant, since W in equation (1) is constant, hence, if C is greater than C_1 there will be a progressive increase in Θ_1, until finally a temperature is reached at which carborundum is decomposed.

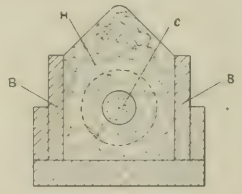


FIG. 2.—TRANSVERSE SECTION OF CARBORUNDUM FURNACE

Without going into more detail it is evident that the consideration of the theory of the carborundum furnace is of importance in its design.

From the theoretical consideration it may be readily deduced that as the carborundum cylinder increases in size the efficiency of the furnace diminishes, since the heat losses become greater, so that after a cylinder of a certain size is formed it is no longer economical to run the furnace. At all times it is necessary to preserve a fairly thick layer of unchanged mixture outside the carborundum cylinder, since it forms a fairly efficient heat insulator. When the current has been cut off from the furnace this unchanged mixture is removed, until a greenish colored amorphous substance is exposed. This is an intermediate product in the formation of carborundum, and after he had treated it "with hydrochloric acid, caustic soda, water, hot oxygen and hydrofluoric acid" was found by Muhlhaeuser to have the following composition[2]

[1] Electrochemical Industry, Vol. II., No. 9.

[2] "Journal of the Franklin Institute," September, 1893.

C	27.93
Si	65.42
Fe_2O_3 and Al_2O_3	5.09
CaO	0.33
MgO	0.21
	98.98

This substance lies immediately outside the carborundum cylinder, and is only a few centimeters thick. The carborundum cylinder, on the other hand, is about 35 centimeters thick, according to the published information on the subject[3], which indicates that the temperatures of its formation and decomposition lie very much further apart than do the temperatures of the formation of carborundum and that of the formation of the amorphous material. In considering this, however, the relative heat conductivities of the two substances would also have to be taken into account.

When the crystalline cylinder is removed from the furnace the granular coke, forming the resistor or core, is found to be graphitized. The graphitized coke may be used as the core of another furnace, and will, of course, have a much lower resistance than a resistor made of ordinary coke. When a furnace is built with a resistor of ordinary coke it is necessary to use a high voltage at the start, and even then it may be some hours before the furnace comes to load. Then follows a great reduction in the resistance, due not merely to the heating of the carbon, but also to its conversion into graphite. The variable voltage is obtained by means of apparatus described in another article[4].

Prof. J. W. Richards has attempted to estimate the efficiency of the carborundum furnace[5]. He defines the efficiency as the ratio between the heat used in bringing up the charge to the reacting temperature plus the heat absorbed in chemical reactions and the total heat generated by the current. The temperature of the furnace is assumed to be 3000° C., and assumptions as to specific heats are also made. From these data the heat necessary for raising the charge of the furnace to the required temperature is calculated, and found to be 42 per cent of the heat energy generated. The heat required for the chemical reaction is calculated as 34.5 per cent, so that the efficiency is 76.5 per cent. Unfortunately, Prof. Richards has failed to take into account the amorphous material that is also formed in the furnace. According to Muhlhaeuser's analysis we may regard this as being amorphous silicon carbide, and in a furnace such as Prof. Richards has assumed the production of this substance may amount to 14 per cent of the carborundum. Moreover, in actual practice it happens almost invariably that a considerable quantity of the carborundum formed in the furnace is decomposed, and this has not been taken into account in the calculations. Finally, the mixture always contains a large quantity of saw-dust, and usually a considerable quantity of water, of which no account is taken. Under these circumstances it is very doubtful if 76.5 represents the efficiency of the carborundum furnace.

In the present state of our knowledge of the working of electric furnaces it is not possible to make even approximate estimates of absolute efficiency, and we must, therefore, be content to compare production and energy. Thus, in his paper before the Franklin Institute in 1893[6], Mr Acheson said he was using 9 kilowatt hours to produce 1 pound of carborundum. According to Prof. Richards, the Carborundum Co. in 1902 produced 7,000 pounds of carborundum with an expenditure of 36,000 hp-hours, that is, 3.8 kw-hours per pound

[3] "Journal of the Franklin Institute," February, 1897.
[4] Electrochemical and Metallurgical Industry, Vol. III., No. 1, p. 9.
[5] Transactions of the American Electrochemical Society, Vol. II., p. 67.
[6] "Journal of the Franklin Institute," September, 1893.

The American Institute of Mining Engineers will hold its next meeting at Bethlehem, Pa., from Feb. 21 to 23. A feature will be a visit to the Hazard works of the New Jersey Zinc Co.

Metallurgical Calculations.

By J. W. Richards, Ph. D.
Professor of Metallurgy in Lehigh University.

CHIMNEY DRAFT AND FORCED DRAFT.

In all problems concerning combustion, we must furnish the air needed for combustion either by suction or by pressure. The original and almost universal method is by chimney draft; the more positive and reliable method is forced draft. Often the two are combined with very satisfactory results.

The waste heat from any metallurgical process or furnace is generally considerable. Most furnaces must be kept above a red heat, and the gases pass directly out of the furnace into the chimney. In such cases the chimney is indicated as the proper source of draft, because it utilizes, although very inefficiently, the ascensive force of the hot gases, and thus works by otherwise wasted energy. In other cases it is practicable to pass the gases through boilers before they go to the chimney, and thus to raise large amounts of steam. The gases are then cooled down so far that they enter the chimney too cold to furnish all the draft needed; in such cases a small fraction of the steam generated will run a steam engine or steam turbine, and run a fan capable of furnishing all the draft needed. In this manner considerable steam is available for other purposes, and great economy is effected.

Chimney Draft.

The principles involved are not obscure or complicated. The total pull, or suction, which a chimney can produce, assuming it to be filled with hot air, is simply due to the ascensive force of the hot air inside, and the measure of this is the difference of weight of the chimney full of hot gases and what it would be if filled with cold air of the temperature outside.

Illustration: A chimney is 6 feet square inside and 100 feet high, uniform, with the gases inside at an average temperature of 500° F., and specific gravity (air = 1) of 1.06. The air outside is at 80° F. What is the ascensive force of the hot gas inside, in total pounds, in ounces per square inch and inches of water gauge?

The volume of the space in the chimney—chimney volume—is $100 \times 6 \times 6 = 3600$ cubic feet. This volume, filled with air at 32° F., would weigh

$$3600 \times 1.293 = 4654.8 \text{ oz. Av.} = 290.9 \text{ pounds.}$$

And, filled with gas at 500° F.,

$$290.9 \times 1.06 \times \frac{491}{500 - 32 + 491} = 157.9 \text{ pounds.}$$

If filled with outside air, at 80° F., the weight would be

$$290.9 \times \frac{491}{80 - 32 + 491} = 265.0 \text{ pounds}$$

We, therefore, see that the hot gases in the chimney, weighing 157.9 pounds, displace 265.0 pounds of cold air, and the tendency of the former to rise upwards in this ocean of air must be

$$265.0 - 157.9 = 107.1 \text{ pounds}$$

To put it in another way, if a piston fitted into the chimney at the bottom, and could move without friction, the piston would have to be loaded with 107.1 pounds to keep it from moving up the chimney. The total upward pull of the chimney is therefore 107.1 pounds.

Since this would be exerted on a piston $6 \times 6 = 36$ square feet in area, the pull or suction per square foot, in pounds, is $107.1 \div 36 = 2.98$ pounds, and in ounces per square inch

$$(2.98 \div 144) \times 16 = 0.331 \text{ ounce per square inch}$$

If the pull or suction is measured on a gauge, as is water pressure, the pressure of a 1 foot column of water at 62°

nary temperatures is 1000 ounces per square foot, or 1 inch of water is

$$(1000 \div 144) \div 12 = 0.507 \text{ ounces per square inch.}$$

The total pull of the chimney is therefore equivalent to

$$\frac{0.331}{0.579} = 0.57 \text{ inch of water gauge}$$

By exactly similar methods of calculation the theoretical total suction of a chimney of any given height and temperature of gases inside and of air outside may be obtained. The suction expressed in ounces per square inch, or in water gauge is, of course, independent of the cross-sectional area of the chimney; it depends only on its height and on the temperatures inside and outside.

The above calculated total suction (allowing nothing for friction, etc.) is called the total head of the chimney, and is usually expressed in terms of cold air (at 0° C.) instead of in water. Cold air is a fluid, and water is 772 times as heavy as it; therefore, a gauge pressure or hydrostatic head of 0.57 inch of water is the same as

$$0.57 \times 772 = 440 \text{ inches of air.}$$
$$= 36.5 \text{ feet of air.}$$

What this head really represents is clearly seen from the above calculations. Its value is to be obtained directly from the height of the chimney, temperature inside and out and specific gravity of the chimney gases (air = 1) by the following relations, in which

h_a = total head in feet of air at 32° F.
h_a = total head in meters of air at 0° C.
t = temperature in the chimney, F°
t = temperature in the chimney, C°.
t' = temperature of outside air, F°.
t' = temperature of outside air, C°.
D = Specific gravity of chimney gases, air = 1.
H = Height of chimney in feet.
H = Height of chimney in meters

$coef$ = coefficient of gaseous expansion, F° $= \frac{1}{491}$

coef = Coefficient of gaseous expansion, C $= \frac{1}{273} = a$

$$h = H \left[\frac{(1 - D) + coef\,(t - 32) - D\,(t' - 32)}{(1 + coef\,(t' - 32))\,(1 + coef\,(t - 32))} \right]$$

$$h = H \left[\frac{(1 - D) + coef\,(t - D\,t')}{(1 + a\,t')\,(1 + a\,t)} \right]$$

The author is not fond of using formulas whenever their use can be avoided. The above formulas express in the simplest mathematical form the principles which have been so far explained and used in the calculations, but it is strongly urged that the formulas be kept "for exhibition purposes only," and that when any specific case is to be worked it be attacked from the standpoint of the principles involved, as explained in the case worked. In other words, if one understands properly and thoroughly the basic principles, he has no need of the formula; if one does not understand the principles, the formula had better be kept forever in "innocuous desuetude."

The total head, obtained as above, is the theoretical head. It is like the pressure on the piston of a locomotive—the total available force for all purposes. Just as the pressure on the locomotive piston is used up in friction in the engine and in moving the engine itself, and the residue is the *available* pull on the draw bar which moves the train, so the total head of the chimney is partly used up in friction in the chimney itself, partly in giving velocity to the gases as they pass out of the chimney, and the residue is the *available* head which draws or pulls the gases through fire grates, furnaces and flues up to the base of the chimney. If the chimney could be momentarily completely closed at the bottom, except for the gauge opening, and the air inside be brought to rest, the gauge would show the total head; as soon as dampers are opened connecting the flues, air moves up the chimney, and the gauge pressure is lessened by the head required to move the gases and that absorbed in their friction in the chimney.

Head Represented in Velocity of Issuing Gases.—This item always exists when the chimney is working, and depends only on the velocity of the gases as they escape and their temperature. The hydraulic head necessary to give any fluid a velocity V is simply the same as the height which a falling body must fall in order to acquire that same velocity; *i. e.*:

$$h = \frac{V^2}{2g}$$

in which expression, 2g is the constant acceleration of gravity, 19.6 meters or 64.3 feet, and the velocity is in meters or feet per second. If we know, therefore, the velocity of the gases issuing from the chimney, or can calculate or assume it, we can get h. In practice the velocity does not vary within very wide limits. In small house chimneys it may not exceed 3 feet per second, in boiler chimneys 6 to 12 feet per second, in furnace chimneys 12 to 20 feet per second. The temperatures of these issuing gases is, moreover

	C°	F°
In small chimneys.....	100 to 200	200° — 350°
In boiler chimneys.....	100 to 300	200° — 550°
In furnace chimneys....	300 to 1000	550° — 1800°

If the chimney in question has, therefore, a known velocity of exit of its gases, h can be calculated; but it must not be forgotten that h will be in terms of the kind of gases which is escaping; *i. e.*, of hot gas, and to subtract it from or compare it with h_a we must reduce it to its equivalent head in terms of cold gas. This is merely a matter of taking into account the specific gravities or relative densities of hot gas and cold air, which are inversely proportional to their absolute temperatures; that is, if D represents the relative density of air and chimney gases at the same temperature:

$$h_a \text{ rel} = h \times \frac{D}{1 + a\,t} = \frac{V^2}{2g} \cdot \frac{D}{1 + a\,t}$$

Illustration: Assuming the actual velocity of the gases issuing from a furnace chimney to be 15 feet per second, and their temperature 500° F., density 1.06 (air = 1), what will be the head represented by the velocity of these gases in terms of cold air at 32° F?

The head represented, *in terms of hot gases at* 500° F., is

$$h = \frac{V^2}{2g} = \frac{15^2}{64.3} = 3.5 \text{ feet.}$$

In terms of air at 500° F. is

$$3.5 \times 1.06 = 3.71 \text{ feet.}$$

And in terms of air at 32° F.,

$$3.71 \times \frac{491}{500 - 32 + 491} = 1.9 \text{ feet.}$$

Out of the total head which this chimney produces (say 36.5 feet) 1.9 feet is represented by the velocity of the issuing gases, or 5.2 per cent of the whole, leaving 34.6 feet to represent loss by friction in the chimney and the available head. We will proceed to discuss the loss of head due to friction in the chimney.

Head Lost in Friction in the Chimney.—This varies with the smoothness or roughness of the walls, and has been determined experimentally for air moving with different velocities. The manner of expressing the friction loss is, to put it as a function of the head necessary to give the gases their actual velocity, assuming there were no friction. Thus, supposing as in the preceding paragraph, the actual velocity of the hot

gases is 15 feet per second, and the head (in terms of cold air) necessary to give that velocity, not considering friction, is 1.9 feet, then the head lost in friction in getting up this velocity will be

$$h_{\text{friction}} = 1.9 \times \frac{H}{d} K.$$

That is, it will be proportional to H, the height of the chimney, inversely as d, the diameter or side, if square, and to a coefficient K, determined by experiment. The latter varies, according to Grashof's experiments, between 0.05 for a smooth interior to 0.12 for a rough one, and averages 0.08.

Illustrations: Assuming the height of the chimney 100 feet, its section to be 6 feet square, the coefficient of friction K = 0.08, and the head represented by the net velocity of the hot gases in the chimney to be 1.9 feet of cold air, what is the head lost in friction in the chimney?

The ratio of height to side is 100 ÷ 6 = 16.67, which multiplied by K gives 1.33 as the value of the function containing these three terms. This means that 1.33 times as much head has been lost in friction as is represented by the net actual velocity of the gases as they pass up the chimney. Therefore,

$$h_{\text{friction}} = 1.9 \times 1.33 = 2.5 \text{ feet cold air.}$$

Another way of looking at this, which is sometimes useful in considering the height of a chimney, is to say

$$h_{\text{friction}} = 0.025 \text{ H.}$$

Or, that in this case, the head lost in friction amounts numerically to one-fortieth the height of the chimney.

If we subtract the head lost in friction plus that represented in the net velocity of the gases, from the total gross head, the residue is that available for doing work external to the chimney. In the specific case of the preceding illustrations we have

h_o = total head	= 36.5 feet	=	100 per cent
h_{velocity} = velocity head	= 1.9 "	=	5 "
h_{friction} = friction in chimney	= 2.5 "	=	7 "
$h_{\text{available}}$ = available head	= 32.1 "	=	88 "

Available Head of a Chimney.—This is the part of the total head which remains after subtracting the head lost in friction in the chimney and that represented by the velocity of the issuing gases. In the specific cases considered in the above illustrations, the net available head amounted to 88 per cent of the whole theoretical head. If we assume limiting conditions as found in practice, we can find the limiting values of this proportion. Calling the cases I and II, those with minimum and maximum absorption of head in the chimney itself, we have

	Case I.	*Case II*
Temperature of issuing gases	100° C.	1000° C.
Velocity of issuing gases per second	1 meter	7 meters
Ratio H to d	10	50
Coefficient K	0.05	0.12
Specific gravity of gases (air = 1)	1.00	1.06
Head as velocity of gases (meters of air)	0.04 m.	0.56 m.
Head as velocity of gases (feet or air)	0.13	1.87
Head absorbed in friction (meters of air)	0.02	3.36
Head absorbed in friction (feet of air)	0.07	11.2
Head used up in chimney (meters)	0.06 to	3.92
Head used up in chimney (feet)	0.20 to	13.0
Water gauge pressure thus lost, m. m	0.1 to	5.0
Water gauge pressure thus lost, inches	0.0031 to	0.2

The available head will, therefore, be the theoretical total head minus a loss in the chimney itself, which may amount to a maximum of 3.9 meters or 13 feet, representing an absorption of water gauge pressure up to 5 millimeters, or 0.2 inch at a maximum. Under ordinary conditions half these quantities would be a rather high chimney loss

In most conditions which confront the metallurgist, the question is to determine how high a chimney should be built in order to supply a certain draft, and that found to be necessary in practice to be necessary. For instance, to burn a certain amount of coal per hour in any grate requires a certain amount of draft. This amount is increased if the fuel is increased, and [illegible] In boilers, 18 pounds of coal burned per square foot of grate surface per hour is [illegible] practice, and requires a draft of 0.4 inch to 0.5 inch of water gauge, according to the kind of coal burned. In furnaces where the amount of coal burned is greater per hour there will be usually a correspondingly greater [illegible] in the chimney. To calculate the height of chimney required it is necessary to assume one, the temperature of the [illegible] the available draft required and an average chimney [illegible]

Problem 19.

It is desired to design a chimney for a puddling furnace, the grate of which is 4 feet by 6 feet, and which is [illegible] pounds of bituminous coal per hour per square foot of grate surface. Temperature of gases entering the chimney 1200° C., at the top probably 1000° C. Specific gravity of gases (air = 1). Draft required 0.6 inch of water gauge. Outside temperature 30° C.

Solution.—We can assume that since the gases will be at an average temperature of 1100° C. in the chimney, their velocity will be high, and that at least 0.1 inch of water gauge pressure will be absorbed by the chimney itself. This makes a total requirement of 0.7 inches of water for total head, or

$$h_o = 0.7 \times 77.2 \times 12 = 45 \text{ feet of cold air.}$$

Or an unbalanced pressure or ascensive force of

$$45 \times 1.293 \times 10 = 3.64 \text{ pounds per square foot.}$$

Considering the air outside the chimney, its weight at 30° C. is equal, per cubic foot,

$$1.293 \times \frac{273}{303} \times 10 = 0.073 \text{ pounds}$$

The gases inside the chimney weigh, per cubic foot,

$$1.293 \times 1.03 \times \frac{273}{1100 + 273} = 0.0806 \text{ pounds}$$

The height of the chimney being called H and its cross-section S, the volume is H × S, and the weight of hot gas inside it is

$$(H \times S) \times 0.0806 \text{ pounds.}$$

And of an equal volume of cold air outside

$$(H \times S) \times 0.073 \text{ pounds,}$$

giving a total ascensive force of

$$(H \times S) \times 0.0564 \text{ pounds}$$

But there is needed a total ascensive force of

$$S \times 3.64 \text{ pounds,}$$

in order to give the pull of 3.64 pounds per square foot, etc., therefore, of necessity,

$$H \times S \times 0.0564 = S \times 3.64,$$

from which

$$H = \frac{3.64}{0.0564} = 64.5 \text{ feet.}$$

Concerning the cross-section of this chimney, it would [illegible] be safe to make it less in diameter than one-tenth [illegible] its height, because of lack of stability; in fact, one-twenty-[illegible] would be better practice. This consideration would make the internal diameter 2 ft. 7 inches, and 5.2 square feet. Another way of arriving at a diameter is to calculate the volume of the hot gases which must pass up the chimney, and provide for them some maximum velocity in the chimney [illegible] as we say, 6 meters (20 feet) per second, and [illegible] area necessary for filling this condition as [illegible]

Coal burnt per hour $4 \times 6 \times 30$ = 720 pounds.
Air theoretically necessary, assuming average bituminous coal (See Prob. 1) $= 123 \times 720$ = 88,560 cubic feet
Products of combustion at standard conditions $= 129 \times 720$ = 92,880 "
Volume chimney gases at 1100° C. $= 92{,}880 \times \frac{1700 + 273}{273}$ = 467,100 "
Volume per second = 130 "
Area of chimney, if maximum velocity is 20 feet per second = 6.5 square feet
Diameter, if round = 2 ft. 10 ins.

This chimney would do its work better, and there would be much less loss in friction, if the internal diameter were made 25 per cent greater than the above calculated minimum, say, therefore, 3 feet 6 inches, making the area nearly 50 per cent greater, and cutting down the velocity in the chimney to 13.5 feet per second.

Problem 20.

In the case of the puddling furnace of Problem 19, assume that the hot gases, instead of going directly into the chimney, are passed through the flues of a boiler placed above the furnace, and thence pass into the chimney at a point 15 feet higher than before. Assume chimney 3 feet 6 inches internal diameter, 64.5 feet high above the furnace flue, and that the gases now passing into it 15 feet higher up are at 350° C., and cool to 250° C. at the top of the chimney. The boiler flues introduce additional frictional resistance equal to 0.1 inch of water. The boiler raises steam at a net efficiency of 45 per cent, the steam engine utilizes the steam at a mechanical efficiency of 20 per cent, and a centrifugal fan supplies the forced draft needed at a mechanical efficiency of 25 per cent.

Required: (1) The total head of the chimney, when the furnace discharged directly into it, and the average temperature of the gases in it was 1100° C., and specific gravity 1.03 (air = 1).

(2) The head absorbed as velocity of the outgoing gases, their temperature being 1000° C.

(3) The head lost in friction in the chimney, in this case.

(4) The head which was available to run the puddling furnace.

(5) The total head of the chimney with the gases entering 15 feet above former flue, and average temperature 300° C.

(6) The head absorbed in this case as velocity of outgoing gases, their temperature being 250° C.

(7) The head lost in friction in the chimney in this case.

(8) The available head to draw gases into the chimney.

(9) The deficit of head which must be made up by forced blast under the grate of puddling furnace.

(10) The horse-power absorbed by the fan which furnishes this blast.

(11) The horse-power furnished by the engine using the steam from the boiler.

(12) The excess of power which is thus saved and available for other purposes.

Solution:

(1) Volume of gases in chimney,
$64.5 \times 3.5 \times 3.5 \times 0.7854 = 620.5$ cu. ft.
Weight at 32° F. (0° C.),
$620.5 \times (1.293 \div 16) \times 1.03 = 51.65$ lbs.
Weight if temperature is 1100° C.,
$51.65 \times \frac{273}{1100 + 273} = 10.27$ lbs.
Weight of equal volume of air outside at 30° C.,
$620.5 \times (1.293 \div 16) \times \frac{273}{30 + 273} = 45.18$ lbs.
Difference of weight = ascensive force,
$45.18 - 10.27 = 34.91$ lbs.
Ascensive force per square foot,
$34.91 \div 9.62 = 3.63$ lbs.
Total head in terms of cold air at 0° C.,
$3.63 \div (1.293 \div 16) = \mathbf{44.9}$ ft. (1)
In terms of water gauge pressure,
$44.9 \times 12 \div 772 = \mathbf{0.685}$ ins. (1)

(2) Volume of gases per hour at 0° C.,
(Prob. 19) = 92,800 cu. ft.
Volume at 1000° C.,
$= 92{,}880 \times \frac{1000 + 273}{273} = 433{,}100$ cu. ft.
Velocity per second,
$433{,}100 \div (3600) \div 9.62 = 12.50$ ft.
Head necessary to give this velocity, in terms of hot gases, at 1000° $= (12.50)^2 \div 64.3\ (2g) = 2.43$ ft.
In terms of gases at 0° C.,
$2.43 \times \frac{273}{1000 + 273} = 0.52$ ft.
In terms of air at 0° C.,
$0.52 \times 1.03 = \mathbf{0.55}$ ft. (2)
In terms of water gauge pressure,
$0.55 \times 12 \div 772 = \mathbf{0.008}$ in. (2)

(3) Assuming K, the coefficient of friction, 0.08, then
$$h_f \text{ friction} = \frac{V^2}{2g} \cdot \frac{273}{273 + t} \cdot \frac{H}{d} K \cdot D.$$
This is only an abbreviated form of the operations done under (2), adding the terms which account for the height, diameter and friction. Now, the velocity per second:
$$V = 92{,}880 \times \frac{1100 + 273}{273} \div 3600 \div 9.62 = 13.5 \text{ ft.}$$
Head necessary to give this velocity in terms of air at 0°,
$$(13.5)^2 \div 64.3 \times \frac{273}{273 + 1100} \times 1.03 = 0.58 \text{ ft.}$$
Proportion of this velocity head lost in friction $= \frac{H}{d} K = \frac{64.5}{3.5} \times 0.08 = 1.47$
Head lost in friction in chimney,
$0.58 \div 1.47 = \mathbf{0.85}$ ft. (3)
In terms of water gauge pressure,
$0.85 \times 12 \div 772 = \mathbf{0.013}$ in. (3)

(4)	*Cold Air.*	*Water Gauge.*
Total head	44.90 feet	0.685 inch
Absorbed in velocity of gases....	0.55 "	0.008 "
Absorbed in friction in chimney..	0.85 "	0.013 "
Available for the furnace.....	43.50 "	0.664 " (4)

(5) Volume of chimney gases,
$(64.5 - 15) \times 3.5 \times 3.5 \times 0.7854 = 475.7$ cu. ft.
Weight at 300° C., specific gravity 1.03 (air = 1),
$$475.7 \times (1.293 \div 16) \times 1.03 \times \frac{273}{273 + 300} = 18.86 \text{ lbs.}$$
Weight of equal volume of outside air at 30° C.,
$$475.7 \times (1.293 \div 16) \times \frac{273}{273 + 30} = 34.68 \text{ lbs.}$$
Ascensive force of air per square foot,
$(34.68 - 18.80) \div 9.62 = 1.64$ lbs.

Total head in terms of cold air,

$$1.04 \times 1.293 \div 1.60 = 20.3 \text{ ft.} \quad (5)$$

In terms of water gauge pressure,

$$20.3 \times 1.2 \div 772 = 0.32 \text{ in.} \quad (5)$$

(6) Velocity of issuing gases, per second, at 250° C.

$$92{,}880 \times \frac{250 + 273}{273} \div 3{,}600 \div 9.62 = 5.14 \text{ ft.}$$

Head as velocity in terms of cold air at 0° C.,

$$(5.14)^2 \div 64.3 \times \frac{273}{273 + 250} = 1.03 = 0.22 \text{ ft.}$$

In terms of water gauge pressure = 0.003 in.

(7) Average velocity of gases in chimney at 300° C.,

$$92{,}880 \div 3{,}600 \times \frac{300 + 273}{273} = 9.62 = 5.63 \text{ ft.}$$

Head lost in friction in terms of cold air,

$$(5.63)^2 \div 64.3 \times \frac{273}{300 + 273} \times 1.03 \times \frac{40.5}{35} \times 0.08 = 0.27 \text{ ft.} \quad (7)$$

In terms of water gauge pressure = 0.004 in. (7)

(8)

	Cold air	Water gauge.
Total head	20.30 feet	0.320 inch
Absorbed in velocity of gases	0.22 "	0.003 "
Absorbed in friction in chimney	0.27 "	0.004 "
Available to draw gases through boiler	19.81 "	0.313 " (9)
Available head needed for puddling (4)	43.50 "	0.664 "
Available head needed for boiler	6.43 "	0.100 "
Total head needed for both	49.93 "	0.740 "
Available head from chimney (8)	19.81 "	0.313 "
Deficit, to be supplied by blast	27.12 "	0.451 " (9)

(10) The 0.451 inches of water gauge equals

$$(0.451 \div 12) \times 62.5 = 2.35 \text{ lbs. per sq. ft.}$$

The volume of air to be supplied is, at 30° C.,

$$88{,}560 \text{ (Prob. 19)} \div 60 \times \frac{273 + 30}{273} = 1{,}638 \text{ cu. ft. per min.}$$

Net work done by the fan,

$$1{,}638 \times 2.35 = 3{,}850 \text{ ft. lbs. per min.}$$

Gross power needed by the fan,

$$3{,}850 \div 0.25 \text{ (efficiency)} = 15{,}400 \text{ ft. lbs. per min.}$$

Horse-power needed to drive the fan,

$$15{,}400 \div 33{,}000 = 0.47 \text{ H. P.} \quad (10)$$

(11) The boiler receives the gases at 1200° C., and discharges them at 350°, and 92,880 cubic feet of gases (measured at standard conditions) pass through per hour. The composition of these gases is not given, but from the specific gravity we might conclude that they contain on an average 10 per cent of carbon dioxide, since if they contained the maximum amount of that gas (about 20 per cent) their specific gravity would be 1.06 (air = 1). Assuming them, therefore, to contain

CO^2	10 per cent
H^2O	10 "
CO, N, O	80 "

their heat capacity per degree per cubic foot would be, between 350° and 1200°

		Oz. Cal.
CO^2	$0.1 \times [0.37 + 0.00022 \quad (350 + 1200)]$	0.0870
H^2O	$0.1 \times [0.34 + 0.00015 \quad (350 + 1200)]$	0.0773
CO, N, O	$0.8 \times [0.3 + 0.00017 \quad (350 + 1200)]$	0.2310
	Sum	0.3953

Heat given up per cubic foot

$$0.3953 \times 1200 = 474.36 \text{ oz. cal.}$$

Heat given up by gases per hour to boiler

$$92{,}880 \times 474.36 = 44{,}060{,}000 \text{ oz. cal.}$$

Heat in the same gases per hour

$$92{,}880 \times 0.3953 \times 350 = 12{,}840{,}000 \text{ oz. cal.}$$

Heat equivalent of mechanical power developed by boiler,

$$44{,}060{,}000 - 12{,}840{,}000 = 31{,}220{,}000 \text{ oz. cal.}$$

$$= 1{,}460 \text{ lb. cal.}$$

Horse-power generated by the engine,

$$1{,}460 \times 60 \div 1{,}400 = 46.8 \text{ H. P.}$$

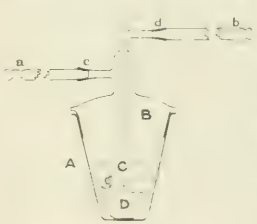
(12) Net available power after supplying fan,

$$46.8 - 0.5 = 46.3 \text{ H. P.}$$

The Determination of Carbon in Ferrochrome and the Eimer Carbon Crucible.

By C. Offerhaus, Ph.D.

In the determination of the carbon contained in pig iron and ferro-alloys we distinguish between direct and indirect methods. In the latter the carbon is first isolated by [illegible] and is then determined by [illegible]. Owing to the very low solubility, only the direct methods in the dry way and the indirect methods in wet operation are available in this case. In the latter methods the carbon is [illegible] by [illegible].



CARBON COMBUSTION CRUCIBLE.

A, [illegible] crucible.
B, [illegible].
C, Glass tubes.
a, [illegible].
b, Rubber [illegible].
c, Perforated [illegible].
d, Porcelain crucible.

In what follows the [illegible] and Wöhler-Heräus methods. A short [illegible] by gravimetric [illegible].

Eimer & Amend.

Weighed portions of the [illegible] and placed in [illegible] of a current of dry hydrogen, the latter is reduced to [illegible] and then [illegible].

The [illegible] Analysis by [illegible] Sulphur [illegible].

In Germany, Thus Ledebur, in his well-known "Leitfaden für Eisenhüttenlaboratorien," only the latter method for ferrochrome.

'actly according to the instructions of Ledebur in his "Leitfaden fur Eisenhutten-Laboratorien." The chlorine was produced, according to Graebe, from potassium permanganate and concentrated hydrochloric acid. Besides, a combustion tube with incandescent pieces of charcoal was employed in connection with this method.

The isolated carbon was oxidized either according to Sarnström in a chromic acid solution or, mixed with copper oxide, in an oxygen current. Both methods of combustion gave the same figures.

When the chlorination was carried out in such a way as to remove all the chloride of chromium by sublimation (so that reduction by a current of hydrogen was superfluous), the results remained unchanged.

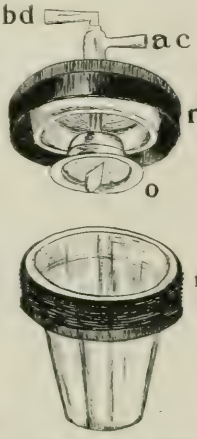


FIG. 1b.—DETAILS OF CRUCIBLE.

m, n. Couplings.
o.—Receptacle for copper oxide.
b d and a c—The same as in Fig. 1a.

Direct Methods

The principal conditions for a successful use of direct methods are a very finely powdered material and sufficiently high temperature. The ferrochrome was crushed and passed through a screen of 100 meshes per 1 inch, and was then ground for 1 hour in small quantities (1½ grams) in a mechanical agate mortar. The powder then no longer shows any metallic lustre.

One-half gram of this extremely fine metallic powder was mixed with 8 or 10 times as much fused and very finely ground lead chromate. The mixture was placed in a porcelain or copper boat, and was then introduced into a porcelain tube containing copper oxide. As combustion furnace, a Fletcher blast furnace was used, which gives a sufficiently high temperature. Combustion was carried out in a dry current of oxygen free from carbonic acid and free from hydrogen.

Although the combustion is in general completed in half an hour, the operation was not stopped before 1 hour, to be on the safe side. In all other details the combustion was carried out as usually.

It was found that the degree of fineness obtained with the treatment described above was sufficient. Another sample which was ground for 3 hours gave the same result. If lead oxide was substituted for lead chromate, the results remained unchanged.

Table I gives a summary of the results obtained by the two methods with one and the same sample of 70 per cent ferrochrome.

TABLE I.

Wöchler's Chlorine Method.

1. The carbon is incidentally removing with the carbon reduced and removed by solution in water and the carbon is oxidized.

	Per Cent C.
(a) [illegible]	[illegible]
(b) [illegible]	[illegible]
2. All [illegible] was removed by solution. The carbon was freed from the adhering [illegible]	
	Per Cent C.
[illegible]	[illegible]
[illegible]	6.45

Direct Methods.

	Per Cent C.
1. The material ground for 1 hour, mixed with lead chromate, gave	
(a) mixed with lead chromate	6.70
(b) mixed with lead oxide	6.72
2. The material ground for 3 hours, mixed with lead chromate, gave	6.79

Table I shows that with ferrochrome containing about 6.5 per cent C., the direct methods give 0.3 per cent more carbon. The direct methods also give higher values of C. than the indirect methods in steel and iron analysis (see for instance, R. Lorenz, *Zeit. f. Angewandte Chemie*, 1893, p. 313, 395, 411).

The direct methods are carried out more quickly than the indirect ones and ought to be considered as the more accurate ones, just for the reason that they give the higher figures. For this reason these methods are now recommended from competent authorities for iron and steel analysis (see *Transactions Amer. Inst. Mining Engineers*, 1905, p. 289).

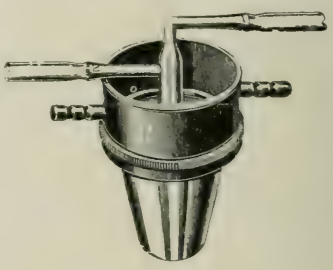


FIG. 2.—CRUCIBLE CLOSED WITH THE GLASS TUBES INSERTED AND WATER COOLER.

We have noticed above that a sufficiently high temperature is a main condition for direct dry combustion. If the porcelain tube is replaced by a platinum tube, it becomes easier to attain the necessary temperature. But the high temperature is required only at the point of combustion proper.

For this reason, the combustion furnace with porcelain or platinum tube was replaced by P. W. Shimer[2] by a platinum crucible with a copper stopper of special construction and by a brass tube with copper oxide.

The essential feature of this crucible is the water-cooled stopper for the protection of the rubber fittings and connections. Comparative determinations of the total carbon in pig iron and steel with this crucible applying an air current on one hand, and with a porcelain or platinum tube applying an oxygen current on the other hand, give a good agreement ac-

FIG. 3.—COMPLETE ABSORPTION TRAIN WITH CRUCIBLE AND BURNER IN PLACE.

a—Air or oxygen inlet, n, sealing bulbs, o, drying tower, p, Finer combustion crucible, r, connecting tube, s, water-cooled lead tube, u, drying tube, q, connecting bulbs, t, safety-tube.

cording to his statements.[3] He does not apply any correction for the flame gases diffusing through the platinum crucible.

A. Emmet goes a step further. He places the copper oxide into the crucible and avoids rubber fittings and rubber connections by carefully grinding and fitting, respectively sealing, the platinum and glass connections.

Water-cooling and the copper oxide tube thereby become superfluous.

The Emmet carbon combustion crucible is therefore a substitute for both combustion furnace and combustion tube; it

[illegible]

combines in one apparatus the functions of the two, and has all advantages of the Shimer crucible to an increased degree.

Since a description of the Eimer crucible has not yet appeared in the chemical literature, it may be given here with reference to Fig. 4. All other figures will be found self-explanatory.

The crucible *A* proper is distinguished from an ordinary

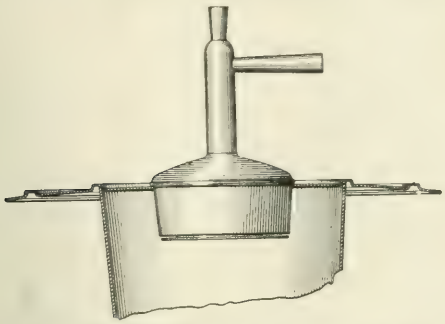


FIG. 4.—CRUCIBLE TOP TAKEN OUT AND USED WITH A PLATINUM DISC ON PLATINUM DISH FOR THE COMBUSTION OF LARGE QUANTITIES OF ORGANIC MATTER IN THE AGRICULTURAL LABORATORY.

platinum crucible by its conical form[1] and by its flat cut rim; it may replace at any time an ordinary platinum crucible.

The stopper *B* which may be made partly of copper, is carefully ground or coupled, so as to fit the crucible air-tight, and the same is the case with the glass tubes *c* and *b*, ground or fused into the platinum or copper tubes *c* and *d*.

The oxygen enters at *a*. It passes along the hot wall of the crucible and is thus preheated when it reaches the material to be oxidized.

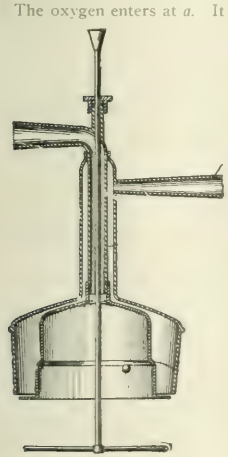


FIG. 5.—CRUCIBLE TOP PROVIDED WITH STIRRING ARRANGEMENT.

The gases of combustion pass out at *b*, after having passed the bell-shaped receptacle *C* placed near the bottom of the crucible and previously filled with sifted granular or wire-form copper oxide. To protect the upper parts, the crucible is set in a special perforated asbestos plate, about 5 inches square.

An essential advantage of the Eimer crucible is the absence of any organic substance, like rubber bands or rubber tubing. The rubber connections, and especially the rubber fittings of the Shimer crucible, cause a feeling of uneasiness. There exists always the danger that a small piece of the rubber, which is and by hardens, drops down when the top is placed on the crucible. The determination of total carbon in ferrochrome was carried out in the Eimer crucible as follows:

0.2 gram of very finely powdered metal with 2 grams of fused and very finely ground lead chromate were mixed in a small porcelain crucible with thin walls, and this was then placed into the platinum crucible. Combustion was carried out in a current of dry oxygen, free from hydrogen and carbon dioxide. A blast lamp was used, and five hours proved sufficient.

Although in this case the combustion is general complete

[1] The same form is also used for Gooch crucibles. See Blair, The Chemical Analysis of Iron, 5th edition.

in half an hour, the operation was carried out for a full hour.

The platinum into the crucible. special experiments, carried out under exactly alike conditions, result.

The good agreement of the results of combustion in the Eimer crucible, also applying the correction, with those in the porcelain tube, render it improbable that fusing from the inside of the crucible to the

The sample of 70 per cent ferrochrome, which gave, in the porcelain tube, an average of 7.74 per cent C., showed, when determined in the Eimer crucible with the application of the above correction, an amount of 7.78 per cent C.

With another sample of 70 per cent ferrochrome, the determination in the Eimer crucible gave 9.00 per cent C. (Messrs. Booth, Garrett and Blair found 9.10 per cent in the same sample).

My conclusion is that for the analysis of ferrochrome the chlorine method of Wöhler is not to be recommended, as it gives too low values for C. Therefore, the direct method is preferable, and the Eimer crucible may be used here to good advantage.

HOLCOMB'S ROCK, VA.

According to a private communication from the chemical laboratory of Messrs. Booth, Garrett and Blair, the presence of a small amount of finely ground [illegible] in the [illegible] causes a low result. [illegible] CARBON.

Bosses and Individualism in the Development of Chemistry.

Prof. Wilhelm Ostwald recently presented in Boston a suggestive address on the development of chemistry in France, England, Germany and the United States.

Prof. Ostwald pointed out that chemistry had its earliest developments in France. Owing to the [illegible] of Napoleon I., science in France has always been imperialistic in its tendencies. There [illegible] Paris, who played the role of "king in chemistry." The succession was Lavoisier, Fourcrois, Berthollet [illegible] Dumas, Wurtz and Berthelot, the present [illegible] already elected the next "king." The result has been to greatly retard the advance of the science.

The opposite conditions have existed in England, where individualism has [illegible] Davy, Faraday and others were [illegible] government, and had to [illegible] support themselves.

In Germany, which consisted of thirty-six separate different [illegible] having the [illegible] large number of centers of science and independent thinkers. At first Germany was far behind France and England. [illegible] carried on in Germany, all of which is attributed to [illegible] methods.

In America, Prof. Ostwald said the [illegible] has been [illegible] and [illegible] has been [illegible] [illegible] [illegible]

Earth Alkali and Allied Peroxides.

In our Vol. III., pp. 255 and 376, we have already dealt with the new series of peroxides which have been placed on the market last year by Roessler & Hasslacher Chemical Co. A special phase of the subject was dealt with in a recent paper by Dr. Richard von Foregger and Mr. Herbert Philipp presented before the New York Section of the Society of Chemical Industry on Jan. 19. The paper dealt with the properties and applications of the following earth alkali and allied peroxides, namely, the peroxides of calcium, strontium, magnesium and zinc.

As oxidizers, they come next to ozone, since after they have liberated their available oxygen, either a colorless solid residue is left which can be easily saturated from the oxidized medium, or a soluble salt—both of a harmless nature.

The constitution of the peroxides is explained by assuming a tetravalent O, thus:

$$R^{II} = O^{IV} = O^{II}$$

The O^{IV} is represented as being slightly wedged in between the R^{II} and O^{II} of the ordinary oxide, ready to escape as the opportunity is provided, thus explaining the action of the available O.

Calcium peroxide as a commercial product is the dehydrated product containing 60 per cent CaO_2 with 13.5 per cent available oxygen, but one containing 80 per cent CaO_2 with 17.8 per cent available oxygen has also been obtained. The specific gravity varies; that of the 60 per cent product is 0.603 and that of the 80 per cent product is 0.74. All the peroxides are practically insoluble in water, but they slowly dissociate, which is thus explained: CaO_2, $2H_2O$ is first formed as a solid, which goes into solution as $Ca(OH)_2$, H_2O_2 and this gives off its H_2O_2, these reactions being reversible according to whether the temperature is high or low.

A comparison of calcium permanganate with Ca peroxide is made, concluding a discussion which was brought up at the Bethlehem meeting of the American Electrochemical Society (see Electrochemical and Metallurgical Industry, 1905, Vol. III., p. 377). The permanganate $CaMnO_4 + 5$ aq. gives 13.2 per cent O in alkaline solution and 21.7 per cent O in acid solution against 13.5 per cent of 60 per cent CaO_2. Ca permanganate is, however, of a deliquescent nature, and dangerous on account of its unstability. CaO_2, like the other peroxides of the series, is comparatively stable in dry atmospheres, and can be heated up to 150° C. and 170° C. without any decomposition taking place. The fact that calcium permanganate decomposes so very easily in the presence of organic substances and that it is hygroscopic, places a barrier to its industrial success, though it could be manufactured very cheaply as a by-product of Weldon slimes. It has only been used for purifying water.

Interesting about strontium peroxide is the dissociability of its O in water, nearly 15 per cent of its available O was titrated in solution at 20° C. Much of what applies to these also applies to magnesium and zinc peroxides.

Calcium peroxide seems destined to be largely used for industrial purposes, on account of its large quantity of available O; magnesium, strontium and zinc peroxides are chiefly of medical interest on account of their non-toxic, non-irritant, physiological and chemical reactions.

The authors described a method by which they have been able to bleach cotton oil in 10 minutes. With the use of calcium peroxide as bleaching agent, there is no danger whatever in handling this peroxide, nor does any saponification take place. With a 60 per cent calcium peroxide the operation is as follows: Add, while stirring the oil, about 1-10 to ⅛ per cent in weight of powdered peroxide. Following this, slowly add 1.6 to 1.5 per cent of sulphuric acid, 66° Be. The reaction is visible, the oil which otherwise would turn black by the addition of sulphuric acid, becomes light at once; when the addition of acid is finished, the bleach is also finished, so that the whole process only lasts about 10 minutes. Filtration can then take place right away. The economy of the process depends greatly on the finely powdered state of the calcium peroxide.

Since oxygen is the most logical disinfectant, and since the germicidal power of active oxygen has been established, the question suggests itself whether some of these non-toxic peroxides are not very useful means with which to sterilize and preserve. The assumption is justified by the harmlessness which is a property of these products. Calcium and magnesium are no foreign substances to water, milk, beer or any food substance, consequently their introduction in the form of peroxides cannot be harmful to our system.

The authors gave a preliminary report on a method of using peroxides as sterilizers. One experiment which they described was interesting. It related to the prevention of sweet cider from fermenting. Raw apple juice treated with calcium peroxide can be made to ferment at an earlier date when in contact with air than it would under ordinary circumstances, and the increased amount of peroxide apparently only accelerates the fermentation.

On the other hand, 350 cc. raw juice as it came from the press was treated with a mixture of 0.1 gr. calcium peroxide of 60 per cent CaO_2 and 0.2 gr. magnesium peroxide of only 15 per cent MgO_2. The liquid was kept in a bottle which was hermetically sealed immediately after the addition of the peroxides, and shaken sufficiently to sterilize the bottle. The first effect is a brownish coloration of the cider, which, however, begins to fade after two weeks, and after another couple of weeks the impurities begin to settle and the original color, but somewhat brighter, is restored. No fermentation as yet, after nearly four months, has taken place. The authors gave a tentative explanation of these facts.

Magnesium peroxide was mentioned by the authors as the ideal means for sterilizing drinking water—not on a large scale, but for the use of the soldier in the field, the workman in tropical countries and, sometimes, the citizen in regions where the water is known to be typhoid contaminated. For this purpose the authors proposed simple tablets, similar, for instance, to effervescent lithia tablets, but in the present case being effervescent magnesium tablets, consisting of magnesium peroxide and citric acid.

Finally, the authors suggested the sterilization of milk by small quantities of calcium peroxide.

Notes on Electrochemistry and Metallurgy in Great Britain

(From Our Special Correspondent.)

Presidential Addresses and Electrochemistry.

In my first letter of this series, written nearly a year ago, I chronicled the items of chief interest to electrochemists in the various presidential addresses delivered at the opening of last winter's sessions of the various scientific institutions. Of this year's addresses, Mr. John Gavey, the new president of the Institution of Electrical Engineers, confined his address to his special sphere of telegraphy and telephony, and did not, therefore, touch upon electrochemical or electrometallurgical processes. Passing reference to electrothermal processes of iron and steel production was, however, made by Mr. L. S. Pearce in his inaugural address as chairman of the Manchester Local Section of the Institution of Electrical Engineers, who followed orthodox views in his presentation of the case, as may be instanced from the following quotation: "The manufacture of pig iron in the electric furnace is of little interest to electrical engineers in this country, as with the cheap supply of fuel available here no electric furnace can compete with a modern blast furnace. It is not improbable that the electrical energy required for the electric furnaces will be ob-

tained from gas engines operated by gases from the blast furnaces. Probably only two of the modern electric furnaces are capable of adoption for English steel smelting, but it has been proved that either of these processes can compete successfully with the Sheffield crucible furnace with electrical energy at anything like a reasonable figure."

Dr. W. M. Thornton's inaugural address to the Newcastle Local Section of the Institution of Electrical Engineers, while in no way electrochemical in its title, is of such striking interest in two of its sections that a reference to them will not be inappropriate. Dealing in the first instance with the value of research as an investment, Dr. Thornton said: "There is speculative search into the unknown on lines suggested by unique chains of thought, arising from new observations or strong co-ordinated thinking. Work of this kind cannot be undertaken to order. There are times of inventive activity in which ideas come rapidly. Then follows the patient experimental search for truth where so many fall out. Now, if it is to come at all, a time of speculative activity, almost inevitable, follows the completion of a university course, and, it may be, the taking of a degree. It is then encouragement and, perhaps, pecuniary help that are of vital importance to a young student. He should have the opportunity of seeing how to carry out ideas by watching or helping in the research work of his professor, and the advantage is mutual. The enthusiasm of a pupil is the most refreshing thing that can come into the life of a teacher, and the friendship or sympathy of the elder man is always greatly prized by the younger. After all, in the light of the immense unknown, the difference between teacher and taught is very small. It is only the case of one student a little in advance helping another along the same road."

Dr. Thornton also appealed to manufacturers to give a freer hand to young engineers, and suggested that those of ability and promise should be drafted into the confidential department, design or test rooms, and set to work on new things. "They say no man becomes a good glass-blower until he has broken 10 pounds of glass in the process of learning. There should also be some similar allowance made for crude ideas in engineering, say £10 sterling. Fuses cover a multitude of sins in the test room, and it would take a good deal to reach this limit. One result of college training is to produce a large crop of ideas, more or less crude, out of which the finished articles may emerge."

As regards the relation of research and practice, a case was instanced of what I may perhaps call "the suggestiveness of parallelism." Some years ago in carrying out some bacteriological experiments, Dr. Thornton noticed, as others had done previously, that alternating currents caused bacteria to place themselves parallel to the lines of flow. Differing from other investigators, Dr. Thornton sought parallels in other fields, and found that the movement depended entirely upon the relative conductivity of the organism and of the surrounding liquid, so that the former behaves as a rod of iron in a magnetic field. Wishing subsequently to design an apparatus to indicate the state of charge on electric cables, to see whether they were safe to handle or not, Dr. Thornton describes his chain of thought in the following terms: "The simple electroscope was first thought of, but rejected as not mechanically strong enough to be placed in a plate-layer's hands. It then occurred that one might use the same thing on a small scale, observing the movement in a kind of linen tester's microscope. On trying this I noticed a small spark pass between two particles of carbon suspended in oil, and the idea at once came, why not look for the sparks? In 10 minutes or so after this the first 'voltascope' was made, and tried on a 2,000-volt circuit, and a little while later on 20,000 volts. The instrument, as it is now made, consists of a glass tube containing oil and short, fine carbon rods. Contact is made across the mains through a heavily-insulated wire, having an ebonite of fiber handle with sharp steel points to get good contact. It works equally well with direct and alternating currents, and in any position. As soon as contact is made the carbon filaments become polarized and form an irregular chain [illegible] small current passes, jumping [illegible] particles, and so forming sparks which are instantly extinguished by the oil, their constant appearance and extinction giving the tube the [illegible]"

All this is exceedingly interesting to all concerned with scientific research, and not least to electrochemists and electrometallurgists. It specially serves to emphasize the [illegible] alert intelligence, broad sympathies, and a high standard of general scientific knowledge to the man who would, in any branch of technology, serve his day and generation both faithfully and usefully.

Mr. Swinburne, in an address to the students of the Manchester Local Section of the Institution of Electrical Engineers, chose as his subject "Efficiencies." The importance of looking at things from a monetary point of view was urged, because "all engineering is a question of money." Money matters on the engineering side are not generally regarded in the manner set out by Mr. Swinburne in the following words: "In making any bargains with Nature we always lose in a sense, and we are so accustomed to it that we take it quite contentedly, and merely try to lose as little as possible in the transaction. As I said before, the relation of what we do not lose to the whole is termed 'efficiency.' In dealing with our fellow men we always hope to make a profit, and we sometimes do. In such a case it might be correct to talk of an efficiency of over 100 per cent, but it would be unusual. It might be still more unusual to say a bankruptcy which resulted in a payment of 6s. 8d. in the pound was a transaction with an efficiency of 33⅓ per cent. But there would be enough justification for such a proceeding to enable me to discuss some aspects, even of electrical commerce, under the head of 'efficiency.'"

Passing to thermo-dynamic questions, Mr. Swinburne was also heterodox. He said: "Heat includes what is called sensible heat, or heat that makes things hot; latent heat, such as the heat that disappears when ice is melted or water vaporized at constant temperature and chemical energy. * * * It is not orthodox to call chemical energy heat; and you will not find any such treatment of the subject in books on thermo-dynamics. At present I believe I am alone in classing chemical energy or defining heat so as to include chemical energy, but the treatment of chemical energy in chemical thermo-dynamics is quite consistent with my definition, so I may use it, provided I give you due warning of any heterodoxy. I think that it is not realized that chemical energy is necessarily low-grade energy, only partially convertible into work. This involves the idea of chemical temperature, which may be important in chemistry in discussing the way any given possible reaction will go; but we are not concerned with that here. It will be said that the coal is cold when it is put into the furnace, as cold as the air in fact; so that if this idea is right none of the chemical energy is available. But I must give some idea of what I mean by chemical temperature. If carbon and oxygen are heated to a high enough temperature you get a state on which, in the least fall of temperature, the carbon and oxygen combine and give out sensible heat of that temperature, under the other circumstances that obtain. The least increase of temperature, however, causes the carbon and oxygen to separate again, absorbing sensible heat. At this temperature, therefore, under the pressure and [illegible] relations of the carbon, oxygen and carbon [illegible] heat and chemical energy are interchangeable. This temperature may be called the chemical temperature of the [illegible] carbon and oxygen."

The bulk of the paper was, however, given to a discussion on municipal trading, but this is too remote and insular a subject for discussion in my letters.

Of direct importance to electrochemists was Prof. Threlfall's

address to the Birmingham Local Section of the Institution of Electrical Engineers, which was entitled "On Some Problems in Electro and Electrothermal Chemistry." Prof. Threlfall's repute as an exponent of general thermo-dynamics is exceedingly high, and his recent Institution paper on the testing of the losses in alternating-current generators by air calorimetry was ingenious and convincing, although too complicated for the commercial testing of a manufacturing firm.

As first in intrinsic importance the atmospheric fixation of nitrogen was discussed. The speaker, after dealing with the commercial prospects (whose foundation he considered to be electrical power at $10 per electrical horsepower-year), proceeded to make certain suggestions, such as whether mercury-vapor air and steam at, say, 160° C., could be made to yield ammonia, if not oxide of nitrogen, a process which would be impracticable if the mercury were not recoverable. For the future, Prof. Threlfall looked forward to a time when the price of fixed nitrogen should have dropped to one-tenth of its present value.

The next subject treated was that of non-conducting carbons, in regard to which a striking parallel was pointed out as existing between aged monoclinic sulphur and graphite crystals of diamond shape, and van't Hoff's pressure-temperature diagram was applied to the manufacture of diamonds. From this it was deduced that considerably higher pressures than had yet been applied would be necessary to produce large diamond crystals. Emphasis was laid upon the interest in the transformations to which boron, silicon and selenium were subject, Prof. Threlfall's view being that "the most interesting phenomena may be expected to occur only after the pressure is high enough to produce a material change of volume in the compressed substance."

The Institution of Mining and Metallurgy.

At the ordinary meeting in November the papers discussed were those crowded out by the length of the virile discussion in October, on Mr. Dietzsch's paper. I am now able to deal with these and the discussion which they occasioned. The first, on "The Occurrence of Gold in Upper Sarawak," by Mr. J. S. Goekie, was mainly a geologist's paper, and save for incursions into the idiosyncrasies of Chinese laborers, the discussion was of the same order. "The Computation of Assay Values" formed the subject of the second paper, by Mr. W. Crossley, in which several methods of computation were discussed and contrasted. The paper applies chiefly to banket reefs of varying width and values. The paper and resultant discussion do not admit of being easily abstracted; they would form good copy for a mining magazine, but are foreign to the scope of my letters, which have to treat generally with the problems of electrochemical and metallurgical processes of treatment. Mr. Crossley also described in the third paper "A Graphic Calculator." The fourth paper was by Mr. E. J. Way, on the "Ore Valuation of a Witwatersrand Mine," and was similar in matter to the second paper.

At the ordinary meeting in December three papers on gold mining were submitted. One, by A. R. Weigall, dealt with "Gold Mining in Japan," the other two, respectively by Mr W. Hamilton and Mr. J. C. Scrutton, dealt with the occurrence and treatment of gold in Sarawak.

The 1906 Meeting of the Iron and Steel Institute.

The Council of the Iron and Steel Institute have arranged that the annual general meeting of the Institute shall be held in London on May 10 and 11, 1906. In place of the usual Autumn meeting a joint meeting with the American Institute of Mining Engineers will be held in London on July 23 to 28. It is intended during the week following to give the American visitors an opportunity of seeing some of the iron-making districts. It is anticipated that the visiting party will include many of the leading ironmasters who entertained the Iron and Steel Institute in America in 1890 and 1904. The Lord Mayor of London has kindly consented to act as chairman of the London reception committee, and to give an evening reception at the Mansion House.

The Faraday Society Meeting.

The December meeting of the Society, held on the 12th of the month, was well attended. Lord Kelvin was unavoidably absent, and we therefore had the pleasure of Mr. Swinburne as chairman, and at the same time as the reader of the first paper ("The Physics of Orè Flotation," by Mr. Swinburne and Dr. Rudorf). Carrying the conscience of the chair with him to the blackboard, Mr. Swinburne ruthlessly closured himself at the end of 20 minutes to make way for the next paper, by Prof. Huntington, on the "Concentration of Metalliferous Sulphides by Flotation."

I understand that the genesis of the two papers,* which were discussed simultaneously, was a battle royal at the Royal Courts of Justice, where Mr. Swinburne and Prof Huntington appeared on opposing sides as expert witnesses. The second round was so interesting that one regrets missing the first.

The discussion was opened by Mr. Bertram Blount, who confessed his ignorance of the subject, despite the fact that he had worked at the problem. He had listened to both papers, and remained entirely unconvinced. He could not subscribe to either hypothesis, and he had not one of his own ready to offer in substitution. As far as he saw, it was necessary to have, firstly, an appropriate gas; secondly, an appropriate solid; and, thirdly, an appropriate temperature of application. What was important was the practical effect that valuable metalliferous particles could be separated from valueless gangue. Experiment was their only guide, and they always had to fall back upon laboratory trials in order to get the conditions under which the valuable and worthless parts could be separated.

Mr. J. G. A. Rhodin, in a fluent and excellent speech, explained that he shared Mr. Blount's difficulties. The question was: how did the gas get there, and how did it get in contact? When an envelope of gas surrounded a solid, the stress was equal all round. When carbon dioxide gases impinged on a solid, a considerable dynamic effect was evidenced. The impact, however, was much greater than could be exercised by carbon dioxide in motion. To account for the dynamic effect they had two theories: one of impact, and a second (and more feasible) one that carbonic acid gas was evolved in close proximity with all the solids in the liquid. The case of calcium carbonate in sulphuric acid was a distinct case of mass action, which Gulberg and Waage's law explained. He believed that in the case of ore flotation part of the solid went into solution as solid. The solubility of calcium carbonate was very low, but the continued evolution of CO_2 would make this a typical case of mass action. Ferric carbonate and manganese carbonate were more soluble, and were simply in a transition stage. Then if gaseous CO_2 was evolved in the solution, the gas would get to the surface, carrying with it the particles of ore and gangue. On these submerged particles occluded gases were actually present, and to get rid of these was well-nigh impossible. He did not believe that Prof. Huntington's attempts to remove these films of occluded gases could possibly have been successful. Finally, he did not understand how a liquid could surround a solid without a layer of air between the liquid and the solid, unless they fell back on the ether as an explanation, which was always their last leg scientifically, when all other explanations failed. It seemed to him that all Messrs. Swinburne and Huntington between them had succeeded in doing was to explain Quincke's formula.

Dr. Steinhart, who followed Mr. Rhodin, began by remarking that there was a third class of flotation, *i. e.*, that of the Zinc Corporation, Ltd., which was the outcome of the first battle of the two theories. Dr Steinhart then asked a general

* A summary of both papers will be found in an article on another page under the title "Ore Dressing by Flotation." Ed.

question concerning the Elmore method of flotation: did not this take place in a vacuum?

Prof. Huntington, intervening, remarked that Elmore used both acid and oil.

Dr. Steinhart's next query was as to whether the original idea had not been that of separating galena from zinc blende, Mr. Swinburne replying in the affirmative, but that it had been found impossible, the process acting as against carrier only.

Mr. F. S. Spiers mentioned the persistence of films of air, and stated that he had experimentally tried the Volta contact effect with plates of aluminium and platinum in the best vacuum attainable. In spite of heating to a certain point, no diminution of contact effect occurred, but above this point the formation of a film of surface oxide on the aluminium reduced the voltage effect to zero. He, therefore, disagreed with Dr. Huntington's claim that the air film had been removed, and stated that his (Mr. Spiers') conviction was that gaseous films were impossible of removal.

Dr. Perkin next deprecated the confusion which seemed prevalent between occlusion and air films, and Mr. Bluman asked why, granted that there was occlusion, the films chose the sulphide and not the gangue.

What was nominally the reply to the discussion, but owing to the interjection of remarks was actually the opening of a further discussion, was then commenced by Prof. Huntington, who remarked that the question of an air film was very different from that of an occluded gas, and that it was doubtful whether one got occluded gas with a sulphide. He could not see what was the necessity for a film. Given a sufficient angle of contact, a sufficient quantity of gas would adhere. He could not accept Mr. Rhodin's views as to the necessity of a film. A member then suggested that the separation of the gangue and sulphide particles was electrical.

Dr. Rudorf opened his part of the reply by asking Prof. Huntington whether if a particle floats or not depends on the contact angle, did not the contact angle depend on the air film? The authors did not lay entire stress on weathering; for instance, molybdenum sulphide was unaffected by weathering but flotation occurred.

After [illegible] Dr. Walker pointed out that the question of the air film was a [illegible] point. Surface tension was affected by adhesion and by temperature. Metallic particles had been observed floating [illegible] Personally, he did not believe in an electrical theory, as the substances which floated best did not conduct. Mr. Swinburne and Prof. Huntington [illegible] of thanks to each other.

Dr. Walker's paper on "The Loss of Heat When Iron" being taken as read, Dr. Lowry, speaking on behalf of Mr. W. R. Bousfield and himself, expressed [illegible] Dr. Walker. So far as he was concerned in his prior remarks on the subject, he had taken figures [illegible] work in 1884. A vote of thanks to Dr. Walker [illegible] Mr. Swinburne, brought an interesting meeting to a close.

Market Quotations During December.

Comparing the first and last days of December, there are very few changes to report in regard to the prices ruling in the chemical trade. Ammonia sulphate is about 5s. per ton dearer. Cyanides are dearer, the price of 8½d. per pound being reported on the 23d. Shellac also has risen to [illegible] per cwt.

Passing to the metal markets, English lead [illegible] £17 [illegible] a rise of £1 11 per ton during the month, and [illegible] the lead derivatives are correspondingly higher. The demand for copper is firm, stores of refined copper scarce, while large deliveries are not anticipated for a little while. Copper reached £79.15 per ton on Dec 30. The rise during the month was £11. Although a further rise is possible, the price of £100 per ton mentioned in some quarters is quite improbable. Straits tin closed at £10[illegible] per ton, after having reached £[illegible] The general advance from £133 on Jan 1 last is really [illegible] Hematite pig iron closed at 71s. 6d., an increase of 1s. 6d. during the month, while Cleveland warrants closed at [illegible] similar advance of 1s. 6d. per ton marking the change of the month.

The general outlook seems to be exceedingly favorable, and Mr. Hadfield's prophecy in May last of a coming trade revival, if not a boom, was well founded.

London, Jan 3, 1906.

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

Electric Furnaces.

Production of Chlorides of Carbon.—F. J. Mechalske, 808,100, Dec. 26, 1905. Application filed June 1, 1905.

Calcium oxide, calcium chloride and carbon (in the form of broken coke) are mixed and heated in an electric furnace to form oxychloride of carbon according to the reaction

$$2CaO + 2CaCl_2 + 10C = 4Ca + 2COCl.$$

The oxychloride of carbon is passed through a mass of boneblack, coke or pumice, which results in its decomposition into carbon tetrachloride and carbon dioxide, according to the equation

$$2COCl = CCl_4 + CO.$$

Both of these products being in gaseous form are separated by passing the same through a reflux condenser, wherein the carbon tetrachloride is converted into liquid form, while the carbon dioxide is withdrawn by a compressor and [illegible] It is stated that sodium or magnesium chloride may be used as starting material instead of calcium chloride.

Metallurgical Furnace.—G. H. Benjamin, 808,187, Dec. 26, 1905. Application filed April 27, 1903.

The furnace contains four essential chambers, the calcining chamber, the reducing chamber, the electric combining chamber, the receiving or oxidizing chambers. These chambers are connected together. The ore is passed by a reciprocating rake through the calcining chamber, and is then subjected in the reducing chamber to the action of [illegible] carbon fumes. In the electric chamber there is a series of electric arcs through which the ore passes, and before and while the ore is subjected to the action of the arcs, finely divided and preheated, other materials, fluxes, carbon, etc., are mixed with the ore. The current transmitted between the electrodes serves not only to form the arcs, thereby creating high-temperature effects [illegible] also to produce electrolytic effects, [illegible] ing apart of the materials which [illegible] form, so to speak, an electrolyte. Such [illegible] termed from ores, carbon, or the [illegible] the electric current [illegible] taking place between the [illegible] and further the [illegible] act mechanically, especially where an alternating [illegible] is used to [illegible] pended [illegible] precipitate [illegible]

Manufacture of Calcium Carbide.—E. F. [illegible] J. G. Marsh, [illegible]

and the high heat conductivity of the latter. On the other hand, graphite has the important advantages of high electric conductivity, uniform composition and ease of machining. The object of the patent is to use graphite rods under such conditions as to not materially increase the cost, while the life is so greatly lengthened that the final cost is reduced to a minimum. The greater cost of the graphite is offset by the reduction in the cross-section and in the length of the electrode rods. The rods may be coated with tar and enclosed in iron

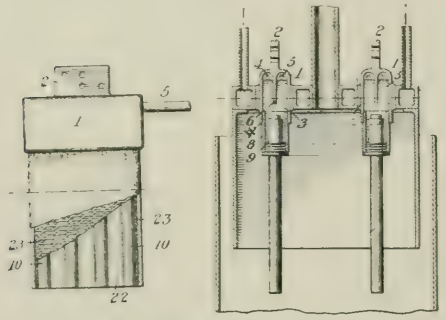


FIG. 1 ELECTRODES FOR CARBIDE FURNACE.

sheaths, so as to exclude air from their surface. They are efficiently strengthened and protected from oxidation by the reinforcing body of cement 22 (Fig. 1), while a support 23 of thin foraminous metal, such as "expanded iron," is wrapped around the rods and plastered over with the cement. The electrode holder consists of a head 1 of cast iron with the terminal 2; within the holder is a water-chamber 3, with supply and discharge pipes 45. A longitudinal depending baffle 6 extends from the top nearly to the bottom of the water-chamber. The upper end of each graphite electrode rod 10 is threaded and screwed into a socket 9, making good electrical contact with it. The water-cooled holders and electrode sockets prevent the metal parts from burning out, and the long socket nipples 8 decrease the stub waste of the electrodes.

Electrolytic Processes.

Treating Molten Matte.—W. E. Koch, 808,849, Jan. 2, 1906. Application filed Feb. 19, 1903.

The object is to treat matte containing a mixture of different metallic sulphides, so as to remove the sulphur and at least partially separate the metals. This is done by electrolysis with simultaneously forcing a gas through the matte. As an example, the treatment of copper matte containing iron is described. The matte is tapped from the smelting furnace into a fore-hearth, where it settles to the bottom and is then tapped into an inclined trough 5. The upper diagram shows a vertical section of trough 5, the lower diagram a horizontal section; 6 and 7 [illegible] made of graphitized carbon and covered with a protecting coating. They are so arranged that the matte flows slowly between them, while water gas is forced through [illegible] into the molten bath. Water [illegible] hydrogen and carbon [illegible]. The hydrogen combines with the sulphur in the matte to form a gas, which passes off, [illegible] to reduce and separate

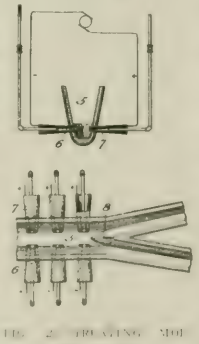


FIG. 2 [illegible] AND [illegible] SECTION.

the iron in the ordinary manner. The electrolytic action of the current on the fused matte causes the iron to be drawn toward one electrode and the copper toward the other, and I utilize this action to separate these two metals. For this purpose I place a splitter 8, of plow shape, at the mouth of the trough, in such a position that the iron will be deflected toward one side and the copper toward the other, thus separating them. The copper will contain the precious metals, if such are present. In practice the iron collects in the metallic state in lumps or masses in the mouth of the trough at one side of the splitter and is drawn out by the operator. The copper drops from the other side and chills in lumps of bullion."

Production of Metallic Calcium.—W. Borchers and L. Stockem, 808,066, Dec. 26, 1905. Application filed Oct. 24, 1902.

Fused calcium chloride is electrolyzed with a comparatively small cathode against a large anode. The cathode is artificially cooled, so as to keep the temperature of the electrolyte in its neighborhood at a moderate red heat, not exceeding the melting point of metallic calcium. Under these conditions calcium is deposited in the shape of a soft sponge upon the cathode. If the sponge is taken out by means of a spoon, etc., and immediately dumped into kerosene oil to prevent oxidation of the spongy calcium by the oxygen or the moisture of the air, a crude metal is obtained containing from 50 to 60 per cent of pure calcium, the rest being adhering calcium salts. If, however, the deposited sponge, before lifting it out of the electrolyzing vessel, is caught by a pair of hot tongs and pressed between the jaws, the larger part of the calcium salt enclosed in the pores of the sponge will squeeze out, and the spongy mass will weld together into a solid plate, which may be taken out of the electrolyte and cooled without serious danger in the open air.

Producing Caustic Alkalis and Chlorine.—H. S. Blackmore, 809,085, Jan. 2, 1906. Application filed July 22, 1903.

H. S. Blackmore, 809,088, Jan. 2, 1906. Application filed May 10, 1905. Patent 809,085 covers the apparatus, patent 809,088 the process. Fig. 3 shows the arrangement for a molten electrolyte. The iron vessel 1 on the right hand of the diagram serves for the electrolysis of fused sodium chloride, a mass of molten lead 23 being the cathode, and the graphite rod 24 the anode. The cell is divided into an anode and cathode compartment. The anode compartment is in the center, and has a cylindrical form, being closed at the bottom by a porous diaphragm (of broken magnetite 22 resting on iron-wire gauze 21). The diaphragm floats upon the molten lead cathode and remains in close contact with it, notwithstanding considerable change in the surface level of the molten lead. Above the diaphragm the cylindrical anode chamber is filled with fused sodium chloride. The chlorine, set free at the anode, is drawn off through pipe 27. The lead-sodium alloy produced at the cathode being lighter than lead, is continuously displaced as formed from the surface of the cathode beneath the diaphragm, and rises to the surface of the cathode around the anode chamber, thus being at a higher level than below the diaphragm.

The sodium of the alloy is continually converted into its hydrate or oxide, the remaining lead being returned by gravity to the bottom of the vessel 1 beneath the diaphragm, to receive further additions of sodium. For this purpose an annular pipe 29 is provided below the surface of the molten lead; it is arranged around and concentric with the anode chamber. This pipe has two rows of downwardly and outwardly opening perforations 30. Connecting with pipe 29 is a pipe 31, having valve 32, which serves for the introduction of molten sodium hydrate, which is injected in a plurality of fine streams into the molten lead-sodium alloy, and is reduced by the sodium to sodium oxide with evolution of hydrogen, passing out through 33. To facilitate the oxidation of the sodium, a layer 60 of loose granular iron, magnetite or ferro-

silicon, is arranged near and beneath the surface of the cathode around the anode chamber. This serves to distribute the streams of sodium hydrate rising from pipe 29, and bring them into thorough and intimate contact with the sodium in the alloy. The oxidizing action is facilitated by the innumerable local couples, due to the presence of the particles of iron, etc.

The sodium oxide thus formed is continually drawn off into a second vessel 35, and is now converted into hydrate. For this purpose a pipe 41 leads upward from beneath the level of the molten oxide in 35, past a steam injector 42, which serves both to introduce the water necessary for hydration and to carry the old material upward to a third vessel 43. The bent upper end of the pipe 41 contains a spring-closed air-inlet valve, to prevent any siphoning of hydrate back to 35. Such amount of sodium hydrate as is required to oxidize the sodium

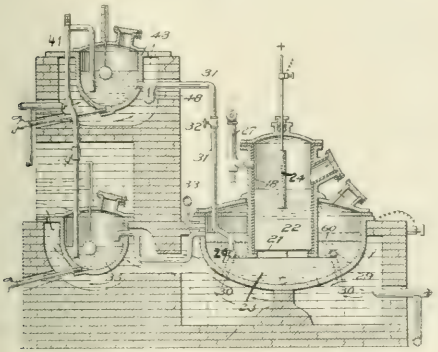


FIG. 3.—ELECTROLYSIS OF FUSED SODIUM CHLORIDE.

from the sodium-lead alloy, continually passes off from vessel 43 through a trapped outlet 48, communicating with pipe 31. The apparatus may be modified so as to be suitable for electrolysis of an aqueous solution of sodium.

The successive steps of the whole scheme of getting chlorine and caustic soda by electrolyzing fused sodium chloride are indicated by the following three equations:

$$mNaCl + nPb = mNa.nPb + mCl$$
$$mNa.nPb + mNaOH + mNa_2O + nPb + mH$$
$$mNa_2O + mH_2O = 2mNaOH$$

Production of Caustic Alkali and Chlorine.—H. S. Blackmore, 809,089, Jan. 2, 1906. Application filed May 10, 1905.

An aqueous solution of sodium chloride is electrolyzed with a mercury cathode, and in a separate vessel sodium hydrate is produced from the sodium amalgam. The chief feature of the process is that the movement of mercury and amalgam through the various vessels is effected solely by the differences of level of the various bodies of mercury and amalgam, and is entirely automatic. The apparatus is shown in vertical and horizontal section in Fig. 4. The vessel 1 contains mercury, and the valve 8 in outlet 2 is controlled by the lever 9 and ball float 4, which serves to maintain a uniform level of mercury in chamber 3. This mercury is made cathode by the connection 13', and communicates through the manifold 10 and several outlet pipes 11, with the mercury 13 at the bottom of the electrolyzing cell, which contains an aqueous solution of sodium chloride 14 above the mercury. Solid sodium chloride in the two receptacles 22, which are through openings 23 in communication with the electrolyzing cell proper, maintains the solution in saturated condition; 19 are the anodes. The chlorine gas, set free at the anodes, is led off through 20 and 21. The sodium amalgam formed at the cathode increases the bulk of the mercury until it rises above the level of the outlet pipe 25, whereupon it overflows and drops through a layer of carbon tetrachloride in vessel 26, and accumulates and passes out into the oxidizing vessel 28 through pipe 27. The carbon tetrachloride [illegible] short circuit between the mercury in the electrolyzing and in the oxidizing vessel 28. The latter is divided into a number of parallel compartments, alternately communicating at op-

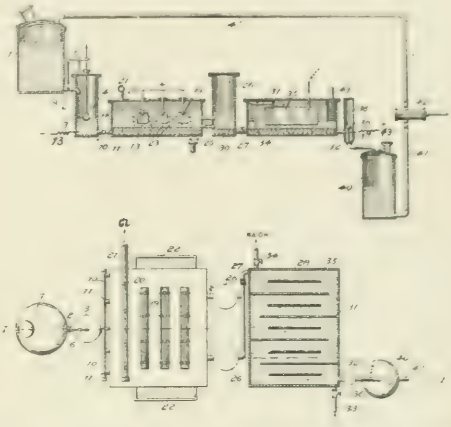


FIG. 4.—ELECTROLYSIS OF AQUEOUS SOLUTION OF SODIUM CHLORIDE.

posite ends by vertical baffle-plates 31. The liquid amalgam is introduced by pipe 27 into one end of the compartments, and flows in zigzag form to the outlet pipe 32. Adjacent to, but above this outlet 32, is an inlet pipe 33 for water, which flows over the amalgam; the resulting sodium hydrate escaping through outlet pipe 34. The rate of oxidation is increased by placing carbon electrodes 35 in the water and connecting them electrically with the amalgam below, either directly or by connecting the amalgam with the positive lead 43 of the electrolyzing current and the carbon electrodes with the negative lead. Pipe 32 carries the pure mercury from the oxidizing cell into the cylinder 36, in the bottom of which is a valve-outlet 37, controlled by a float 38. The mercury then collects in vessel 40, and is returned by means of pump 42 through pipe 41 to reservoir 1. After the electrolyzing current is turned on the whole procedure is automatic, due to the differences of level in the various bodies of mercury and amalgam, and these differences are maintained by suitable predetermined adjustment of float-feed 8 and float-controlled outlet 37, and of the height of the connecting pipes.

Diaphragm Cell.—H. Koller and P. Askenasy, 809,116, Jan. 2, 1906. Application filed Oct. 29, 1903.

If the anodic product is a gas and insoluble in the electrolyte, detrimental secondary reactions, due to the cathodic passing into the anode compartment, may be avoided according to the inventors, by systematically [illegible] the diaphragm, or using a diaphragm of such a construction that the electrolyte which is introduced into the anode compartment is forced to flow over to the cathode compartment [illegible] direction substantially at right angles to the line of diffusion. If the diaphragm is constructed, for instance, of wedge-shaped parts, superposed as shown in Fig. 5 (the upper figure being a cross-section through line 33 of the lower figure), the electrolyte is caused to flow in zigzag form [illegible] to the cathode. On its way it meets the [illegible] diaphragm which are [illegible] with [illegible]. The fresh electrolyte absorbs the [illegible] and carries them along back to the cathode. This results in [illegible]

tematic lixiviation of the diaphragm; *f* is the anode, *n* the cathode. The electrolyte is introduced through pipe *c*, which is provided on its sides with openings *d*. It flows freely through the interstices *b*. The channels *g* facilitate the flow of the electrolyte. In an ordinary diaphragm cell, at a temperature of 60° C. and at 4.5 volts, the inventors obtained a cathode lye of 90 grams NaOH per liter, with a current efficiency of 92 per cent. By substituting a diaphragm of their own concentration they obtained, with the same current and voltage, a concentration of 189 grams NaOH per liter, with a current efficiency of 87 per cent; it was possible to increase the concentration to 120 grams NaOH per liter, but the current efficiency decreased simultaneously to 65 per cent.

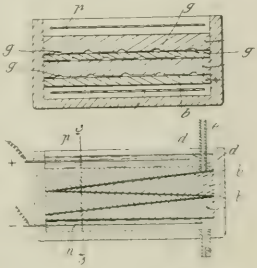


FIG. 5.—DIAPHRAGM CELL.

Production of Organic Compounds by Oxidation.—W. Lang, 808,095, Dec. 26, 1905. Application filed Aug. 28, 1902.

The inventor oxidizes organic compounds by means of manganic-oxide salts (generic formula Mn_2R_6), which are thereby reduced to manganous-oxide salts. The latter are then reoxidized by electrolysis; since manganic-oxide salts "are not subject to reduction at the cathode, a simple cell without a diaphragm may be used. The process is intended chiefly for the production of aldehydes, ketores and quinores. The following example is given: As starting point mangano-ammonium sulphate is used, which may be obtained in the shape of yellow anhydrous crystals from a hot acid solution with an excess of ammonium sulphate. Its formula is $MnSO_4 + (NH_4)_2SO_4$, and on the addition of cold water it splits up into $MnSO_4(NH_4)_2SO_4$ - aq and $MnSO_4$ + aq. The inventor mixes 47.5 kg. of mangano-ammonium sulphate together with 45 kg. of water and 75 kg. of 98 per cent sulphuric acid in a lead-coated receptacle provided with a stirrer, and subjects the mixture to electrolysis with an anodic current density of 3.5 amps. per square decimeter. The lead coating of the vessel serves as anode; the cathode is also made of lead. After 6,000 ampere-hours the manganese is present as a manganic-oxide salt, *i. e.*, as ammonium-manganese alum, the greatest part of which is separated out in a state of very fine division. This dark red salt undergoes decomposition when brought together with water, thereby forming manganic oxide in the shape of brown flakes. After the water evaporated during the electrolysis has been replaced, the entire electrolyzing mixture is stirred together with 4 kg. of toluene in a closed lead-coated receptacle, provided with a stirrer, the temperature being maintained at about 50° C. During the reaction about 8 kg. of water are gradually added. After 2 or 3 hours the reaction is complete, and the solution and the undissolved salt have become perfectly clear. The oil mixture is driven out by passing steam through the vessel, and is worked up in the usual manner. The manganese mixture goes back for repeated electrolysis. From the oil mixture about 0.6 kg. of toluene are recovered, and about 4.7 kg. of aldehyde are obtained, "which is over 80 per cent of the theoretical quantity in relation to the toluene consumed. With lower chain acid, benzoic acid would be formed besides the aldehyde; while a stronger acid would lead to an increased quantity of condensation products, but would accelerate the reaction. If the sulphuric acid is considerably stronger than 70 per cent, the electrolytic oxidation does not take place. If it is too dilute, peroxides and permanganic acid are produced at the anode. Similar effects are produced if instead of toluene, the higher homologues are employed, and many substituted toluenes behave in the same manner. Isvengenol is oxidized to vanillin. In case the products of oxidation cannot be driven out with superheated steam they may be shaken out with a solvent which is insoluble in water. The process is stated to be of extensive application in the aliphatic series. For instance, methyl alcohol is oxidized either to formic aldeheyde or to formic acid.

Production of Hydrochloric Acid.—I. L. Roberts, 807,640, Dec. 19. Application filed June 25, 1895. Assigned to Roberts Chemical Co.

Electrolysis of sodium chloride yields (besides caustic soda) chlorine and hydrogen gases in equivalent proportions. The gases are conducted in separate streams to a common point, where they impinge directly upon each other and are burned, one supporting the combustion of the other without the presence of any other gas. The resulting hydrochloride acid gas is conducted off to receivers containing water, by which the gas is absorbed. (The Roberts Chemical Co. are using this process at Niagara Falls for the production of caustic potash and hydrochloric acid from electrolysis of potassium chloride.)

Zinc Plating.—G. L. Meaker, 808,103, Dec. 26, 1905. Application filed June 18, 1902. Assigned to American Steel & Wire Co., of New Jersey.

An adhering and coherent coating of zinc with the desired metallic lustre is obtained by electroplating with the following electrolyte: 450 gallons of water are mixed with 500 pounds of commercial muriatic acid of about 18° Baume, and then 200 pounds of commercial sulphuric acid of about 66° Baume are added, in which is suspended metallic zinc until a saturated solution is formed. By adding sufficient water, the density is reduced to about 0° Baume, whereupon 1 pint of a saturated solution of a vegetable acid—such as oxalic acid, citric acid and tartaric acid and the like—is added to each gallon of the zinc solution.

Separating Galvanoplastic Deposits from Metal Matrices.—E. Albert, 808,331, Dec. 26, 1905. Application filed June 21, 1905.

The process is specially intended for electrotypers, and the object is to provide such an easy separation that both the electro and the matrix are uninjured. For this purpose sudden heating in or floating upon a metallic bath is employed, which causes the electro to spring off from the matrix. The inventor uses a metallic bath with a considerably lower fusing point than that of the metal matrix, and heats it almost up to the fusion point of the metal matrix. In electrotyping, with copper deposits and a lead matrix, the bath may comprise the ordinary backing metal (fusion point 220° C.), or an alloy of 57 parts of lead and 33 of tin (fusion point 150° C.). "In many cases the fusing point of the backing metal will be sufficient to effect the separation, and, accordingly, the separation may be effected simultaneously with the casting of the backing metal upon the deposit. The tinning of the deposit, which is done in order to secure the backing to the matrix, may be effected by placing the leaflets or foils of tin upon the deposit after its deoxidation, and thus the deposit may be tinned over simultaneously with the detaching of the same from the metal matrix.

Electrolytic Production of Lustrous Metallic Coatings on Metals.—A. Classen, 809,092, Jan. 9. Application filed Sept. 28, 1905.

The object of the inventor (who is best known through his work in electroanalysis) is to produce directly by electrolysis a firmly adhering deposit with a bright metallic lustre, without heating or polishing. It is said that iron plates coated with zinc by his method can be bent to any extent without the zinc splitting off. The object is obtained by adding to the electrolyte "a certain quantity of glucoside, or a substance classed among the glucoside, or an extract of plants, roots, barks, seeds, such as Althæapanama or licorice extract, which con-

tain the said substances or their derivatives, such substances being used either singly or in combination." For zinc coatings the following recipe is given: To the electrolyte, consisting of 20 kilograms of crystallized zinc sulphate, 4 kilograms crystallized sodium sulphate, 1 kilogram zinc chloride, and 0.5 kilogram boric acid in 100 liters water, is to be added the extract of 5 kilograms licorice root

Electroplating Apparatus.—A. W. L'Hommedieu, 809,309, Jan. 9, 1906. Application filed April 22, 1905.

A barrel for electroplating small articles in quantities. It is revolved upon an axis laterally inclined from a vertical plane The anodes are secured near the inner surface of the barrel, while one or more resident cathodes are provided near the axle. The articles to be plated are tumbling around, and are held in contact with the resident cathode, and are prevented from making contact with the anode by means of a perforated receptacle of non-conducting material.

Agitation of Electrolyte.—W. C. Wood and B. Oaksford, 808,789, Jan. 2, 1906. Application filed Aug. 31, 1905.

To agitate the solution used in the electrodeposition of metals, the inventors use the arrangement shown in Fig. 6. At the bottom of the tank *a* are one or several troughs *b*. The perforated plungers *c* fit into and slide within the troughs *b*. By reciprocating the plungers the electrolyte is pressed through the holes in the same and thus agitated.

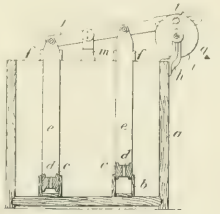


FIG. 6.—AGITATION OF ELECTROLYTE

Electrolytic Deposition.—P. Steenlet, 807,973, Dec. 19, 1905 Application filed Nov. 7, 1903.

The inventor claims that in electroplating a very material improvement ("an increased and regular deposit") results from adding to the electrolyte small quantities of certain organic materials, as albuminoids, but that when these added materials get occluded in the deposit, the latter is rendered brittle and irregular. He claims he can get the good effect without the bad one, "by employing a diaphragm impregnated with or otherwise made to carry in suitable disposition and quantity the organic substance, which latter is rendered in soluble."

Treating Hides.—F. B. Hinkson, 807,930, Dec. 19, 1905. Application filed Dec. 27, 1904.

Details of construction of an electrolytic tank with a series of bars supporting the hides, and a series of electrodes in parallel with them. The direction of the electric current through the tanning solution is automatically reversed in certain intervals.

Asymmetric Cell; Aluminium Electrolytic Valve.—M. Buettner, 809,770, Jan. 9, 1906. Application filed Aug. 4, 1904.

For electrolytic rectifiers and condensers with aluminium electrodes an electrolyte consisting of a solution of caustic ammonia and boric acid is used. "It is not necessary that the solution contain the substances in the proportion of their chemical equivalents, as more or less of either substance can be used."

Treating Hides. F. B. Hinkson, 810,144, Jan. 16, 1906 Application filed Dec. 27, 1904

The claim reads: "A process of treating hides which consists in suspending a plurality of hides between a plurality of electrodes in such manner that each hide is disposed between one pair of electrodes, passing an electric current transversely through each hide, and reversing the direction of the current at suitable intervals."

Purifying and Filtering Water.—L. [illegible] 1905 Application filed Dec. [illegible] 1904

Details of construction of an apparatus in which the water is first subjected to electrolysis using [illegible] vertical electrodes and then passed through [illegible]

BATTERIES

Dry Cell. C. Jaeger, 808,755, Jan. 2, 1906. Application filed Feb. 27, 1904

A battery cell having a zinc cup, a carbon pencil within the cup. A compound of [illegible] powdered graphite, containing a [illegible] dium," ferrocyanide of potash, [illegible] surrounds the carbon pencil. The remaining [illegible] with flour and plaster of Paris.

Storage Battery Plate.—D. P. Perry, 809,742, Jan. 9, 1906. Application filed June 8, 1903.

The grid comprises three flat superimposed lead plates. The two outer plates are provided with a plurality of vertical columns of obliquely-arranged outwardly-tapering apertures, the obliquely arranged aperture of one plate being arranged at an angle to those of the other plate. The intermediate plate is provided with evenly distributed small openings.

Electric Cell.—K. Tsukamoto, 809,647, Jan. 9, 1906. Application filed Jan. 14, 1904

The cell has an outer zinc cylinder, and a [illegible] porous carbon tube, filled with the following depolarizing mixture, 240 grams graphite, 100 manganese dioxide, 8 [illegible] chloride, 20 potassium permanganite, 50 ammonium chloride are boiled with 500 grams of a saturated solution of [illegible] permanganate and pressed to a definite form. A fibrous substance, or blotting paper, is bound around the carbon tube, and the space between the inner surface of the zinc shell and the blotting paper is filled with a mixture, consisting of 100 grams ammonium chloride, 8 potassium chloride, 400 gypsum, mercuric sulphate, and 500 dextria, well mixed and dried. When the cell is to be used water is added

DISCHARGES THROUGH GASES

Manufacture of Nitric Acid.—H. Pauling, 807,491, Dec. [illegible] 1905 Application filed April 2, 1902

Atmospheric air is subjected to spark discharges, whereby the nitrogen of the air is completely converted into NO. The mixture of gases thus obtained is mixed with fresh [illegible]

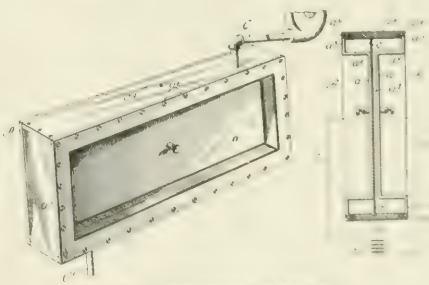


FIG. 7.—[illegible] FOR [illegible]

pheric air, and the whole is exposed in a [illegible] the influence of dark electric discharges. The oxygen of the air is thereby converted into ozone, which [illegible] bined with the nitrogen treated in the [illegible] and thereby forms anhydrous nitric acid

$$NO + 2O = NO_3 + O$$

Upon adding water, nitric acid is obtained: $2NO_3 + H_2O = 2HNO_3 + O$. In order to form nitric acid [illegible]

$$N_2O_5 + H_2O + 2O = 2HNO_3 + 3O$$

the air may be charged with moisture before undergoing the electrical treatment. To this end the air is conducted through a vessel filled with water, where it is charged with moisture, and then into the electrical apparatus.

Production of Ozone.—William P. Rice, 807,964, Dec. 19, 1905. Application filed May 9, 1904. Assigned to National Ozone Co.

The apparatus is a closed box of the construction shown in Fig. 7; a a_2 a_3 are thin metal plates, a_4 and a_5 insulating strips. The silent discharge takes place between the flat parallel plates, a through the air gap a^1. A glass plate c is placed against one of the electrodes. The air is introduced through C, and the ozonized air is carried off through C^1. A modification is also described which permits treatment of the air under pressure. Since both electrodes a a are exposed to the direct cooling action of the atmosphere an undue rise of the temperature is prevented.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

Industrial Electrochemistry.

Tin Refining.—The conclusion of the articles by Dr. H. Mennicke has appeared in the *Elektrochemische Zeitschrift* of December. The author used alloys of tin, 60 per cent, and lead, 40 per cent, as anode in tin fluosilicate solution, leave a residual alloy of tin 30 per cent and lead 70 per cent, with a 45 per cent current yield of tin. Lead which went into solution deposited on the cathodes, but it was noticed that up to a certain concentration lead did not deposit out on the cathode. The question then arose whether lead-tin alloys could be refined, if the solution was always kept below a certain percentage of lead by some other process, which might be complete electrodeposition of the lead and tin, or by precipitation with hydrofluoric acid, or sodium chloride, sodium sulphate, etc. The author's work shows that the electrolytic refining of "work-tin" with a tin fluosilicate solution would be practicable and profitable. In comparison with electrolytic refining of lead, tin is under disadvantage of a smaller market, and has not the quantities of precious metals lead is apt to have, while the advantages are the preparation of chemically pure tin, which commands a higher price and is especially suited for making tin salts, tin oxide, etc. Comparative experiments showed that tin fluoride-hydrofluoric acid and tin fluoride-fluosilicic acid solutions give solid deposits, with a higher current yield than tin fluosilicate-fluosilicic acid solutions. As pointed out, these former solutions may not be used for alloys, on account of the character of the anode residue, but are suitable for work tin. The author tried steel cathodes, and found they were somewhat attacked, and that the resulting solutions gave spongy tin. He also noted that the presence of free acid tends to make the tin deposit more spongy, and also when the acid went below 1 per cent. From 2 per cent to 12 per cent the deposits are solid. When refining anodes of solder the antimony and copper contained do not dissolve, only the lead and tin, and as soon as lead begins to separate with the tin (after reaching a certain concentration) the deposits get spongy. With more lead in the anodes, as the solution approaches in composition the Betts lead electrolyte, the deposit becomes stronger again. The author believes a seperation of lead and tin with this solution would be practicable only if the lead is repeatedly removed from the solution by precipitation, and then the lead would be won only as a lead salt. Experiments along this line, especially with sulphuric acid as precipitant for lead, gave negative results. The author found that compressed tin-sponge behaved well as anode. The conclusions of the author are as follows:

1. The preparation of the tin electrolyte is much more difficult than that of the corresponding lead solution.
2. The removal of hydrofluoric acid from the solution is not necessary.
3. The regeneration and purification of electrolyte is more difficult than in the Betts lead process.
4. The current yield is lower than in Betts process.
5. Solid refined tin can be obtained under certain conditions.
6. The electrolytic refining of tin is economical in the absence of much lead. The removal of copper, bismuth and antimony is easy.
7. The extraction of tin from lead-tin alloys is possible, but not economical.
8. A combination of electrochemical and chemical processes for refined lead alloys is not economical.

Conclusions 1 and 3 would have to be modified if the author used for the preparation of solutions a diaphragm cell with tin as anode in a solution of fluosilicic acid, while hydrogen is liberated at the cathode.

Welding by Acetylene.—In the *Bulletin* of the Italian Engineers and Architects, Mr. Memmo has recently discussed the welding of metals with acetylene gas. According to an abstract in the London *Electrical Review*, Dec. 29, the author found the following arrangement for getting a mixture of the acetylene and oxygen gases for the production of the oxyacetylene flame very satisfactorily: He passes the two gases through several extremely fine tubes, contained in one large tube, beyond the *T* piece connected to the separate acetylene and oxygen holders, and this has afforded excellent results, when working with an initial pressure equivalent to 1 or 1½ meters of water. When intending to use this apparatus, rubber tubes are connected between the receptacles containing the oxygen and the acetylene, both of which must be provided with pressure regulators, and on the acetylene tube there should also be provided a non-return valve, so that under no circumstances can the greater pressure of the oxygen gas cause it to invade and become mixed with the acetylene in the generator or holder of the latter; such a mixture might be extremely dangerous. The spear-like flame should have a diameter of 3 to 4 millimeters, and a length of 10 to 15 millimeters. If there is too much acetylene, there will be a luminous white sheath due to the free carbon. With an excess of oxygen, the flame becomes too clear, and there is the probability of frequent extinction of the flame. With care in the quantitative adjustment of the gases, an extremely neat weld can be effected in about one-fourth of the usual time. Experiments made on welded steel show that the results are as strong as those obtained by electrical welding. Examples are given of practical work done in repairing boiler plates by welding in new portions, and in repairing a vulcanising machine worked at a pressure of 5 atmospheres. The high temperature effects the fusion of platinum and all the other more refractory metals. Whilst approaching the high temperature of the electric arc, acetylene admits simple means for the regulation of the working temperature.

Electric Induction Furnace.—In the "Jour. Soc'y Chem. Ind." of Oct. 31, a French patent (353,391, April 15, 1905)

granted to Soc. d' Exploitation des brevets Dolter is mentioned. There is a single single-phase transformer. Around each of its two vertical legs the primary coils are wound. Around each of these primary coils, but, of course, properly insulated from them, a closed secondary is provided but the central legs of these two secondaries (inside the transformer core) are made into one. The cross-section of the secondary is, therefore, ∞ — shaped. The transformer is made in two separate movable pieces; one portion consists of the upper horizontal part of the core, together with the two vertical legs with the primaries; they fit into appropriate cavities in the stonework of the furnace. The other portion is the lower horizontal part of the transformer core, which completes the magnetic circuit; it is carried on a wagon. A single transformer may thus be used to serve a series of such furnaces.

Electric Glass Furnace.—In the "Journal de l' Electrolyse" of Sept. 15, an electric furnace of Sauvageon for glass manufacture is described and illustrated. It is a continuous resistance furnace. The cross-section of the charge varies according to the heat to be produced in different compartments of the furnace. The author gives some theoretical calculations, to show that the use of an electric furnace is profitable in this case. Another description of the same furnace may be found in the September issue of "L' Industrie Electro-Chimique."

Electrolysis of Water.—A summary of the phenomena of the electrolysis of water with a discussion of the theoretical problems involved and a description of some commercial types of electrolyzers for the production of hydrogen and oxygen is given by J. W. Richards in the November issue of the "Journal of the Franklin Institute."

Castner Mercury Cell.—M. Le Blanc and C. Cantoni have studied the Castner mercury cell for electrolysis of alkali chlorides, and describe in "Zeit. f. Elektrochemie," Sept. 8, a construction suitable for laboratory experiments. At a current density of 0.1 amp. per square centimeter, with slow movement of the mercury and with high concentration of KCl, they obtained an ampere-hour efficiency above 90 per cent, the temperature being up to 40° C.

Electrolytic Meters.—Two new electrolytic meters were described by S. H. Holden before the Birmingham Section of the British Institute Electrical Engineers. The paper is printed in full in the London *Electrician* of Dec. 22. The first meter is of the simple type of gas voltameter, the whole current passing through the meter. A caustic soda solution is used with nickel electrodes, and an old suggestion of Bunsen is made use of to weigh the electrolyte and use its loss of weight as a measure of the coulombs instead of the volume of evolved gases. In the second meter only a portion of the total current is sent through the meter. For a meter to be shunted, the first condition is that the current passing through it is exactly proportional to the difference of potential between the terminal electrodes. In other words, non-polarizable electrodes must be used.

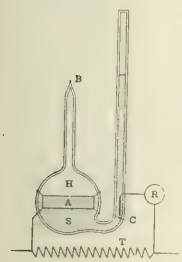


FIG. 1.—ELECTROLYTIC METER.

The author uses hydrogen electrodes, that is, platinized platinum electrodes, saturated with hydrogen, which are well known from scientific research, but have not been used before for practical purposes. The current is in this case proportional to the potential difference, provided that oxygen is not produced faster at the anode than the anode can absorb hydrogen from its hydrogen atmosphere to form water. This condition is easily realized in practice. Fig. 1 shows the essential parts of this meter. It consists of a vertical tube with bulb attached. C is the cathode, a platinum wire covered with platinum black. A is the anode of sheet platinum, also covered with platinum black. S is the electrolyte, consisting of [illegible] of sulphuric acid, and H is an atmosphere of hydrogen. R is a resistance coil equal to about [illegible] which in a 5 amp. meter has a resistance of [illegible]. To use the meter it is held with the tube nearly horizontal and [illegible] the bulb. All the gas then collects in the bulb, while the tube fills with electrolyte, which remains in it, after it is restored to the vertical position for use. When a current is passed through the shunt a steady, fine stream of gas bubbles rises from the cathode gradually, and replaces the liquid in the tube. This gas is measured.

Ozone. A paper by E. Warburg, abstracted in *Physikalische Zeit.*, Jan. 1, discusses the different factors which may increase the efficiency of the production of ozone from oxygen by silent discharges, under the supposition that the zone of the formation of ozone is limited to the luminescent portion of the gas. With a given amount of electricity, the endeavor should be to bring as large a part of the gas to luminescence. Further the increase of temperature, and, therefore, the current density in the zone of formation of ozone should be as small as possible, since the expansion of the gas, due to the increase of the temperature, is unfavorable. Finally, the concentration of ozone in the zone of formation of ozone should be as small as possible.

THEORETICAL AND EXPERIMENTAL

Electrolytic Estimation of Cadmium.—In the December issue of the "American Journal of Science," [illegible] gives some additional notes on the estimation of cadmium with the aid of a rotating cathode, and then summarizes the results of his work on this subject as follows. Cadmium taken in the form of the sulphate may be very accurately and satisfactorily estimated by deposition from solutions containing sulphuric acid, sodium acetate and acetic acid, or potassium cyanide; but little less satisfactorily from solutions containing urea, formaldehyde or acetaldehyde; and also, with proper precautions, from solutions containing pyrophosphates, phosphates, tartaric acid or formic acid. From solutions containing oxalates or oxalic acid, ammonium tartrate, or potassium formate, however, the author was unable to obtain satisfactory deposits. When taken as the chloride, cadmium does not permit such a wide range of conditions. Nevertheless, from solutions of the chloride containing sulphuric acid or potassium cyanide, or the pyrophosphates, the metal is deposited in a form comparable with that [illegible] when cadmium sulphate is taken. Solutions of the chloride of cadmium, to which is added hydrogen disodic phosphate, gave less desirable results, while solutions containing urea, formaldehyde or acetaldehyde, gave deposits free from sponginess only after careful regulation of the conditions. In addition to the solutions containing the oxalates, [illegible] acid, the formates and the tartrates, negative results were obtained in the case of the chloride by solutions containing the acetates, formic acid and tartaric acid. The nitrate of cadmium is ill fitted for electrolytic estimation, the cyanide solution being the only one from which satisfactory results were obtained. From solutions containing 1 per cent or more of free nitric acid the cadmium is not deposited by the current. The same issue contains a note by the same author on the estimation of cadmium as the oxide.

Solution of Platinum in Sulphuric Acid by Alternating Currents.—This problem has been studied very carefully by R. Ruer, who gives his results in "Zeit. f. Elektrochemie," Oct. 13. In the electrolyte [illegible] in sulphuric acid the effect of the cathodic half-wave of the alternating current may also be obtained by means of a reducing agent. In an analogous manner the effect of the

anodic half-wave may be obtained by means of an oxidizing agent. This proves that the solution of platinum in sulphuric acid by means of alternating current is equivalent to alternating oxidation and reduction. When platinum is made alternately cathode and anode in a sulphuric acid solution of 66 to 50 per cent by weight, it goes into solution only if an anodic potential above — 1.20 volt (against a hydrogen electrode in the same electrolyte) is impressed upon it. The higher the potential above this value the stronger the solvent action. With respect to the cathodic potential, the solvent action becomes strongest when the cathodic potential impressed on the platinum is — 0.7 volt. The behavior of platinum with alternating current shows a complete analogy with the phenomena of the passive state. All observations made by the author may be easily explained by the assumption of a peroxide film.

Electrolysis of Fused Nitrates.—The electrolysis of nitrates of potassium, sodium and lithium with different electrodes was studied by A. Bogorodski, who gives an account in "Jour. Russ. Phys.-Chem. Ges." Vol. 37, p. 703, an abstract being given in the issue of Oct. 31 of "Jour. Soc'y Chem. Ind." The author studied the various products of electrolysis under varying conditions, and especially the formation of peroxides of the metals.

Electrolytic Conduction and Specific Inductive Capacity.—The November issue of the "Jour. of Phys. Chem," contains a paper by J. H. Mathews on the relation between electrolytic conduction, specific inductive capacity and chemical activity of a large number of solutions. In many cases, at least, chemical action was found to be wholly independent of any electrochemical phenomena. The Nernst-Thomson rule has no foundation in fact and the exceptions are as numerous as the confirmations. A long and very useful bibliography on dielectric constants is added.

Glass.—In "Zeit. f. Elektrochemie," Sept. 22, E. Zschimmer discusses the physical properties of glass as function of its chemical composition. He shows that this is an extremely difficult problem and that it would generally lead to very wrong results to assume additive properties.

Dilute Amalgams.—A preliminary account of an experimental investigation by J. F. Spencer of the electromotive activity of dilute amalgams is given in "Zeit. f. Elektrochemie," Oct. 13.

Metallurgical Furnaces.

Losses of Heat by Radiation.—J. Bied relates in the *Revue de Metallurgie* for September, the results of experiments made to determine the losses of heat from the sides of a flue or furnace. He had boxes made of galvanized iron with a surface of 0.25 square meter, pressed closely against the brickwork, and water circulated through them. While the conditions do not represent exactly those of free radiation to the air, but rather of conduction to iron cooled by water, yet they are worth recording. The losses through refractory materials of various kinds and thicknesses, for varying differences of temperature, are given in kilogram calories lost per hour per square meter of radiating surface:

Nature of the Wall. Thickness in Centimeters.	Difference of Temperature.	Losses per Hour per m² in Calories.
14 c.m. alumina brick	500° C.	1,600 Cal.
	700	2,208 "
5 c.m. magnesium carbonate	700	2,506 "
Ditto—with 20 c.m. small cinders and 3 c.m. of cement	700	526 "
5 c.m. of lightly compressed infusorial earth	650	1,800 "
20 c.m. alumina brick	700	2,570 "
Ditto—with 20 c.m. small cinders	700	900 "
45 c.m. magnesia brick	1,300	1,300 "
6 c.m. magnesium carbonate		
80 c.m. ordinary fire brick	1,660	1,000 "

Electric Power from Coke-Oven Gases.—The London *Electrical Review* of Jan. 5 contains an abstract of a paper by G. J. Hooghwinkel, read before the Manchester Geological and Mining Society. The author first points out that the generation of electric power from the waste gases of blast furnaces has already proven a commercial success. He refers, for instance, to the gas plant at the Ilseder Iron Works, supplying also the necessary power to the Peine rolling mills near Hanover. The entire steel plant and rolling mills at Peine are driven by electric power generated from waste blast-furnace gases. He then passes over to a discussion of the use of the surplus gases from coke ovens direct in gas engines. He has found in many cases that the cost per unit kw-hour in coke-gas power stations of medium size was about 0.2 to 0.4 cent. The production of cheap power at collieries is a question gaining in importance every day proportionately to the drop in the price of the saleable article and the increased depth of the pits. The firing of steam boilers by means of small coal at the pit is an expensive and, if no rubbish and refuse from the washeries are available, a very wasteful process, having regard to the low efficiency of the boilers. Even taking the value of this small coal at $1.00 per ton at the pit's mouth, the saving of coal by using gas engines is a substantial item. Every modern colliery should, therefore, either burn its rubbish and waste in gas producers, or else compress and coke it in by-product ovens, using the gases obtained in both processes for generating electrical power by means of gas engines. The amount of power to be derived from a battery of 80 ovens of modern type, coking a charge of 7 tons in 32 hours, and using coke of a medium percentage of volatile matter, giving off about 7,000 cubic feet of gas per ton per hour, or in all 122,500 cubic feet per hour would be as follows: If the ovens used 70 per cent of the gases for heating the flues, there should remain about 36,000 cubic feet for power, and taking a consumption of 75 cubic feet per hp-hour, about 1,500 horse-power should be available, or nearly 20 horse-power per oven. This amount varies, of course, with the coal and the type of oven; and the author states he knows of cases where the power is nearly 30 indicated horse-power per oven. The paper is reprinted in the *Mining World* of Jan. 20.

Slags.

Heat of Formation of Silicates.—A very important piece of experimental laboratory work has been done by D. Tschernobaeff, at the suggestion of Prof. Le Chatelier, and reported in *Revue de Metallurgie* for October. In the oxygen bomb of a calorimeter were placed a mixture of charcoal, silica and the salt of the base to be studied. After combustion in the compressed oxygen there remained the silicate of the base, melted by the heat generated.

Barium Silicate.—The heat of the following reaction was measured:

$$BaCO^3 + SiO^2 = BaSiO^2 + CO^2$$
$$= -41{,}000 \text{ Cal.}$$

from which we have

$$(BaO, SiO^2) = BaO, CO^2) - 41{,}000$$
$$= 55{,}700 - 41{,}000$$
$$= 14{,}700 \text{ Cal.}$$

Calcium Silicates:

$$CaCO^3 + SiO^2 = CaSiO^3 + CO^2$$
$$= -27{,}300 \text{ Cal.}$$

Whence

$$(CaO, SiO^2) = CaO, CO^2 - 27{,}300$$
$$= 45{,}150 - 27{,}300$$
$$= 17{,}850 \text{ Cal.}$$

Again:—

$$2CaCO^3 + SiO^2 = Ca^2SiO^4 + 2CO^2$$
$$= -62{,}000 \text{ Cal.}$$

Whence

$$(2CaO, SiO^2) = 2(CaO, CO^2) - 62{,}000$$
$$= 90{,}300 - 62{,}000$$
$$= 28{,}300 \text{ Cal.}$$

Again:—

$3CaCO^3 + SiO^2 = Ca^3SiO^5 + 3CO^2$
$= -107{,}900$ Cal.

Whence

$(3CaO, SiO^2) = 3(CaO, CO^2) - 107{,}900$
$= 135{,}450 - 107{,}900$
$= \mathbf{28{,}550}$ Cal.

Strontium Silicate:

$SrCO^3 + SiO^2 = SrSiO^3 + CO^2$
$= -35{,}100$ Cal.

Whence

$(SrO, SiO^2) = (SrO, CO^2) - 35{,}100$
$= 53{,}000 - 35{,}100$
$= \mathbf{17{,}900}$ Cal.

Magnesium Silicate:

$(MgO, SiO^2) = \begin{cases} +329 \text{ Cal.} \\ +450 \text{ "} \end{cases}$

were two results obtained by directly burning a mixture of MgO, SiO^2 and carbon. The experimenter considers these results unreliable, and considers the heat of the formation of this silicate as not yet known.

Lithium Silicate:

$Li^2CO^3 + SiO^2 = Li^2SiO^3 + CO^2$
$= -24{,}900$ Cal.

Whence

$(Li^2O, SiO^2) = (Li^2O, CO^2) - 24{,}900$
$= 90{,}000\ (?) - 24{,}900$
$= \mathbf{65{,}100}\ (?)$ Cal.

Sodium Silicate:

$Na^2CO^3 + SiO^2 = Na^2SiO^3 + CO^2$
$= -30{,}400$ Cal.

Whence

$(Na^2O, SiO^2) = (Na^2O, CO^2) - 30{,}400$
$= 75{,}600 - 30{,}400$
$= \mathbf{45{,}200}$ Cal.

Aluminates of Calcium.—Seven experiments made with mixtures of Al^2O^3, $CaCO^3$ and carbon gave very small heats of combination, almost within the errors of experiment. They were

$(Al^2O^3, CaO) = 450$ Cal.
$(Al^2O^3, 2CaO) = 3{,}300$ "
$(Al^2O^3, 3CaO) = 2{,}950$ "

Aluminium Silicate.—This was determined very neatly. The heat of the two following reactions was determined, using calcined kaolin for the $Al^2Si^2O^7$:

$2SiO^2 + Al^2O^3 + 3CaCO^3 = Ca^3Al^2Si^2O^{10} + 3CO^2$
$= -101{,}900$ Cal.
$Al^2Si^2O^7 + 3CaCO^3 = Ca^3Al^2Si^2O^{10} + 3CO^2$
$= -116{,}800$ Cal.

Whence

$(2SiO^2, Al^2O^3) = 116{,}800 - 101{,}900$
$= \mathbf{14{,}900}$ Cal.

Calcium Silico-Aluminate (Gehlenite).—The first reaction given above enables us to calculate the heat of formation of the ternary compound so common in blast-furnace slags. Thus,

$(3CaO, Al^2O^3, 2SiO^2) = 3(CaO, CO^2) - 101{,}900$
$= 135{,}450 - 101{,}900$
$= \mathbf{33{,}550}$ Cal.

Kaolin.—By burning dry kaolin with carbon, the reaction was assumed to be during the burning off of the carbon:

$H^4Al^2Si^2O^9 = Al^2Si^2O^7 + 2H^2O$

And thence

$(Al^2Si^2O^7, 2H^2O) = 28{,}900$ Cal.
since $(Al^2O^3, 2SiO^2) = 14{,}900$ "
then $(Al^2O^3, 2SiO^2, 2H^2O) = \mathbf{43{,}800}$ "

Quartz.—It was attempted to measure the heat of transformation of quartz into amorphous, melted silica. The reactions were:

$10\,SiO^2$ (quartz) $+ Na^2CO^3 = -9{,}600$ Cal.
$10\,SiO^2$ (fused) $+ Na^2CO^3 = -8{,}700$ "

whence $10\,SiO^2$ (quartz) $= 10\,SiO^2$ (fused) $= 900$ Cal.
or $10\,SiO^2$ (fused) $= 10\,SiO^2$ (quartz) $= -900$

While some of the values given in this paper are only probably only first approximations, yet as such they will come, and will be gratefully received by both theorists and practical metallurgists. A résumé is as follows, giving the total heat of formation of the various compounds from their elements:

	Cal.		*Cal.*
(BaO, SiO^2)	= 14,700	(Ba, Si, O^3)	= 328,100
(CaO, SiO^2)	= 17,850	(Ca, Si, O^3)	= 329,150
$(2CaO, SiO^2)$	= 28,300	(Ca^2, Si, O^4)	= 471,300
$(3CaO, SiO^2)$	= 28,550	(Ca^3, Si, O^5)	= 603,050
(SrO, SiO^2)	= 17,900	(Sr, Si, O^3)	= 327,100
$(Al^2O^3, 2SiO^2)$	= 14,900	(Al^2, Si, O^5)	= 707,500
$(3CaO, Al^2O^3, 2SiO^2)$	= 33,550	$(Ca^3, Al^2, Si^2, O^{10})$	= 1,195,550
$(H^2O, Al^2O^3, 2SiO^2)$	= 43,800	(H^4, Al^2, Si^2, O^9)	= 927,420
(Li^2O, SiO^2)	= 65,100?	(Li^2, Si, O^3)	= 347,100?
(Na^2O, SiO^2)	= 45,200	(Na^2, Si, O^3)	= 326,100
(CaO, Al^2O^3)	= 450	(Ca, Al^2, O^4)	= 524,550
$(2CaO, Al^2O^3)$	= 3,300	(Ca^2, Al^2, O^5)	= 658,900
$(3CaO, Al^2O^3)$	= 2,950	(Ca^3, Al^2, O^6)	= 789,950

COPPER AND BRONZE.

Oxidation by Fusion.—The German Bureau for Testing Materials, at Berlin, has published the results of an investigation on the oxidation of melted copper-tin bronze (*Revue de Metallurgie*, July, 1905). The tin oxidizes first, forming SnO^2; or, if Cu^2O is already present, the tin added reduces it, forming SnO. The latter solidifies throughout the mass as infusible crystals, making the bronze pasty. The crystals can be recognized in such oxidized bronze, under the microscope, as lozenge-shaped crystals or pellicules. The addition of 1.25 per cent of phosphorus to a badly oxidized bronze caused fluidity, and the disappearance of the SnO^2 crystals, 0.16 per cent of phosphorus remaining over in the deoxidized alloys.

Aluminium Bronze.—L. Guillet, in *Revue de Metallurgie* for Aug., 1905, gives a very thorough study of the alloys of aluminium and copper. His conclusions confirm, in general, those of Le Chatelier, proving the existence of Cu^3Al and Al^2Cu, and of intermediate eutectics. The article is a very complete résumé of present information about these alloys.

Magnalium.—Some notes on light alloys are given in London *Engineering* of Jan. 5. Magnalium, which is the name given to an alloy of aluminium with magnesium, is manufactured in three varieties—*X*, *Y* and *Z*. Alloy *X* contains copper 1.70 per cent, magnesium 1.00 per cent, nickel 1.10 per cent, and antimony and iron in smaller quantities. Alloy *Y*, which appears to be intermediate in composition between *X* and *Z*, except as regards nickel, contains copper, magnesium, tin, lead, a small amount of iron and a doubtful trace of antimony. Alloy *Z*, in the form of soft sheet, contains tin, 3.15 per cent; copper, 0.21 per cent; magnesium, 1.58 per cent; lead, 0.52 per cent, and about 0.3 per cent of iron. Magnalium has two advantages over pure aluminium, for its tensile stress is decidedly higher, and it behaves better when subjected to turning. The tensile strength ranges from 8½ to 10 tons in the case of ordinary castings. Some further notes on researches by Barnett on magnalium are given. G. I. Petrenko (*Zeit. Anorg. Chemie*, 1905) has investigated silver-aluminium alloys. From the complete fusion diagram constructed by him on Tammann's method, he shows that two distinct compounds of aluminium and silver exist—namely, $AlAg_2$ and $AlAg_3$. The melting point curve falls from the melting point of aluminium to the eutectic point at 567°, then rises to the melting point of silver, on the rising point of the curve are two breaks at 926° and 11.15 per cent of aluminium, and at 775° and 7.72 per cent aluminium respectively, corresponding with the two compounds mentioned. Both exist in two polymorphous forms. The alloys, with from 7.72 to 11.3 per cent of aluminium take a good polish; those with from 0 to 7.72 per cent of aluminium are quite stable in the air.

RECENT METALLURGICAL PATENTS.

In view of limitations of space only a few miscellaneous patents can be noted in this issue. The bulk had to be reserved for our next issue.

Roasting Furnace.—For cooling the shaft and stirring arms of a roasting furnace, Ottokar Hofmann (809,953, Jan. 16) makes them hollow, and uses a vertical water-pipe extending through the shaft with branch pipes, each of which enters one of the hollow stirring arms through an opening at the inner end, while at the outer end a spraying means is provided by which the water is delivered to the arm in the form of a spray or plurality of streams. The water evaporates rapidly and the steam goes off upwards through the vertical hollow shaft. If more water is injected than will be evaporated, it flows down the inside of the shaft, accumulates at the bottom and discharges by means of an overflow. The chief advantage is that the degree of cooling is under the complete control of the operator.

Heat in Solution.—If in the treatment of an ore or a metalliferous sand it is necessary to use a hot solution, W. J. Jackson (252,879, Jan. 2) proposes to supply the heat by adding a material which results in an exothermic reaction, so that heat is produced chemically within the solution itself. In case of an aqueous solution he proposes to add calcium oxide, the heat being derived from the formation of the hydroxide.

Oxide Ores of Nickel and Cobalt.—The New Caledonian nickel and cobalt ores, which contain the metal in the form of an oxide, are of themselves melted with difficulty and are very unsatisfactory in their working in a shaft furnace, as the greater part of them are in the form of powder or fines. It has been the practice to mix sulphur with the ores and then briquet the mixture and charge the briquets into the furnace, in order to form the matte. J. Savelsberg (810,572, Jan. 23) endeavors to avoid the briquetting step in the following way: The ores are mixed with sulphur or sulphur-bearing materials (sulphates, gypsum, sulphides) and flux, if necessary, and a suitable quantity of pulverized coal or coke, and the mixture is charged into a converter on a bed of glowing fuel or ore, and simultaneously force-blast is blown through the mass. The charge is so heated that the sulphur reacts upon the ores containing the metals to form sulphur compounds of these metals, and the sulphurous acid is reduced by the glowing carbon again to sulphur. These reactions during the process produce sufficient heat to bring the mass to the sinter, containing sulphides of the metals, and the sinter is then broken and charged into a shaft-furnace and melted down as usually.

Iron.—C. Ellis (Nov. 7) proposes to produce iron by feeding the ore through a rotary kiln in a way somewhat resembling that used in the manufacture of cement, and by applying to the ore a reducing flame produced from pulverized fuel. The difficulty is that in order to secure complete reduction the flame must have a high temperature and reducing nature. To accomplish this, C. Ellis employs two flame strata above the ore. The upper one, which is more remote from the ore mass, is maintained at a high temperature by the supply of plenty of oxygen or air. The lower flame, which is between the upper flame and the ore, and comes into direct contact with the ore, contains an excess of combustible matter, such, for instance, as carbon monoxide, and has, therefore, marked reducing properties.

Zinc Coatings.—In order to get resistant durable coatings of zinc, it is necessary to prevent the formation of the so-called "hard zinc" and zinc oxide. This object is obtained, according to P. and A. Gührs (804,006, Nov. 7), by incorporating with the zinc about 0.5 to 2 per cent of aluminium and less than 1 per cent of a metal producing great fluidity, like tin. In the preparation of this alloy it is necessary to first melt the aluminium and then add the zinc.

Reception to Dr. Ostwald.

Previous to this departure for Europe, after a season of lectures at Harvard, Columbia and elsewhere, Prof. Wilhelm Ostwald, of the University of Leipzig, Germany, has consented to meet the members of the New York Section of the American Electrochemical Society and their guests on the evening of Feb. 2 at the Chemist's Club, 108 West 55th Street. A number of prominent scientific men will make short addresses, and a souvenir will be presented as a tribute of affection from his former students and as a recognition of his services to electrochemistry by his admirers. Members of the affiliated scientific societies of the city are especially invited to be present.

BOOK REVIEWS.

Soils and Fertilizers. By Prof. Harry Snyder. Second Edition. Easton, Pa.: The Chemical Publishing Co. Price $1.50.

Our farming interests are the largest and most important of our great country. Proper scientific methods of agriculture will make not only "two blades of grass grow where one grew before," but will render the often dull and monotonous life of the farmer intensely and happily interesting.

Much has been done in the past few years throughout the United States in this line, following the thoughtful paternalism of Germany, with changes to suit the individualism suited to American conditions.

Prof. H. Snyder, of the University of Minnesota, has been a leader in this propoganda, and in his book, "Soils and Fertilizers," has given us evidently the ripe fruit of his professional activity. We regret that the subject, though cognate to ours, is not one to which we can give an extended review.

Gas, Gasoline and Oil Engines. By Gardner D. Hiscox. New York: The N. W. Henley Publishing Co. Price $2.50 net.

This book deals broadly with the subject of internal combustion engine. The treatment is both logical and detailed. The author first reviews the history of past progress and sketches the theory of the different cycles, etc. Practical engineering details are next set forth.

We are particularly interested in the chapters on producers, blast furnace and coke-oven gas. We have been long of the opinion that at some future date power will be generated at a gas-engine by-product coke-oven installation cheaper than at most water powers, and that this will have a strong influence on the industrial development of coke-producing centers. This book confirms this belief. The utilization of blast-furnace gas, of course, has been given much attention in this journal since first founded. Both these lines of probable progress have important bearing on the electrochemical industry as well as on the metallurgical industry.

However, for perfect reliability the steam engine still holds undisputed sway, although this reign is drawing to a close, for economy of fuel is the order of the day, and for this reason, if for no other, we commend this book to the metallurgist.

Electrolytic Dissociation Theory. By H. P. Talbot and A. A. Blanchard. 84 pages, illustrated. New York City: The Macmillan Co. Price $1.25.

This excellent little book, written by two members of the faculty of the Massachusetts Institute of Technology, presents in a clear and concise form the principal facts on which the dissociation theory is founded. While intended as an elementary text-book for college students, it is recommended to others who wish to obtain a working knowledge of this theory and its application to the methods of qualitative analysis. The chapter giving the list of the more common ions formed with their characteristics will prove of great help to the student in applying this theory.

Acetylene for High-Temperature Work in the Laboratory and Workshop.

In our last issue (page 28) we noticed that the use of the oxyacetylene flame for welding and repairing iron and steel is now attracting considerable attention in Europe. In this country Mr. John Harris, of Cleveland, has been one of the chief pioneers of the use of acetylene for general heating purposes in laboratory and workshop practice. His exhibit at the acetylene convention three years ago, demonstrating the very high temperatures which may be easily produced by the oxyacetylene flame, first attracted the attention of the general engineering public to the great possibilities of this system. The results of his experimental work are now being placed on the market in form of a series of practical and handy heating devices by the Harris Calorific Co., of Cleveland, Ohio.

The oxyacetylene flame is, of course, exactly analogous to

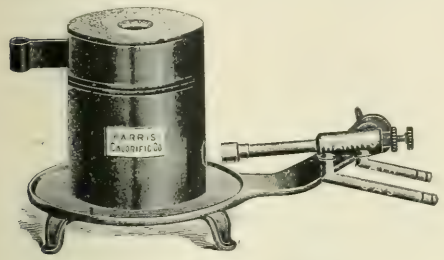


FIG. 1.—ACETYLENE FURNACE.

the oxyhydrogen flame, which has long proven useful in laboratories; the only difference is that acetylene gas is substituted for hydrogen. This has two important advantages. The first is that higher temperatures can be produced with the oxyacetylene flame than with the oxyhydrogen flame; the second is due to the very convenient generation of acetylene gas from calcium carbide and water. This is more convenient and safer than the use of compressed hydrogen, and all dangers due to the use of gases, compressed to high pressure in cylinders, are avoided if the oxygen is also generated in a similar way as the acetylene. This is now decidedly practical since the introduction of "oxone" by the Roessler & Hasslacher Chemical Co. (see our Vol. III., p. 255, and the paper of Dr. R. von Foregger, Vol. III., p. 376). The analogy between the generation of oxygen from oxone and water and of acetylene from calcium carbide and water is indeed significant.

However, for most purposes in the laboratory and workshop the highest temperatures which can be obtained with the oxyacetylene flame are not necessary, and in all such cases it means a great simplification, to use not pure oxygen, but atmospheric air. Of course, the air acts here simply as diluted oxygen, so to speak, since the oxygen in the air is the active element and the other constituents are inert. The acetylene is, therefore, used in combination with simple air blast. The heating devices described in the following notes are made by the Harris Calorific Co., of Cleveland.

Fig. 1 shows a small acetylene furnace equipped with a 10-ounce black-lead crucible. The casing holds the heat so perfectly that most refractory substances can be fused with ease, using about two or three pounds pressure air blast from an air compressor or foot blower. By means of the two-in-one valve the supply of gas and air is adjusted in right proportions A larger type of furnace, with a 2-pound crucible for use of metallurgists, jewelers, chemists and manufacturers of artificial gems, is operated with an air compressor or a very large foot blower.

The blow-pipes made by the Harris Calorific Co. are so designed as to combine in the simplest form every feature required of a perfect blow-pipe. One of their specialties is to direct the flame inside of a crucible for crucible work They may be used for heavy brazing in factories, also for heating tools for hardening, for welding light pieces in machine shops, and meet the requirements of glass workers and experimental laboratories. The tubing is never in the way during the operation of the blow pipe. Fig. 2 shows a hand blow-pipe, with which any desired length of flame may be obtained by adjusting the two-in-one valve in rear of the barrel.

Fig. 3 shows an acetylene Bunsen burner for various laboratory uses and for jewelers, dentists, etc. It is made of the lightest shell brass obtainable, to prevent it from getting hot. The holes are bored accurately to ensure perfect combustion.

Besides the great convenience of these devices, they have the advantage of low cost of apparatus and operation, and the further advantage of safety. Any acetylene generator using a low pressure meets the requirements of the blow-pipes. Portable apparatus is made for places where there is no acetylene available, especially for prospecting work.

* * *

This development is gratifying, since it opens a new and promising field to the acetylene industry, and thereby indirectly to the calcium carbide industry. The time is past when the use of acetylene was thought to involve inherent dangers. This prejudice was due to the early rush of would-be inventors into the acetylene-generator field; most of them undoubtedly possessed mechanical skill, but not the necessary knowledge of the chemistry of the problem of generating acetylene from calcium carbide and water. All this has naturally changed long ago. The less responsible manufacturers have been crowded out: and with the recognized acetylene generators now on the market, accidents are possible only by gross carelessness, just as is the case in electrical engineering or in the use of illuminating gas. An authoritative exposition of the chemistry involved in the generation of acetylene was given in an article by Mr. J. M. Morehead, on page 479 of our Vol. I., to which the reader may be referred.

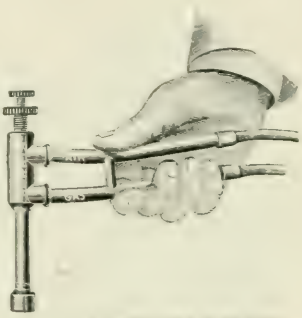


FIG. 2.—ACETYLENE BLOW-PIPE.

FIG. 3.—ACETYLENE BUNSEN BURNER.

Since, however, many engineers and chemists are still unfamiliar with the working of an acetylene generator a brief description of a successful modern type should be welcome as the conclusion of these notes. We have chosen Model N, made by the J. B. Colt Co., New York, which, without any reflection on other manufacturers, may undoubtedly be called one of the best developed and most carefully worked-out acetylene generators now available. It is made in capacities varying from 10 to 500 pounds of carbide. Like all modern types now on the market, it is on the list of permitted machines issued by the National Board of Fire Underwriters. The following description will indicate how exceedingly simple it is to install and to operate:

The Colt generator is shown diagrammatically in Fig. 4. It will be seen that it consists essentially of two parts, the acetylene generator proper at the left hand and the gasometer on the right hand.

The generation of acetylene from calcium carbide takes place according to the equation $CaC_2 + H_2O = CaO + C_2H_2$, and it is evidently possible to design a generator in either of the two following types: The first type comprises those in which a limited amount of water is dropped on to a mass of calcium carbide. This type is known as water-feed generators. The other type comprises those in which limited amounts of carbide are dropped into a large volume of water. They are known as carbide-feed generators. Both types may be so constructed as to operate automatically. In the first case just enough water drops on to the calcium carbide to produce the amount of gas required at that moment. In the second type just enough carbide drops into the water to yield the required amount of gas. But the second type has the inherent advantage that with a large amount of water it is possible to quickly dissipate the heat invariably produced by the chemical reaction. As was pointed out in Mr. Morehead's article above referred to, this is one of the most important requirements of a well designed acetylene generator. For this reason this second principle has been adopted in the Colt acetylene generator.

The automatic carbide-feeding device is at the top. Being

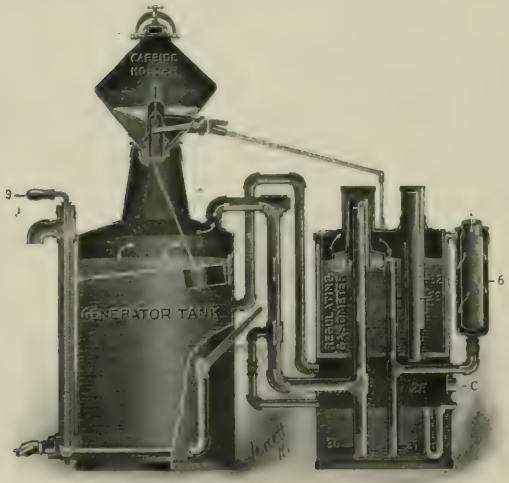


Fig. 4.—Acetylene Generator.

located within the carbide holder, it is not affected by the dampness of the water beneath. The feeding of the carbide is so perfectly controlled by the type of valve used that only sufficient is dropped into the water to produce the exact amount of gas that may be required by the burners in operation, whether it be one single burner or the maximum load, so that the gasometer may therefore be made exceedingly compact.

Below the feeding valve is a float cut-off, which serves the important purpose of insuring against a possible operation of the machine when it is not properly filled with water. When the water is at the proper level this cut-off is open; at all other times it is automatically closed. At the bottom of the generator is the water tank, of heavy galvanized sheet steel.

The second main portion of the whole apparatus is the gasometer, which is shown at the right hand of the diagram, and is also constructed of the best grade of galvanized sheet steel. It contains a floating trap, under which the gas is washed and cooled as it passes into the gasometer, and it prevents the gas from escaping when the generator is open for refilling. There is also an improved filter of large area, through which all the gas passes before reaching the service pipe. This filter prevents the passage of lime-dust to the pipes and burners.

The compartment at the bottom of the gasometer serves as the condensation chamber, into which all the condensation in the pipes of the generator is conducted. In this condensation chamber the gas service pipes are water-sealed. This compartment is connected with the open air, so that in case of a slight increase in pressure in the gasometer the excess gas is allowed freely to escape out of doors. It is impossible, therefore, even under conditions due to accident or carelessness, for the pressure to exceed the safety value. On the other hand, as was mentioned above, the automatic carbide-feeding device at the top of the generator works so perfectly that there will be no danger of an undue waste of gas through the exhaust or blow-off pipe from the gasometer.

Air Separation in Grinding Mill.

The adjoining illustrations show the design of the "Cyclone Pulverizer," built according to the patents of Mr. Sydney Straker by Messrs. E. H. Stroud & Co., of Chicago.

In one and the same apparatus a grinding mill, located at the bottom, is combined with a separation chamber, located above, in which the fine particles of the product of the grinding mill are separated from the coarse particles by a method of air-separation. By adjusting the air current the fineness of the product is varied.

The grinding mill works on the principal of attrition grinding, the work effected being due to the high velocity attained by the rotating star or mill wheel. The lumps of raw material, fed to the mill, are reduced by shattering each other to pieces and by the action of the beater arms. This is the same principle as is used in the Cyclone disintegrator of E. H. Stroud & Co., but the distinguishing feature is in the method of air separation.

The mill has no screens, the bottom of the grinding chamber is closed, and above the grinding chamber the air-separation chamber is located, with an exhaust fan on top thereof. The principle of the separation chamber will be easily understood from the illustration. The grist is flung up by the grinding wheel, through the double row of impinging bars in the top of the grinding chamber, into the separating chamber, and this brings it within the range of the exhauster.

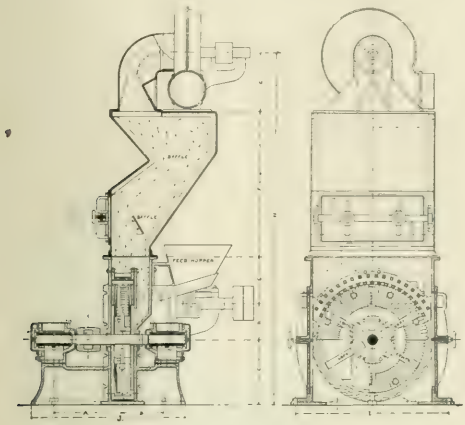
The separation is then effected by specific gravity; *i. e.*, the stronger the current of air induced by the fan the larger are the particles, and the weaker the current of air the smaller are the particles which are delivered from the pulverizer into the settling chamber; and it is only when the material has been reduced to a uniform specific gravity, or to the size of particle which the mill has been "set" to make, that the particles can be and are caught up by the current of air created by the fan and leave the pulverizer.

As the product-laden air rises up through the grinding chamber of the mill into the larger separation chamber it expands, and any heavier particles than the mesh desired fall back automatically by gravity upon and down the incline into the grinding chamber for further reduction. There is no residue. Everything that enters the feed hopper is ground up in one operation. The machine being once "set" to do any specific grade of work it will run day after day without further trouble.

Since a large volume of product-laden air is being delivered every moment from the pulverizer, the high suction-draught induced by the rotating mill wheel and by the fan keeps all passages clear, keeps the mill and its product cool and prevents

objectionable clogging with damp substances. The air will usually absorb and carry off about 10 per cent of moisture, if that much be in the raw material, and with this mill coal and some other materials are readily ground when the moisture runs as high as 15 per cent, so that previous drying of the raw material is frequently unnecessary. There are no sieves to clog.

To change the mesh or fineness of the product of the mill all that is necessary is to open or close a gate at the fan, to reduce or increase the air current, or to move a belt upon a



AIR SEPARATION IN GRINDING MILL.

pair of step-cone pulleys, to alter the speed of the fan to the number of revolutions which will give the required mesh.

The powdered product is discharged into a settling chamber, which may be a simple square room with hopper bottom, built of steel or wood, and with canvass sides to permit the air to escape when the product is delivered.

It will be understood that the mill is specially adapted to reduce dry materials to an impalpable powder, to grind paint pigments, dry colors, drugs and spices. In the Portland cement industry the mill may be used directly in front of each rotary kiln to supply both powdered coal and air to the kiln.

For the treatment of gold and silver ores by the cyanide or by the chlorination process, Messrs. E. H. Stroud & Co. build an extra heavy mill, and are planning also to build this machine in sections small enough for transportation by mules in the mountains. As this pulverizer receives the ore in pieces of large size, say cubes of 2 inches, no other machinery but a crusher is needed. The mill delivers a fine mesh or a coarse one, as desired. If much dust in the product is objectionable, the plant can be adjusted accordingly. If all dust is wanted, this can also be arranged. For use to regrind ores the mill also has many advantages.

Perfect Combustion by Down-Draft.

The problem of producing perfect combustion attracts, just at present, considerable attention. In view of the proverbial London fog, London was recently chosen as the ideal place to hold a conference on smoke abatement, which lasted several days and produced a number of instructive papers. That the matter is an important one in this country also, is indicated by the fact that the New York Section of the Society of Chemical Industry will devote one of its next metings to a general discussion of the same subject.

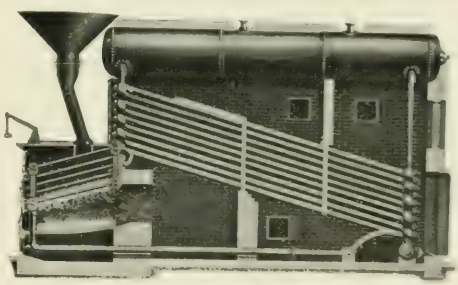
In this connection a brief description of the Hawley down-draft system of the Hawley Down-Draft Furnace Co., of Chicago, should be interesting, since it has already proven its commercial usefulness for quite a number of years in a great many installations.

When coal is fed on a [illegible] the moisture and volatile matter pass off, leaving each lump of coal an incandescent mass of coke. Much valuable matter passes off with the smoke and is lost, as fuel or heating value, unconsumed. Had sufficient air or oxygen been admitted to produce perfect combustion, no smoke would have been made; but sufficient air cannot be forced through the gate and coal to produce perfect combustion before the gases strike the cold [illegible] the boilers, and separated carbon is the result, which cannot be burned, and smoke is produced. Could sufficient air be introduced, and could this air be at a temperature sufficiently high, no smoke would be made.

The Hawley down-draft system is based on the principle to get perfect combustion by mixing hot air with the gases of combustion before they strike the cold boilers and form smoke.

In the Hawley furnace there are two separate grates, one above the other. The upper one is formed of a series of tubes opening at their ends into steel drums, or headers, which in turn are connected with the boiler, through which the boiler water continually and rapidly circulates. It is this upper grate only which is fired, and the down-draft of air being passed from the upper fire doors, the gaseous matter from the green coal consumed on the upper water-tube gates is passed right through this mass of fuel. Whatever gases escape unconsumed are then burned by the flame from the lower grate.

The lower grate, formed of common bars, is entirely fed by the half-consumed fuel falling from the copper grate; and as



DOWN-DRAFT FURNACE

the flame from this source ascends it meets the downward-burning fire from the upper grate, and the joint draft current passes through the flues in the usual way. The water tubes and connections give much additional heating surface.

The Hawley down-draft system may be applied to any type of water-tube, tubular or flue boilers. The adjoining illustration shows a Babcock & Wilcox boiler and Hawley down-draft furnace with automatic gravity coal feed.

The manufacturers state that when burning any grade of bituminous coal this system will prevent 95 per cent of the smoke; that it will increase the capacity of the boiler from 25 to 50 per cent, and that it will save from 10 to 40 per cent in cost of fuel. A special advantage is attended in the fact that the heating surfaces are exposed to practically constant temperatures. There is no alternate heating and cooling as is the case with the common setting, when the doors are opened to admit fresh charges of fuel.

Gas Reversing Valve.

It is evident from the large number of practical experiments and theoretical suggestions that in order to [illegible] making a suitable valve, that there are certain [illegible]

culties in connection with such devices. What is desired is an absolutely tight valve which will keep tight. The valves up to the present time either work with faced surface or with water seal. Both of these ways have certain disadvantages. At the high heat the iron warps after a short time and is then rapidly destroyed. In the case of the water seal, the hot gases come in contact with water and are considerably cooled off, a part of the water being carried in form of vapor by the gases to the furnace, thereby deteriorating the gas.

An important advance in this field has been made by Mr. Fischer, a well-known German engineer. This valve, which does away with the above difficulties, can be put in without changing the brick work and the present gas connection; it is of simple construction, as can be seen from the illustrations and from this description.

The Fischer valve consists of three main parts, an inner cylinder with a wall across and an outer cylinder, which is made in four compartments with four ribs, and then the base plate.

The inner cylinder *A* is pressed by the outside weight, shown in one of the diagrams, to the lower face *C* of the upper ring,

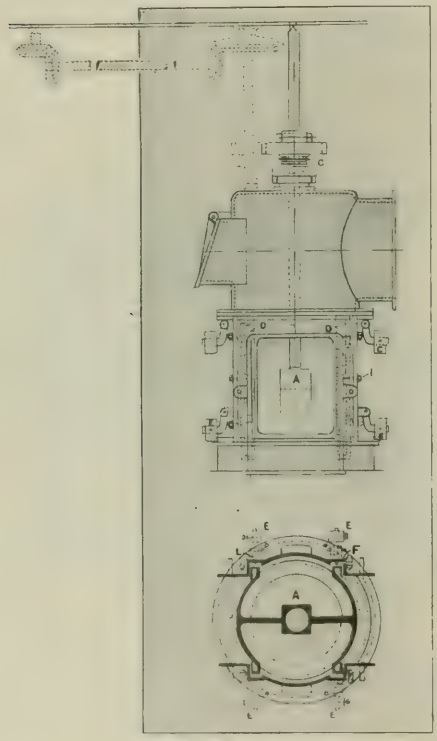


DIAGRAM OF GAS VALVE.

whereby the upper joint is kept tight. The space of play between the inner cylinder and outer cylinder can be made as desired, and the connection between the two cylinders is made by four ribs, which are effecting a tight joint on the sides. These four ribs are pressed by the weight *L* against the inner cylinder. The ribs *D* can be lifted by loosening the bolt *I*, whereby a connection is made from producer to stack for burning out the pipes, etc. The lower joint is made tight by having the inner cylinder resting in sand on the base plate. The pieces *L* prevent a throwing out of the center. The inner cylinder rests in the ball-bearing *O*, which allows an easy handling, and rests on the button, whereby an absolute horizontal position of the inner cylinder is effected.

It can be seen from this description that the Fischer valve is absolutely tight and does not allow an escape of the gas in any direction. Another important advantage is the point that during changing the position of the valve the connection between stack and producer is absolutely shut, and that the gas contained in the chambers and pipes is going to the furnace.

With the present valves even at a short shut-down the chambers and furnace cool off considerably as the slide valve in the stack is never absolutely tight.

The Fischer valve, on the other hand, is absolutely closed by a half-turn. By the use of this valve every leakage is prevented and also the loss and deterioration of the gas when changing the position of the valve. This results in a saving of fuel up to 30 per cent with the Fischer valve, so that it is to the interest of every steel furnace management to use this valve, which, besides the mentioned advantages, has a very long life and is subjected to very much less repairs than the present system.

The advantages of this valve are simple construction, a large saving in fuel, long life, small repairs.

This valve has now been on the market in Germany for about three years, and in that short time it has been introduced into practically every important steel plant in Germany, Austria and Italy. Some of the largest steel plants have en-

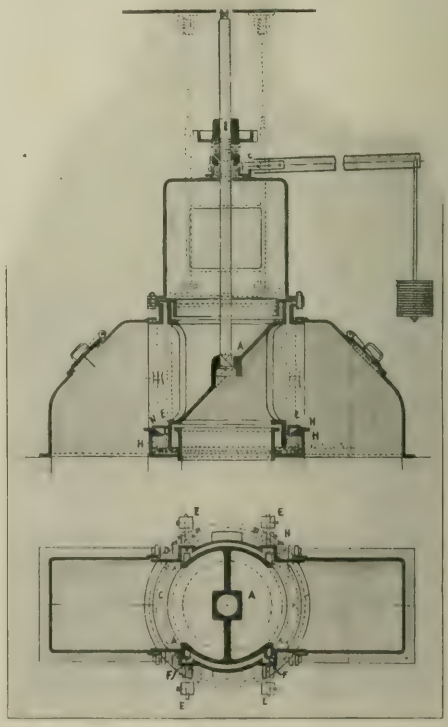


DIAGRAM OF GAS VALVE.

tirely removed their old valves and have replaced all of them to the very best of satisfaction with this valve.

These valves are built in this country by Dr. Oskar Nagel, engineer, 90 Wall Street, New York City, who holds the American rights.

Industrial Notes.

Niagara Falls.—The first permanent aluminium cable which supplies the Niagara, Lockport & Ontario Power Co. with power from the Ontario Power Co. was stretched across the Niagara gorge just below the whirlpool in January. The transmission line will extend from this point to Syracuse, and power is contracted for in Syracuse for March delivery. A contract has been made by this company for 25,000 horse-power to be used in the electrification of New York Central branch lines. The trunk lines, it is understood, will be electrified soon after. At the point where the transmission line strikes the American side, the Ontario Power interests have bought 1,200 acres of land just outside the city limits of Niagara Falls. A big industrial development is looked for here, as every inducement in the nature of cheap power and land and excellent transportation facilities is offered a prospective manufacturer. The Ontario Power Co. is developing 250,000 e. h. p., machinery for 50,000 e. h. p. being already installed. Most of this power will be available for American use. It is said that the Ontario Power Co. is about to close with several electrochemical concerns.

Calendars.—We have received a number of artistic calendars for 1906. The ferro-alloy calendar of Roessler & Hasslacher Chemical Co. has a picture of Patzcuaro Lake, Mexico. Messrs. E. A. Dempwolf & Sons, makers of hydrofluoric acid, York, Pa., have as picture the wedding breakfast of an Indian girl. The calendar of the National Carbon Co. shows a rather pretty girl with a muff. The calendar of the Charles E. Sholes Co. and J. T. Baker Co. has the official list of atomic weights for 1906. The Westinghouse Co. has again issued their extremely neat and useful diary, bound in black leather, and containing besides a number of maps, a large amount of concise information and useful data for electrical engineers.

Metallurgical Machinery.—Among the well-written and neatly, as well as profusely, illustrated new publications recently issued by the extremely active department of publicity of the Allis-Chalmers Co., of Milwaukee, Wis., are the following: Bulletin 1,408 gives a very full description of the Tremain steam stamp, which has been designed especially to meet the demands of those who own mining properties in the first stages of development, and enables them to establish a thoroughly good crushing, amalgamating, or concentrating plant of moderate capacity, in the shortest possible time, with the least material and for the least money. Bulletin 1,704 deals with lath mills and bolters, Bulletin 1,202 with the Reliance friction clutch, which has been designed for the heaviest work. It is of the disc type, in which a cast-iron disc is clamped between two continuous wood surfaces. The clutch is made with either three or six arms, arranged in sets of three. Each of the sets is connected to an equalizing ring by toggles, and this ring is free to move sideways, so as to equalize the strain on the three arms. The power is thus distributed uniformly over the entire friction surfaces, and the clutch does not have to carry any one-sided strains. There are no springs used in this mechanism, the pressure on the friction surfaces being set according to the load the clutch has to carry by means of adjusting nuts on the eye-bolts. The space between these nuts is filled by a ferrule, so that the nuts do not bind on the clamp ring, and, when set up, they form a positive lock. The sleeve carries the operating mechanism, and when the clutch is not in operation is the only wearing part.

Aluminium.—The Pittsburgh Reduction Co. has acquired a parcel of land in the neighborhood of Main and James Streets, Niagara Falls, as a site for a mill for the rolling of aluminium plates and wires. The company now has a rolling mill in New Kensington, Pa., but the market for aluminium plates and wire has increased so rapidly lately that the Kensington mill has not been able to supply the demand. The mill at Niagara will cover 4 acres of ground, and the dimensions of the main building will be 200 x 100 feet.

Pyrometers.—The reunion of the American Society of Mechanical Engineers, held on Jan. 30, had for its central feature a discussion by Prof. W. H. Bristol, of Stevens Institute, on thermoelectric pyrometers, adapted for general industrial purposes. The address was illustrated with demonstrations of the speaker's new type of thermoelectric pyrometer, which was already exhibited by Mr. F. F. Schuetz at the Bethlehem meeting of the American Electrochemical Society. We hope to give a more detailed description in one of our next issues.

Electric Equipment of Iron Mine.—A report from Duluth, Minn., states that the Penn Iron Mining Co. has made a contract for the erection of a hydro-electric plant and the equipment of its entire mines with electrically-driven machinery. The water power of the Sturgeon Falls, a few miles from the mines, will be developed, and a total of 4,000 hp. will be used. The General Electric Co. has the contract. This is believed to be the most complete electrical equipment that has ever been put into an iron mine. The Penn production is about 450,000 tons a year, and is gradually increasing. The cost of fuel consumed during the past year is estimated as more than $100,000, which item will, of course, be eliminated in future by the generation of electrical energy from water power.

Storage Battery Manufacture.—We are authentically informed that on account of the great increase in the demand for its "Bijur High-Duty" cells, the General Storage Battery Co. has decided to enlarge its factory at Boonton, N. J., by erecting buildings furnishing an additional 48,000 square feet of floor space; they will install hydraulic and steam or gas generating machinery aggregating over 1,000 horse-power.

Oil as Fuel.—A recent fully illustrated catalogue of the De La Vergne Machine Co. deals with "Hornsby-Akroyd" oil engines. There are now over 10,000 of these engines in operation. They use as fuel ordinary kerosene, crude oil or fuel oil. Their ability to operate successfully under varying loads with these oils, which can be bought for from 1 to 5 cents per gallon, make them valuable where such oils can be cheaply procured. A test of a 25-horse-power Hornsby-Akroyd oil engine showed the following consumption of oil in pounds:

	Per Indicated Hp-hour	*Per Brake Hp-hour*
Full load	0.62	0.74
Two-thirds	0.74	0.91
One-third	0.91	1.3
No load	1.3	

The actual cost of power for a 125-horse-power single-cylinder engine for 1 hp-year, equal to 3,000 hp-hours, is stated to be $25.37 (including crude oil at 3 cents per gallon, lubricating oil, attendance and interest on investment, depreciation at 12 per cent). Of the total cost of $25.37 the single item of the cost of oil is $9.27, at the rate of 3 cents per gallon. In many localities oil can be procured at 1.5 cents or less per gallon. In this case the cost of power will be correspondingly less

Coating for Iron Structures, Tanks, Pipes.—Several years ago the Solvay Process Co., of Syracuse, N. Y., found it necessary to procure a coating for the protection of their buildings, machinery, tank cars, submerged and structural iron work and piping, which extends for miles about their plants. The task was not an easy one, but was well worth trying, in view of the endless destruction and expense caused by the chemical fumes, gases, etc., in the atmosphere. The problem which the chemists of the Solvay Co. had to solve was the preparation of an elastic, impervious substance that would be rust, water, sun, weather, acid and fire proof. The outcome of these experiments is a material which has now been placed in the market under the trade name "Crysolite," Messrs. Fuerst Bros. & Co., of New York City, being the distributing agents. The analysis of crysolite proves it to be the purest possible form of carbon combined with a special oil as carrier, so that it forms a smooth, impervious, elastic, durable coating. Before being

placed on the market, crysolite was subjected to very severe tests, which it stood very satisfactorily. Thus Mr. J. D. Pennock, the well-known chief chemist of the Solvay Co., reports that wrought-iron tubes, 3 inches in diameter, were painted and exposed, after a baking process, to the action of ammonia and ammonium chloride liquors for three months, and when the pipes were removed they were found in perfect condition. In another test sheet-iron plates, 6 x 3, were painted with crysolite and exposed for two months in sea water, at the end of the two months the sheets showed no tendency whatever to rust, and the coating was in a hard, tough, yet elastic condition. The corrosion of iron has been a large factor in the past in increasing the cost of maintenance of plants in the chemical, metallurgical and mining industries, and there should be, therefore, an immense field of usefulness for a coating with the described properties in these fields.

Cyanide Process Developments in South Dakota.—It is reported from Deadwood, S. D., that the Clover Leaf Mining Co. is installing a powerful pumping plant to keep the water in the Uncle Sam mine, located south of Deadwood, under control. The company has a sixty-stamp mill at the mine, and has an abundance of ore of an average grade of $6 a ton. The mine has some rich streaks. In the early days it made several fortunes for the owners, who stamped out the gold in a crude way. Mr. Pierre Wiebaux, of Miles City, Mont., is president of the company. Mr. J. V. N. Dorr, of Terry, is drawing the plans for the remodeling of the Kildonan mill at Pluma for the Horseshoe Co. The Lundburg-Dorr-Wilson mill at Terry is among the most complete and profitable cyanide mills in the Black Hills. It saves the values in the slimes by the use of the Moore slime process. It is possible that the same methods used in the Terry mill will be put into use at the Horseshoe plant at Pluma. A description of the Moore slimes process and of the working of the Terry mill was given on page 257 of our Vol. III. Reports from Hill City state that the famous Clara Bell mine southeast of this place is rapidly becoming a big mine. A shaft that was put down a few weeks ago 300 feet encountered a vein where the ore averaged $16 a ton. Another shaft is being sunk further along in the dip. Several Ohio capitalists have bought the control of this mine from Herbert Bros., and a large mill is to be built.

Electric Hoist.—Electric Hoisting Machinery is the title of the latest illustrated bulletin issued by the Crocker-Wheeler Co. Electric hoists are now employed in all places and under all conditions where steam power was formerly used, and in many instances, such as in mines and tunnels, electric apparatus are the only type which has proven successful. As compared with steam or pneumatic hoists the motor-driven machines have the great advantage of being able to dispense with the boiler or compressor, which renders them much easier to transport and at the same time does away with the service of the firemen.

Personal.

Mr. Woolsey McA. Johnson has returned from a commercial trip to Mexico.

Mr. B. C. Hesse has opened offices as chemical expert at 90 William Street, New York City, making a specialty of organic chemistry.

Mr. Edgar A. Ashcroft, the distinguished British electro-metallurgist, is at present in this country and recently visited several electrochemical plants at Niagara Falls.

Messrs. C. C. Speiden and Marion Speiden, having for many years past been associated with Messrs A. Klipstein & Co., have recently severed this connection to become directors of the firm of Innis, Speiden & Co., successors to the old firm of Innis & Co.

Digest of U. S. Patents.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

Production of Nitrogen Compounds.

No. 569,122, Oct. 6, 1896, A. A. Naville and P. A. and C. E. Guye, Geneva, Switzerland.

The apparatus is a eudiometer designed to operate continuously and to expose every portion of the gases to the action of the electric spark. It comprises a vessel having opposed tubular spark terminals, between which are a number of intermediate tubular electrodes, all of these tubes being connected to a single discharge pipe, and the arrangement being such that the entering gases flow simultaneously into and through several tubes connected electrically in series and serving as discharge surfaces. The terminals and intermediate tubes may be of carbon or of metal, as of platinum, bronze, copper, nickel, ferro-nickel, etc., the selection of the electrode material depending on the character of the gas reaction sought. Examples given are the production of nitric acid from moist air in the presence of carbon electrodes, and the production of acetylene from hydrogen in presence of carbon electrodes.

No. 569,325, Oct. 13, 1896. Paul Danckwardt, New York, N. Y.

Fused alkali chloride, as NaCl or KCl, or a mixture of these with calcium chloride, is electrolyzed between a carbon anode and a sheet-metal cathode, the latter constituted by the walls of the containing vessel, the top and bottom of the vessel being suitably insulated. The anode is surrounded by a perforated clay tube, which serves to lead the chlorine from the apparatus. The liberated alkali metal rises to the surface, and is there brought into contact with carbon and nitrogen, for the purpose of producing the corresponding cyanide. The carbon may be present as a body of coal floating upon the fused salt, or as a gaseous compound as carbon monoxide. The nitrogen may be introduced as free nitrogen, as a nitrogen compound, as uric acid, or preferably in the form of ammonia gas. The reaction between carbon, sodium and ammonia proceeds with separation of hydrogen, in accordance with the equation

$$C + Na + NH_3 = NaCN + H_2$$

No. 579,988, April 6, 1897, Carl Kellner, Vienna, Austria-Hungary.

Cyanides are produced by direct reduction of nitrates or nitrites by means of an electric arc between carbon electrodes, the reaction proceeding according to the equations

$$(1)\ KNO_3 + 4C = KCN + 3CO$$
$$(2)\ KNO_2 + 3C = KCN + 2CO$$

In order to facilitate the reaction the nitrate or nitrite may be mixed with carbon. The apparatus may comprise vertical or horizontally-disposed carbon electrodes, with any suitable means for feeding the charge to the zone of reduction.

No. 607,943, July 26, 1898, H. Mehner, Berlin, Germany.

Cyanides are produced in an electric furnace by reaction between alkali or alkali earth compounds, carbon and nitrogen, the preferred material being the alkali or alkali earth carbonates. Owing to the high temperature the resulting cyanide is vaporized; the vapors are withdrawn from the furnace at a point immediately below the zone of highest heat, and are condensed in a column of coke, contained in a shaft above the electric furnace. This coke, containing the condensed cyanide, is steamed at suitable temperature, whereby the cyanides are converted into ammonia, leaving a residue consisting of the coke and the corresponding carbonate. This forms a suitable charge for the electric furnace, and is admitted directly thereto for the continuance of the reaction. The construction is such that the condensation occurs only in the remoter layers of the coke column, and arrangements are made for preliminarily heating the air which serves as a source of nitrogen supply for the reaction furnace

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Electrolytic Copper and Lead Refining.

In an editorial note of our issue of last [illegible] we discussed copper smelting as a method of recovering gold and silver and gave some figures indicating the rapid commercial growth of an electrolytic copper refineries. A suggestive illustration of this development is the modernizing of the Mountain Copper Co., in California. This is a very progressive concern, as is indicated by the fact that as time passes they became chemical manufacturers, producing sulphuric acid and selling fertilizers. They have a large new plant at Martinez, where the copper ores, after roasting and recovering the sulphur as sulphuric acid, will be smelted in the blast furnace with silver ores from Nevada, using the copper as a carrier for the silver. This is, of course, to be followed by electrolytic refining. In view of the growing Pacific trade of this country, it may be expected that the company will have full success as the due reward of its progressiveness. Another new and very interesting development in electrolytic refining is reported in the field of lead. The United States Mining, Smelting & Refining Co., owning the De Lamar Copper Refinery, in Chrome, N. J., copper smelters in California and lead properties in Utah, could not, it is said, make satisfactory terms with the American Smelting & Refining Co., and will now become its competitors by erecting a lead plant at Hammond, near South Chicago, in the heart of a good lead center. Mr. A. G. Betts' electrolytic lead refining process—well known to our readers—will be employed. This will be the third large commercial application of this process, the first and oldest one being in Trail, B. C., the second in Newcastle-on-Tyne, England. The great purity of electrolytic lead and its freedom from bismuth are features of the process. If the carefully devised slimes treatment, described last year by Mr. Betts in our columns, turns out on a large scale as successful as is hoped for, the recovery of the values will be complete and the Parkes process will be put most decisively on the defensive. In comparing electrolytic lead and copper refining, attention should be paid to the much higher electrochemical equivalent of lead, compared with copper. This means that the power cost will be a much lower item in lead refining than in copper refining.

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Power Plant Economics.

A paper recently presented on power plant economics before the American Institute of Electrical Engineers, by Mr. Henry G. Stott, superintendent of motive power of the Interborough Rapid Transit Co., of New York, is exceedingly valuable and interesting. The first feature is the author's analysis of a modern steam power plant from the coal to the bus bars in the same way as it is now customary to analyze a metallurgical or chemical process on the basis of thermo-chemical data. Mr. Stott's analysis is based upon tests [illegible] and is probably one of the most efficient plants existing [illegible] using reciprocating steam engines, and [illegible]

present state of the art. The total final result of his analysis is that out of the heat units supplied in form of coal, 10.3 per cent are delivered to the bus-bars in form of electrical energy. Almost 90 per cent are, therefore, lost, but it is most interesting that of these total losses the whole electrical losses do not amount to more than three-tenths of 1 per cent. This point is significant, since it indicates one of the chief underlying reasons which have made the modern electric power development possible. The electrical equipment itself, the design of generators, transformers and motors has practically reached ideal perfection, as far efficiency is concerned. It is this efficiency of the electrical equipment, together with its flexibility, which lies at the bottom of our systems of electric power generation and distribution, and which has made possible the large modern "syndicate power plant" with its supply of power in bulk to a number of sub-stations. On the other hand, this fact should teach a lesson to users of electric power; in discussions of projects it is not unusual to make much of the superiority of one type of electrical machine over another type, the difference being generally something like 1 or 2 per cent. If we look on the power-plant as a whole, this difference will be perfectly insignificant in view of the above facts. To increase the efficiency of the power plant, we should learn to be satisfied with the electrical machinery as it is and to look for improvements in other directions.

* * *

Mr. Stott's energy balance sheet shows three large sources of loss. The loss in boiler radiation and leakage is 8.0 per cent, the loss of heat to stack is 22.7 per cent, and the heat rejected to the condenser amounts to 60.1 per cent. New methods of boiler setting may diminish the first loss. The loss to stack can be considerably diminished by the use of more scientific methods in the boiler room. Care should be taken not to admit too much air to the combustion chamber, and in this respect a CO_2 recorder, when watched by the fireman, has proven a very valuable instrument. The third and largest loss—the heat rejected to the condenser—is, of course, intimately connected with the thermodynamics of the reciprocating steam engine, and it is superfluous to dwell here on this point. The engine losses may, however, be somewhat reduced by the use of superheat. In the whole, Mr. Stott concludes that by applying judiciously the above remedies it will be possible to increase the total thermal efficiency from 10.3 to 14.44 per cent. A further important improvement may be obtained at a small cost by running the present types of reciprocating engines at high pressure, adding a steam turbine in the exhaust between the engine and the condenser. These methods appear to exhaust the present possibilities of the reciprocating steam engines. But, if we leave water-power plants outside of the realm of this discussion, there are now the steam turbine and the gas engine looming up as formidable rivals, and it is with respect to them that Mr. Stott makes some very interesting suggestions.

* * *

The steam-turbine plant has an inherent economy 20 per cent better than the best type of reciprocating engine plant, not so much due to its higher thermal efficiency as to a variety of other causes. Moreover, during the last years the steam turbine has shown itself to be able to do in serious actual work all that had formerly been claimed for it on paper. The advances made by the gas engine have been steady, but slower. It represents undoubtedly at present the most efficient motive power. Its mechanical defects have now been overcome, but there still remains that inherent difficulty—lack of overload capacity. On the other hand, the advances made in gas producers have been so remarkable recently and the possibilities of the use of the waste gases from blast furnaces are so great, as has been repeatedly pointed out in these columns, that a more rapid development of the use of gas power is bound to come in near future. Mr. Stott makes a simple, but ingenious suggestion concerning a combination of steam and gas power. A combination of gas engines and steam turbines working in multiple in a single plant offers the possibility of producing the kw-hour for less than one-half its present cost, and overcomes the disadvantage of lack of overload capacity of the gas engine. The gas engine will carry the constant portion of the load and the steam turbine the fluctuating load. Both types of machines are well adapted to do their share, and both will, therefore, be well adapted to work together for ordinary service. Such a technical combination of steam and gas would be quite in line with modern commercial developments, competition being replaced by consolidation.

Conditions in the Spelter Market.

It is extremely difficult to make anything but most vague and general prognostications about the commercial future of any metal. Metal brokers as a rule rely mostly on temporary conditions, such as "offerings" and "visible supplies," and base their diagnosis on daily symptoms rather than on the general physical tone of the body of the patient. Nevertheless, broad generalizations are always of value, if adherence is kept to the idea that anyone of several circumtsances may alter the case substantially. Now, in the zinc business, it may be remarked that the conditions are unique and radically different from those of any other metal. In the first place, in the past year over 60 per cent of the metal was produced from "concentrates" from the Joplin mines. In this ore market perfect and free competition exists. There are at times as high as 300 producing mines offering in the market zinc concentrates to the weekly value of $250,000. There are seven or eight smelting concerns who compete most strongly for the product of these mines. The scattered and "pockety" nature of the Joplin deposits causes the diversification of the mining, and cheap production is due to close personal attention of the mining boss. Hence, we do not have exhibited the centralization of management, which has been the well nigh universal tendency of American industrial development in the past ten years, and cheap production is the criterion on which all "trusts" must be judged.

* * *

In the treatment of ores for the production of metal (zinc smelting is an absolute misnomer), the smallness of the unit, the 50 pound retort, produces exactly the same state of affairs. Conditions are such that despite the efforts of the irrepressible and buoyant promoter no "zinc trust" has been formed as yet, though strenuous efforts have been and are now being made to such an end. The final result is that free and open com-

petition, both in ore market and in the metal market, allow quick and free play of economic forces. Consequently we see, often in the price of metal, sharp weekly fluctuations as well as movements caused by underlying conditions. There are "choppy seas" and "ground swells" as well as large ocean currents. The character of the consumption of the metal is also unique. About half of the spelter of the country is absorbed by the galvanizing trade and about 30 per cent by the United States Steel Corporation alone. The steel business is proverbially a barometer of general trade, and it is apparent that the spelter business responds to the steel business with which it is so vitally connected. Therefore, the market for the metal is supersensitive and shows at once any slight recession or advance in the country's prosperity.

* * *

The advances in the metallurgy of zinc and in ore dressing have opened up the large fields of Wisconsin, Leadville, New Mexico, Utah, Montana and Mexico, so that Joplin is slowly but surely losing its pre-eminence as a factor in the market, though the camp is now and will be for some time most prosperous. In time, also, these efforts to consolidate several large companies into a "trust" will find success, for it is the logical outcome of the situation. Stability is always needed to ensure perfect efficiency of industrial production. With respect to the market, there are several general and broad reasons why the consumption of galvanized sheets and other forms of galvanized iron will increase in the future. First, manufacturing plants, now being built, are erected as permanent structures. This is due to the change of the United States to a country where solid construction is a paying proposition. We are no longer considering cheapness of the first installation as the prime requisite, but are looking for the highest efficiency for a long term of years. Thus galvanized sheets are used in larger quantities each year. In the next place, the general expansion of business has forced up the price of lumber inordinately. Galvanized sheets are, everywhere, the best substitute for wood. Indeed, Western farmers are using them at present for roofing both barns and dwellings. A third consideration is that spelter is usually consumed, and once leaving the metal house is lost forever. Vast quantities of lead, iron, steel and copper return as "scrap" to the market, quite the contrary to the case with zinc. Taken in its entirety, the zinc business from the mine to the market is in good shape, and there is every indication that progress in the treatment of zinc ores will increase the production of spelter, so as to meet the large probable future basic demands.

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The Old Pioneer Spirit of America Manifested in Our Life of To-Day.

Every American delights to "rough it" for a few weeks in each year. Contact with nature is a return to primitive conditions and, by the psychology of play, in camp-life atavistic instincts are allowed free play in the individual of any country. But especially, here in America we are all descendants from pioneers, from men and women who dared break away from fixed society of Europe and settle in a country of unknown terrors. This "selection of emigration," as we may term it, has brought about a nation in which all the best and most adventurous blood of the world has been centered. Thus, we are, as a nation, [illegible] immortal spirit of the [illegible]. And it is but natural that [illegible] rude and trying but delight[illegible] spirit, such as our play, is [illegible] is our real national genius—a genius peculiarly our own. The same pluck and intelligent daring that pervaded the minds of Lewis and Clark in their expedition to [illegible] every recurring [illegible] gigantic United States Steel Corporation. The same fighting blood that enabled some poor farmers and backwoodsmen at Bunker Hill and later at New Orleans to repulse and shatter the flower of the English army is seen on the football field, where our young men gladly risk life and limb to uphold the honor of some modern monastery of learning—a strange disharmony of endeavor. This same daring and bull-dog grit evidenced itself in our dreadful fraticidal Civil War, and also in Dewey's "walk-over" at Manilla. In all philosophy, there is unity and the persistence of this pioneer strain in America is in accordance with the general nature of things.

* * *

In metallurgy, as well as in business, we find the same spirit of the pioneer. It takes no whit less courage to build a blast furnace that makes in a day 950 tons of pig iron than to open up a new country and repel the Indian raiders. Life in the backwoods develops resourcefulness and ingenuity. The men whose fathers built mills entirely of wood, wooden pins, shafts, gears, etc., make the automatic charging machine for the open-hearth furnace. The patent office is filled with the ideas of the national pioneer mind, working to-day throughout our whole land. We would also trace that imaginative and spiritual quality which is our best trait to the close contact that the leaders of our thought and life have ever had with nature. The modern artistic triumphs of America, such as those of Whistler, Sargent, St. Gaudens and MacMonnies show the freshness and originality of youth. It is, however, in human achievements rather than in expression of human desires and crises that America shines. No one who has ever seen the bones and tufts of hair of the lonely prospector bleaching beside those of his faithful horse, needs any picture to call up the sadness of life. He knows the real thing. To the old age of a nation belongs the flower of expression and art.

* * *

In new developments and in new enterprises the influence of America's early struggles is apparent. It is no more masterly achievement to unite two large copper furnaces and put through that new furnace 1,500 tons of ore. It is the triumph of an idea, and German philosophy has taught us that the existence of the ideal in everyday life. The mammoth power in the Mesabi iron district is another instance of [illegible] and youthful imaginations, of our quick readiness to make a change and "to do something big." Our [illegible] and physical strength [illegible] medicines to kill us, all are type characteristics of America, but especially of the growing, busy and [illegible]. We [illegible] the bright lexicon the world calls youth [illegible] as fair." We are as a nation [illegible] those of crude, virile youth. But we [illegible]

A. E. S. Election.

In our correspondence columns we print two letters, one by Prof. C. F. Burgess and the other by Mr. C. J. Reed, requesting that they be not considered candidates for the presidency of the American Electrochemical Society. We join Prof. Burgess in the hope that Mr. Carl Hering, who received the largest number of nominating votes for president, will be chosen as the next president by a practically unanimous vote, to which he is entitled as one of the founders and most active members of the Society. It is hoped, however, that the Society will not be entirely deprived of the services of Prof. Burgess and Mr. Reed on the Council. Since their terms as vice-presidents will expire this Spring, they cannot, according to the constitution, be re-elected vice-presidents, but are eligible for election as managers. In view of the persistent and able work which they are doing for the Society and for the profession which it represents, it probably can be taken for granted that those who proposed Prof. Burgess and Mr. Reed this year, will, with many others, put forward their names at future dates for the presidential nomination.

The Electric Furnace of Heroult for Iron Reduction at the "Soo."

Our readers are well acquainted with the praiseworthy and systematic endeavor of the Canadian Government to test the possibilities of the electric furnace in the metallurgy of iron and steel, with a view of making both the water-powers and iron ores of Canada available for a new important industry in that country.

As has already been mentioned repeatedly in our columns, the experimental tests which are being made at Sault Ste. Marie under the direction of Dr. Paul Heroult, of La Praz, France, and under the supervision of Dr. Eugene Haanel, Superintendent of Mines, Ottawa, representing the Canadian Government, relate to the broadest and most difficult problem of applying the electric furnace in the iron and steel industry, namely, for the reduction of iron from the ore. It is the more gratifying to read the following telegram, which is reported to have been sent by Dr. Haanel on Feb. 24, to Hon. Frank Oliver, Minister of the Interior, Ottawa, Can.:

"Successful demonstration of all points stated in my memorandum on electric smelting of Canadian iron ores requiring investigation. Output greater than figure adopted by Harbord in report of Commission. Successful smelting of magnetite and desulphurization of pig. Successful substitution of charcoal, and, therefore, of peat, for coke. Consumption of electrodes insignificant. Production of nickel pig of fine quality from roasted pyrrhotite. Forty tons of pig have so far been produced. Process admits of immediate commercial application. Experiments will be completed in about two weeks."

More detailed accounts of the experiments are not yet available for publication. The figures of the report of Harbord, referred to in the above dispatch, were published and discussed in our issue of Dec., 1904, page 483 and following.

Thermochemistry of Chemical Combination.

In an interesting paper, recently presented by Prof. J. W. Richards before the American Chemical Society, the pertinent question is proposed: The combination heat of H, and O is 30,683 calories to vapor, 69,000 to liquid, 70,440 to solid water, which of these represents best the true chemical heat of union of the elements? Dr. Richards' answer is: The first, since the second and third figures contain amounts of energy of purely physical origin, namely, the latent heats of vaporization and of fusion. In general, comparisons and discussions of thermochemical data should be made on the basis of the heat evolved in the combination of the *solid elements* to the gaseous products. As a further refinement, correction should be made for the external work done when the gaseous product is formed, $Q = 2T$. But even then the temperature is still arbitrary, and the ideal way would be to refer the combination heat to the zero point of absolute temperature.

The second part of the paper deals with a discussion of the formation heat to dilute aqueous solutions and the well-known additive law which holds there. On this thermochemical basis Prof. Richards points out that the inference is inevitable that the state of being in dilute solution is "the state of most uniform chemical combination." Very complete tables are given of the "constants of heat of chemical combination." To call them ionisation heats, as Ostwald does, is a misnomer, according to the author.

Tendencies in the Foundry Industry.

Dr. Richard Moldenke, the ever active secretary of the American Foundrymens' Association, delivered a suggestive paper on this subject before the New England Foundrymens' Association in Boston.

In his introductory remarks Dr. Moldenke emphasized that there is manifest to-day a tendency toward refinement in method, little dreamed of by the most ardent iconoclasts of a decade ago. Perhaps our peculiar situation in this country, where the foundry is engaged in pushing specialties to the limit, has enabled us to take the lead in many directions. Great stress is laid, in our strictly modern establishments, upon system and organization, and there is a strong tendency towards specialization of our shops. Since specialty work means the production of the same casting in very large quantities, we can see why machine molding in its various phases is beginning to play such an important rôle. There is also a noticeable tendency to economize shop room, and hence, molds are piled as high as they can be conveniently poured. Dr. Moldenke suggests that the idea should be followed out to its logical end wherever work is made in sufficient quantity. This would seem to be the stacking up of the molds in fairly deep foundry pits, the iron being brought in large ladles by traveling cranes and poured from the bottom of the ladle.

Greater attention will ultimately be given to the preparations of the molding sand. "Our European brethren are far ahead of us in this respect, every foundry of importance grinding its sand to a given standard, then passing it through mixers, which incidentally fill the perfectly uniform material with a maximum of air, thus cushioning the sand and keeping it as open as it may be. When a highly refractory molding sand is used, this treatment assists in turning out the smoothest kind of work."

Dr. Moldenke then referred to the growing tendency to perform as many operations about the foundry as possible by mechanical means, and also mentioned the growing demand for foundry standards. The makers of molding machines will be greatly benefited by the desired standardization of flasks.

Turning to the melting and the closely allied laboratory, Dr. Moldenke pointed out that the strongest marked tendency of the day in the industry is the production of higher grade castings by the addition of steel scrap, or in other words, the reduction of the total carbon in cast iron, making the crystalline structure more closely adherent, and hence the whole casting stronger. Here the skill of the melter is joined with that of the chemist, so that value is received from the higher priced stock charged into the cupola. The cupola process is being studied more carefully, and experts in that line can produce results close to those of the air furnace, or open hearth.

This tendency has somewhat retarded the introduction of air furnaces in general foundry practice. However, taking into consideration the demand for better irons, better coke, better melting methods, the specialist founder will soon tire of buy-

ing high-class material and taking big chances in spoiling it. He will naturally revert to the air furnace as better able to give high-class results from high-class material, and the consumer will be glad to pay the difference.

Whereas, once the melting point of the various cast irons formed the one absorbing topic for articles and discussion, at present it is the elimination of sulphur in the cupola. This is a good sign, and even if one man finds that he cannot get the results of the other, and therefore we are at present quite at sea regarding a desirable method, yet it shows a tendency, and a good one.

With more light on this question, it is but a step to work on others, and finally we may get down to a closer study of the burning effects manifested in cupola and air furnace practice, which Dr. Moldenke holds, are the most important by far of all the phenomena of the foundry melting processes, granted that the chemical composition is otherwise correct.

This leads to the use of the ferro-alloys in the foundry to correct the evils we know, and others we have a vague apprehension of. To-day the use of ferro-manganese and of aluminium is quite extended, and is really a help in many instances. With the poor pig-iron, and especially the very poor coke the smaller founder is forced to put up with, the use of either of the above mentioned correctives is to be recommended. Unfortunately ferro-manganese is hardly powerful enough to do much good in the foundry, unless the mixture has a goodly proportion of steel in it. This means a higher melting point, and with higher temperatures than ordinary cast iron is apt to be poured at, much of the oxidation is removed, and we avoid the pin holes, other weakening and unsightly effects, which seem to be somewhat the rule at the present day. "The real foundry alloy, however, is still to be discovered and made commercially, and the day cannot come too soon for this desideratum to be realized."

Dr. Moldenke looks forward to great improvements in the making of producer gas. Whoever has worked with gas instead of coal, will stick to the former until the price soars out of sight. Just how this tendency will work itself out in the foundry is hard to say. Whether it means the use of the open-hearth furnace in the big foundries, or the making of electricity with the gas engine in the smaller foundries for use in melting, time will determine, but it is well to keep this development before us, and to profit thereby.

This led up to the consideration of electric smelting. "The Canadian Government is doing much to foster the development of electric smelting. The makers of the latest types of electric furnaces are constantly getting better efficiencies. Now, with a cheap gas, in a gas engine, and for the making of very small castings in small quantity, there is no reason why the small steel, malleable, and even gray casting industry, not to speak of the brass and bronze contingent, should not melt their stock in the electric furnace. The higher price of the work sold will probably stand this. This tendency will bear watching. I have recently noticed a neat suggestion by Dr. Waldo along this line, where he proposes to make Bessemer steel rings, or suitably-shaped small ingots, these to be of the composition of the steel casting. Now, by charging these small ingots into the electric furnace, one obtains a fluid steel of the proper composition to cast. I would suggest as an improvement that the ingots be pre-heated up to the highest point possible, without injury to the metal; say just as the ingots for rails or structural steel are treated. This would leave for the more expensive electric current only the finishing heat, necessary to melt and superheat to the point right for pouring."

Finally, Dr. Moldenke discussed the general problems of the foundry, in which there are also marked tendencies for a change. This will probably affect only the very large industries for the present and next generation, but what may come after is hard to say. Foremost stands the rapidly increasing use of direct metal. "This is more marked across the water, but is bound to come here also. The big segments for the tunnel shields and linings should [illegible] remelted metal. However, this does not mean that it is advisable from the first to [illegible] vast contracts. Here again, we [illegible] use of steel scrap in gray iron castings [illegible] total carbon. I would not advocate the [illegible] just as it is, but to have the stream of metal from the furnace go into large heated mixers, where [illegible] in fact, more easily attained uniformity than [illegible] with the cupola. Now, into this mixer [illegible] successively, quantities of steel scrap [illegible] up to almost melting. This accomplishes a two-fold [illegible] First, it gives a reduction of total carbon without lowering the temperature of the metal in the mixer too [illegible] corrected at once by the heating method used as well as the comparatively large quantity of metal that [illegible] mixer all the time. Now, the first result is the making of a high-grade cast iron. This iron should be good enough for all the purposes the process could be used for, such as the making of pipe, tunnel segments, ingot-molds, etc.

"The second effect of the addition of the preheated steel scrap is that the problem of 'kish' is done away with. This is the bug-bear of the direct metal founder, and to [illegible] retarded the introduction of direct metal for important work so much that at one time it was thought out of question to think of ever going back to this century's old way of making castings.

"What then will be the result of this tendency? It means that blast furnaces will be located wherever the raw material not only are favorable, but where there is a good shipping point for pipe, possible car wheels, and not impossible, malleable castings for car work. The furnace proposition will then become one of allied industries to give an outlet for the pig-iron made, which will fit these uses, and all the pig iron not of composition to go into the various mixers will be put upon the market. While this seems visionary, it is nevertheless a fact that one very large combination is about to break ground for just such an establishment, and we may hear from direct metal pipe very soon."

CORRESPONDENCE.

A. E. S. Election.

To the Editor of Electrochemical and Metallurgical Industry:

Sir:—Will you please announce my request to the members of the American Electrochemical Society that my name be withdrawn from the list of nominees for the presidency of the American Electrochemical Society. I am impelled in making this request by the hope that Mr. Carl Hering will be chosen as the next president with a practically unanimous vote, such as his effective and enthusiastic services in the organization and conducting of the Society entitles him to. I am further impelled by knowledge that should the remote possibility of my election to the office be realized, it would be impossible for me to administer the duties thereof during the coming year in a manner such as the needs of the Society require.

Madison, Wis. C. F. Burgess.

To the Editor of Electrochemical and Metallurgical Industry:

Sir:—Will you kindly announce through your journal my declination of the nomination for president of the American Electrochemical Society, as it would be impossible for me to serve if elected. My declination would have appeared on the official circular had the usual opportunity been afforded. I did not know that my name was to appear as a nominee until after receiving the printed circular, and I take this as the only available means of notifying my friends not to waste their votes.

Philadelphia, Pa.

A Co-operative Analysis of a Copper Slag.

To the Editor of Electrochemical and Metallurgical Industry:

Sir:—In the February issue of this journal you print my reply to Mr. Thorn Smith's article on the above subject (wherein he severely criticizes my zinc method) and Mr. Smith's rejoinder thereto. As I still maintain my position at every point, I feel I am fairly entitled to reply to Mr. Smith's latest statements.

I decline to admit any "difficulty or misunderstanding" in the matter of precipitating copper with aluminium. Materials may doubtless be encountered that present difficulty when treated by my regular method, but Ducktown slag is not one of them. True, I cannot take an unusually large amount of the slag and precipitate the copper therefrom in 7 minutes by my regular method, designed for the treatment of 0.5 gram. I can, however, take 0.5 gram and do so, as demonstrated by actual experiment.

If I desire to start with a larger amount of such material I may vary the method to correspond. For instance, taking 1 gram of Ducktown slag, the addition of 2 grams of 20-mesh zinc to the usual precipitating arrangement results in the complete precipitation of the copper in 7 minutes' boiling. The zinc is quickly dissolved, and serves mainly for the rapid reduction of the large amount of ferric iron in the solution.

Mr. Smith does not consider 0.5 gram of slag a sufficient amount to take. I beg to differ on this point. My results are sufficient evidence of the correctness of my position. Three determinations show an extreme variation of only 0.02 per cent, viz. 0.85, 0.83, 0.85. Mr. Smith reports 0.81 per cent copper on the same sample. It may be of interest to know that my last determination required just 30 minutes from the beginning of the weighing to the end of the final calculation. If 1 gram is taken and put through in the same time, no zinc being used, the precipitation of the copper on aluminium is not entirely complete, hydrogen sulphide showing a plain brown coloration, tho the final result is the same.

In the matter of zinc determination, Mr. Smith quotes Rieger and Furman as stating that the ferrocyanide method fails to give concordant results with small quantities of zinc, Furman placing the minimum at 4 per cent, using 1 gram of material, which is equivalent to 8 per cent on the basis of 0.5 gram. My own experience is quite the reverse. On the basis of 0.5 gram of material I have found no difficulty in determining known quantities of zinc as low as 0.5 per cent to within 0.1 per cent. There is evidently more than one way to conduct the method.

Mr. Smith claims that his criticism of my method was based on the results of the "zinc committee" and not on those of the slag analysis. I am quite willing to accept Mr. Smith's statement as evidence of his intentions, but his article unmistakably reads quite to the contrary. His severe condemnation of my method is in the paragraph wherein only his own set of results are under consideration, and no mention of the results of the zinc committee is made until the following paragraph. As the article actually stands, its inherent inconsistency is an evidence of

Referring to my remark at the close of my last letter, Mr. Smith observes that this does not prove the infallibility of the method. Such an observation is entirely superfluous, as no such claim was made or implied. I had previously stated that the accuracy of the method was demonstrated by [illegible] of known ore contents, and the [illegible] plainly given to [illegible] the capabilities of the method in [illegible] work," of which Mr. Smith had spoken in his paragraph.

[illegible] that the [illegible] was claimed [illegible] it, but hardly [illegible] error or a mistake in transcribing from my original manuscript, where the reference is to No. 14.

Denver, Col. A. H. Low.

To the Editor of Electrochemical and Metallurgical Industry:

Sir:—In Mr. Smith's recent report, which appeared in the November and December issues of this journal, reference is made to my method for the direct estimation of alumina. It may interest some of your readers to have my comments upon his method, more especially as Mr. Albert H. Low also mentions it in his recent excellent book on "Technical Methods of Ore Analysis."

Experience with the method since its publication has borne out its claim to accuracy, at least in my hands. However, my original expectations of its becoming a technical method are not likely to be realized, partly because it is not sufficiently rapid, but mainly on account of the difficulties which some chemists experience in operating it successfully. Mr. Low, for instance, states: "My own trials with this method have not proven satisfactory." Nevertheless, if properly performed, the results are correct, and it will check closely an alumina determination by difference. It is, of course, assumed that the latter method is also properly performed, which is frequently not the case.

It is not my intention to take up valuable space by again describing this method, which appeared in the *Engineering and Mining Journal* of March 3, 1904. However, for the benefit of those not familiar with it, I may mention that the alumina is precipitated by means of sodium sulphite from a solution very faintly acid with HCl. The assumed reaction is $Al_2Cl_6 + Na_2SO_3 + H_2O = Al_2(OH)_6 + NaCl + SO_2$. It is, therefore, analogous to Woehler's hyposulphite precipitation. The liberated sulphur dioxide naturally reduces the iron to the ferrous state. The success of the method depends on a rather nice degree of acidity, and this is undoubtedly the point where some chemists fail in reaching the correct conditions. Simple as these conditions are when once acquired, it seems difficult to convey an adequate description of them. I shall herein make an effort to clear up this stage of the procedure.

We will assume that the iron and the alumina are in solution as chlorides. The solution must be perfectly cold. Nearly neutralize the free acid with sodium carbonate until a slight tinge of light brown appears, but not to a port wine color. Now add a large excess of sodium sulphite, which on stirring will produce a precipitate. The solution and precipitate leave a deep red-brown color. The addition of HCl will gradually dissolve this precipitate until a port wine color is reached, and the solution shows signs of clearing. At this point HCl should be added slowly, drop by drop, with very thorough stirring, until the color of ferric iron shows signs of fading and the solution just clears. The odor of sulphur dioxide will now be faintly perceptible, and upon heating to boiling the color of ferric iron will rapidly disappear. The alumina should commence to precipitate on or before the boiling temperature is reached, otherwise the solution is too acid. Much material contains titanium, flocks of which will remain after the alumina has gone into solution. This is likely to mislead the operator inexperienced in this method into adding too large an excess of HCl. Probably the best way of becoming familiar with the scheme is to experiment upon solutions containing only alumina until the eye becomes accustomed to the necessary conditions for complete precipitation. The filtrate can then be tested with ammonia and ammonium chloride.

Incidentally, it may be remarked that this method has certain scientific possibilities in that it seemingly gives a sharp group separation of alumina, titanium, thoria and zirconia from the balance of the earths, although its limitations in this respect have not been fully confirmed by the writer.

Great Falls, Mont. Chas. E. Rueger.

BETHLEHEM MEETING OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.

The ninetieth meeting of the American Institute of Mining Engineers was held in the Bethlehems from Feb. 21 to 24. The programme included professional meetings on Feb. 21 and 22, a reception and ball on the evening of Feb. 22, and two all-day excursions on Feb. 23 and 24. The pouring rain of the first evening washed away all the winter snow that was still some inches high on the campus of Lehigh University, and on the following three days of the convention the most delightful spring weather was enjoyed.

For the excellent arrangements and for the resulting success not enough credit can be given to the various local committees, all of which did splendid work. It is impossible to mention everyone who contributed to the success of the meeting and to the comfort of the visitors, but the names of the officers of the general local committee shall be given: Mr. John Fritz, chairman; Major A. B. De Saulles, of the New Jersey Zinc Co., vice-chairman; Mr. R. M. Bird, of the South Bethlehem Steel Co., treasurer; President Henry S. Drinker, of Lehigh University, secretary.

PHOTOGRAPH TAKEN ON THE EXCURSION OF FEB. 24, AT THE WORKS OF THE INGERSOLL-RAND CO.

Opening Meeting on Feb. 21.

The opening session was held on the evening of Feb. 21 at Lehigh University. An address of welcome by President Drinker and a speech by the president of the Institute, Mr. James Gayley, opened the proceedings. Then followed, according to the usual practice of the Institute, the presentation of biographical notices of prominent members, recently deceased; namely, a sketch of the life and work of George H. Eldridge, by Dr. S. F. Emmons, Washington, D. C., and biographical notices of Edward Cooper and of Alexander B Coxe, by Dr. R. W. Raymond.

Mr. Howard W. DuBois, of Philadelphia, presented an interesting address on British Columbia and Alaska, which was illustrated with colored lantern views of scenery, mining operations, etc. Mr. DuBois was professionally engaged in extensive explorations in the Northwest (the technical and geological sides of which were covered more fully in a paper presented by him at the meeting the next afternoon), and being a skilful photographer he took many pictures of extraordinary beauty and interest, which were colored under his direction by the foremost artists in such work. The exhibition of these pictures was greatly enjoyed by the audience, which numbered some 200 or 300.

The evening closed with a delightful informal reunion and reception on the invitation of Lehigh University.

Professional Meetings of Feb. 22.

The two meetings, held in the morning and in the afternoon of Feb. 22, were devoted to the presentation and discussion of papers. Both meetings were held in the Physical Laboratories of the University. After the morning session a luncheon was given by the University at the gymnasium, and in the interval between luncheon and afternoon session an opportunity was provided for the inspection of the buildings, laboratories, etc., of the University. Prof. J. W. Richards conducted the visitors through the Chemical Laboratory, with its specially fine electrochemical and electrometallurgical equipment, while at the Physical Laboratory, Prof. W. S. Franklin was assisted in drawing a crowd by demonstrations of the speaking arc lamp and other experiments.

Most of the ladies attended in the forenoon an oratorical contest of students, held at the University Chapel, in connection with the celebration of Washington's birthday. The

lady—who, of course, did not belong to the Institute party—came by mistake to the Institute meeting. After having listened for some 15 minutes to a discussion of dry blast, per cents of saving of fuel, etc., she turned somewhat in disgust to an interested listener nearby, and asked: "This isn't the oratorical contest? Is it?"

Dry Blast.

The first paper of the morning session was presented by Mr. C. A. Meissner, and was in form of a discussion of Mr. James Gayley's famous paper on the application of dry-air blast to the manufacture of iron. Since Mr. Meissner is the engineer of the United States Steel Co., in charge of the Gayley dry-blast experiment, his account of the results of practically three years working of the system on a commercial scale will be read with much interest, when available in print. The results may be summed up briefly in the statement that the experience of three years has confirmed all that has been claimed for the method by Mr. Gayley.

The paper elicited considerable discussion, in which Prof. Henry M. Howe, Dr. R. W. Raymond, Dr. J. W. Richards and Mr. James Gayley participated. Some remarks of Mr Gayley were of historical interest, showing his farsightedness at an early date. While others had only considered the saving of fuel necessary for the decomposition of the moisture of the blast, and thus calculated a possible saving of 3 per cent of fuel (or under special conditions a saving of 7 per cent), Mr. Gayley was from the start more concerned in the indirect savings due to the regularity of working of the furnace. When the experiment was undertaken, Mr. Gayley guaranteed a saving of 12 per cent, personally he expected 15 or 16 per cent; he obtained, as is now well known, 20 per cent.

Where does this saving come from? As Prof. Howe emphasized, a statement to the effect that the saving is due to the regularity of the working, is no real explanation. Dr. Raymond thought that metallurgical furnaces are not such ideal apparatus as to permit exact mathematical analysis, since they have fevers like human beings, and in times of "fever" some suppositions of the analysis are reversed. Prof. J. W. Richards sided with Prof. Howe, and then proceeded to give the different items to which the saving of fuel is due. His theory has already been given on page 240 of our last volume, to which the reader may be referred. Dr Richards showed that in his analysis the only item which needed further explanation was the saving of 7 per cent, due to better combustion of carbon. This, he explained, is due to a higher temperature being available in the region of the tuyeres. Carbon is saved not for reduction but for producing a higher temperature. The important point is the increased smelting power at the tuyeres. Prof. Howe quite agreed with Prof. Richards' explanation, which he stated in a somewhat modified form.

Piping in Steel Ingots.

A paper on this subject was presented by Mr. N. Lilienroth of Philadelphia. The author first discussed briefly several methods of keeping the steel liquid as far out towards the mould as possible, in order to get an even sinking of the top surface. The chief part of the paper dealt, however, with the methods of compressing the steel in the moulds. There are three methods of this kind: pressing from the top, from the bottom, from the sides. In the last method the top is left open, permitting the gases to escape, and the pressure is applied directly in proportion to the formation of the pipe, so as to keep the ingot always full to the top. It is obvious that the pressure can be better timed in this way than in the other two cases.

The author then gave a review of the work of Mr. John Ellingworth in developing a practical method of the last type mentioned. He stated that this work has now been completely successful. Mr. Ellingworth, together with Mr. S. Robinson, has built casting machines for compressing large sized ingots, cast from a ladle, at the steel works of Jessop & Sons, Sheffield, England, turning out uniformly ingots of the largest size, absolutely solid from top to bottom. There are two kinds of casting machines, one for large and the other for small ingots.

The principle is to cast the steel in molds divided in halves and held together, during the casting, by hydraulic pressure. In the planed side-edges of the molds are grooves which admit bars. After the metal has been poured into the mold, sufficient time is allowed for a crust to form while the interior is still liquid. The bars are then withdrawn by chains suspended from an overhead hoist, which leaves an empty space between the two halves of the mold. Hydraulic pressure is then applied from a ram, and the outer half of the mold is moved forward slowly so that the liquid interior is always kept full to the top until the mold is closed. The volume of the bars corresponds as near as possible to the volume of the cavities which would otherwise be found.

In the discussion which followed, Prof. H. M. Howe objected to the explanation of the formation of pipes as contraction cavities, and emphasized that on the contrary they are expansion cavities. Solid steel floats on molten steel, because the former is lighter, hence the passage from molten to solid steel is an expansion. The formation of pipes in steel ingots is the same as in ice blocks. Take a molten portion of steel inside a solid steel mass; pipes will result, according to Prof. Howe, if the expansion of the outside outruns the expansion of the inside.

The Aluminium Industry.

Prof. J. W. Richards, of Lehigh University, then spoke on the development and present status of the aluminium industry. In view of his connection with this industry he was not at liberty to present a printed paper; his lecture was therefore oral only, very concise, yet full and most interesting, and was highly appreciated. On account of the author's position, as he stated it himself, it is to be regretted that some unpublished interesting details which came out in his lecture will remain unpublished.

Dr. Richards first pointed out that the metallurgy of aluminium differs from the treatment of other metals in one important particular. With other metals reduction comes first, and the metal thereby produced is later refined. With aluminum we have not yet a method which has proven commercially successful for the purification of impure aluminium. In the aluminium industry the first step is, therefore, purification of bauxite, which is then followed by electrolytic reduction of aluminium.

Dr. Richards then gave a very appreciative sketch of the work of Charles M. Hall, and discussed both his methods of purifying bauxite and of producing metallic aluminium by electrolysis of fused baths. The reader may be referred to former articles on this subject in our journal, especially in our Vol. I., pp. 49 and 158. The commercial revolution in uses of aluminium, due to the price reduction, brought about by the application of electrochemical methods, was most happily brought out by the speaker.

Papers on Mining and Geology.

The papers presented at the afternoon session dealt with subjects of mining and geology.

Dr. R. W. Raymond first read a paper by Dr. A. R. Ledoux, on a novel method of mining kaolin from deep deposits. The method consists essentially in disintegrating the kaolin in situ by jets of water and then floating it to the surface. The paper was not discussed.

Mr. Howard W. De Bois presented a paper on the reconnaissance for the platinum metals in British Columbia. The paper was fully illustrated by lantern slides and aroused much interest. The speaker referred to the enormous increase in the price of platinum in recent years, and expressed the opinion

that new rich sources of platinum would become available in the near future in British Columbia and Alaska.

The paper was discussed in various respects. Prof. J. F. Kemp, of Columbia University, referred to the recent investigation of various black sands by the United States Geological Survey, and expressed the hope that these black sands would turn out to become a valuable commercial source of platinum.

The last paper of the meeting was presented by Mr. Thos. T. Read, the subject being the secondary enrichment of copper iron sulphides. After that Dr. Raymond, as secretary, made some announcements and gave some interesting and amusing personal reminiscences of Detective McKenna (McParland), so successful in detecting mining crimes, recently as well as some twenty years ago.

Reception on Evening of Feb. 22.

A most delightful reception was given in the evening by the citizens and industrial establishments of the Bethlehems and Lehigh University. The old historic Sun Inn in Bethlehem was entirely occupied for the occasion by the Institute and its guests. The reception took place in the large hall on the third floor, refreshments being served on the second floor. Dancing began at 10 o'clock. The hostesses of the evening were Mrs. Robert H. Sayre, Mrs. Henry S. Drinker, Mrs. E. M. McIlvain, Mrs. A. B. De Saulles, Mrs. John Fritz, Mrs G. B. Linderman, Mrs. Charles M. Dodson, Mrs. A. B. Fichter and Mrs. William B. Myers. It was a representative gathering of American beauty and brains.

Excursion of February 23

On Friday morning the members of the Institute and their guests took bright and early, a special train, courteously provided by the Jersey Central Railroad, and first went to Palmerton, near Hazard, to inspect the large plant of the New Jersey Zinc Co.

Hazard Zinc Works.

The New Jersey Zinc Co., as is well known, is the strongest factor in the zinc business of the United States, and is equal in size and importance to the celebrated Vieille Montague, of Belgium. Its plant for the production of zinc oxide and spiegel is at Newark, N. J., but at the present time only the spiegel furnace is being operated. It also has a spelter plant, zinc oxide plant, and a small spiegel furnace at South Bethlehem, Pa. In the South it has a spelter works and mines operating as the Bertha Mineral Co., at Pulaski, Va. At Mineral Point, Wis., in the Wisconsin zinc fields, it has a large oxide plant, and also an acid plant using the zinc sulphide ores of this district. At this plant sulphuric acid is produced by the Grillo-Schroeder process. It has also a spelter works at Waukegan, North Chicago, and operates in Gas City and Iola, Kan., three large spelter works. At Cañon City, Col., it operates an ore-dressing plant, working on the low-grade Western ores, using dry, wet and magnetic concentration. At De Pue, near Springfield, Ill., it is building a large spelter works and also an acid works, also using the Grillo-Schroeder process. (Concerning this contact process see our Vol. II., p. 347.) It has a smelting plant for the manufacture of high grade zinc oxide, used as a pigment, at Florence, Pa., where the oxide is made from burning metal or skimmings.

The plant at Palmerton is designed with the ultimate idea of making it the largest zinc plant in the world. The ores come from the famous Franklin furnace mine of New Jersey, where, by the Wetherill magnetic concentration process, the product from the wet concentrating mill is separated into willemite (the anhydrous silicate of zinc), and franklinite (the combined oxides of zinc, iron and manganese).

The Palmerton plant is situated in the anthracite coal district, and by an advantageous contract with nearby collieries the company is assured of a supply of anthracite coal for a long term of years. The plant consists of a spelter plant and zinc oxide plant, a large demonstrating Grillo-Schroeder [illegible] phuric acid plant and a plant for making lithophon, the white enamel pigment made by the double precipitation of barium sulphate and zinc oxide and sulphide.

Spelter Works.

The Palmerton plant is most substantially built, as the result of the combined metallurgical experience of all the plants of the New Jersey Zinc Co. The [illegible] being everywhere solid, brick, steel and reinforced concrete being everywhere used. The spelter works are designed for a future equipment of twelve furnaces of 200 retorts each. Ten of these furnaces will be of the type known as the Convers De Saulles. This furnace uses the counter-current system of regeneration, the heat being transmitted from the products of combustion through special fire-brick to the flues in which the air for burning the producer gas comes up.

The producer gas coming from producers of the Taylor revolving-grate type, made by R. D. Wood & Co., of Philadelphia, and also from a new type of producer adopted by the company—the Hughes—is not preheated, but is delivered to the furnace at as high a temperature as is possible. The Hughes producer, made by the Wellman-Seaver-Morgan Co., Cleveland, Ohio, is of very solid and substantial construction. Its chief distinction is that the whole producer revolves. This enables mechanical poking and stirring of the anthracite slack to be employed and also insures a good distribution of the coal on the top of the producer.

A special apparatus of French design (the "Ados") mechanically records every 4 minutes the percentage of carbon dioxide in the producer gas. This the company's engineers have found to be most advantageous, for the reason that the workmen know that any neglect in operation of the producers is shown up at once by the automatic recording device.

Two other furnaces will be of the Siemens regenerative type. This type, as is well known, has a single arch spanning both sides of the furnace with the regenerators underneath the furnace; the gas sweeping from the bottom of one side of the furnace up through the four rows of retorts over the middle wall and down to the four rows of retorts on the other side, where it meets the regenerative bricks and gives its heat to the same bricks. The company's engineers state that at times the waste gases are lower than 150° C. in temperature.

As would be expected the Siemens type of furnace shows a better fuel economy than the Convers-De Saulles, for the reason that the absorption of heat is by direct contact on the regeneration bricks and not by conduction through 1.5 mm of fire-clay as in the Convers De Saulles. But this advantage is more than neutralized by the fact that no reversals of gas and air are necessary in the Convers-De Saulles, and also that the temperature in the Convers-De Saulles is uniform, whereas in the Siemens there is a fluctuation of 150° C. in half an hour. This means that in the Siemens furnace there is a large consumption of retorts, due to this sudden change of 150° C. every half hour, and also a larger consumption of condensers, for the reason that the change in temperature of the furnace is communicated to the gases of zinc and carbon monoxide that are produced in the reduction of the ore, and the sudden change is manifested in a less degree in the condensers. Consequently there is a temperature change at the condensers, and checking, and cracking ensues.

The retorts are made in the Muehlen [illegible] process, out of a well pugged clay. This clay is made of seven parts of St. Louis plastic clay for a binder, and two parts of [illegible] ground fire brick for the cement or [illegible]. The press is operated at as high a pressure as possible, from 1,500 to 2,000 pounds. The clay is pugged twice.

Mr. A. L. Quereau, consulting engineer of the New Jersey Zinc Co., has patented an [illegible] smoke [illegible] composite retort with a lining of carbonaceous [illegible] bauxite, made from a composite [illegible] which is [illegible]

the stamp mill. This is patented, and the company anticipates making large-scale tests in one of their Western plants to investigate this process for the treatment of the iron-bearing ores of Leadville and the West.

The retorts or muffles are 7 inches x 9 inches inside and 48 or 60 inches long. The walls are 1¼ inches thick, and the butt is 1¾ inches thick. The average life of the retort is 28 days.

The charge consists of willemite analyzing from 48 to 50 per cent zinc, 7 to 10 per cent iron plus manganese, and 55 per cent of fine anthracite coal dust, used for reduction material. This anthracite analyzes from 79 to 81 per cent fixed carbon, and proves to be an ideal fuel for the zinc reduction purposes.

At present the retorts are charged by hand, but Mr. Queneau has invented and patented a mechanical charging device, consisting of a high-speed impeller, which makes a more uniform and better charging than can be done by hand charging. (See the department on Recent Metallurgical Patents in our present issue.) A large experimental machine has been built and operated, which machine has shown up certain advance in practical operation. A new machine is being designed taking advantage of this very valuable experience, and when once proven will be of great value to the company.

The charge is made much "heavier" than is the case in the Kansas practice. An average of 37 pounds of willemite per cubic foot of retort space is recommended by the company's metallurgists. The temperature of reduction of willemite is higher than that or roasted blende or calamine ores, and lies between 1,100° and 1,120° C. At the end of the operation the temperature inside the retort is as high as 1,450° C. The consumption of coal per ton of ore treated is quite high, from 1¾ to 2 tons per ton of ore.

The spelter produced at this plant is of extreme purity, analyzing 99.85 to 99.95 per cent zinc. This spelter is consumed for making the finest bronzes and for special zinc castings, spun brass work and cartridge cases, and also for the finest galvanizing, such as galvanized wire, and in general for all work where extreme ductility and tenacity are essential. At present only four furnaces are in operation. This spelter, sold as the famous "Horsehead brand," commands a price of from 3 cents to 4 cents above the market for "prime Western spelter."

Zinc Oxide Plant.

The zinc oxide plant is an immense affair. It has 408 Wetherill furnaces and several large bag-houses. The low-grade willemite and the magnetic heads, containing franklinite, are mixed with finely divided anthracite slack, and thrown on a bed of burning anthracite on the Wetherill grate. The gentle and uniform blast coming up through the perforations of the Wetherill grate burns the bed of anthracite below and reduces the zinc oxide in the ore to metallic zinc, which, of course, at the temperature is a vapor. This is burned under the arch of the Wetherill furnace to zinc oxide in the form of a very fine powder.

The products of combustion with the zinc oxide in suspension are led through a system of cooling pipes by the suction of electrically-operated water-cooled Sturtevant fans. Thence it goes to the bag-houses, where the zinc oxide is filtered out by the well-known system of filtration through cotton sheeting. Four long canvas bags discharge the zinc oxide into a hopper, whence it is collected in barrels and sold for zinc pigment.

There is now very little prejudice, if any, in the paint trade against zinc oxide as a paint. Indeed, it is believed that a mixture of white lead and zinc white produces the best pigment.

About 7% per cent of anthracite slack and a small percentage of limestone is charged with the zinc ore in the Wetherill grate. The ore averages about 20 per cent zinc. The residues average from 2.50 to 3 per cent zinc. A recovery of 85 per cent is considered fair work, while 90 per cent recovery is considered very good work.

The clinker from the Wetherill grates is drawn out by hand into chutes, and falls through these chutes into cars below. It is usually charged directly from these cars into the iron-blast furnace with additional limestone. This is a short blast furnace, 14 feet at the bosh, with 5-inch tuyeres, without any bosh plates, but instead cast iron segments cooled with a flow of water.

The iron and manganese in the clinker are converted into spiegeleisen. It is well up to Bessemer grade, analyzing 0.08 phosphorous (which phosphorous is principally from the coal and coke), or an iron-manganese alloy containing 20 per cent manganese; it is sold to steel makers for use in the converter or open-hearth furnace.

The fuel used in this blast furnace is large anthracite coal and Connelsville coke, in about equal proportions, though the tendency is to increase the percentage of coke. The blast of this furnace is heated in U-pipe stoves to about 500° C.; Le Chatellier pyrometers recording the temperature of this blast. It is the intention of the company to put in fire-brick stoves, but the difficulty in cleaning the secretions of zinc oxide from these stoves has prevented their use up to the present time. The company operates also two smaller spiegeleisen furnaces at South Bethlehem and Newark.

Some zinc oxide is collected in the furnaces from the burning of the gases in the U-pipe stoves. This oxide, called by the company spiegel oxide, is delivered to the spelter furnace and used to enrich the charge of zinc ore. This oxide is very easily reduced, and, in the parlance of the zinc man, "works very easy in the retort." Impure oxide from the Wetherill furnace, having been turned to oxide at the temperature of burning zinc or above 1,600° C., is on the contrary very hard to work in the retort, due to the fact that the combustion at a high temperature has produced a very strong union between the zinc and the oxygen, due to the violence of their combination. This, it may be remarked, is in line with the results found by W. McA. Johnson, published in our Vol. II., p. 185.

Perhaps one of the most striking features of the works in general is the consequence with which labor-saving devices are used throughout. Wherever possible the operation is automatic. The comparatively small number of workingmen which the visitors saw while being conducted through the very extended works, was indeed remarkable.

Electric motors are, of course, used everywhere. The direct-current two wire system is employed, with a pressure of 240 volts. The power plant contains three 600-kw. generators, built by the Crocker-Wheeler Co. The visitors were shown through the plant by Major De Saulles, Mr. Stone, Mr. Queneau and other engineers of the New Jersey Zinc Co. Not the least pleasant part of the hospitality was the liquid refreshment, of purity equal to their famous "horsehead spelter," tendered to the visitors at the head office of the company. The acid plant and the lithophon plant were quite rightly kept secret, for several practical points of working in the same are better protected by secrecy than by patents. But this journal takes great pleasure in recording its sincere appreciation of the open kindness of the New Jersey Zinc Co.'s New York office in throwing open to the Institute the best spelter and zinc oxide plant in the world. Of course, it is hardly necessary to state that this fact was appreciated by all the visitors.

Visits to Cement Plants and the Thomas Iron Works.

In the afternoon visits were paid to the works of the Atlas Portland Cement Co., the Lehigh Portland Cement Co., and the Thomas Iron Co. The special train brought the party first to the plant of the Atlas Portland Cement Co., where lunch was served in the power house, which was very nicely decorated in honor of the Institute.

The visitors were then conducted to the different departments of the Atlas Portland Cement Co. and of the Lehigh

Portland Cement Co., and it was quite interesting to compare the difference in the equipment for carrying out the various stages of making cement in the two plants. At the Lehigh Cement Works the process of quarrying was also shown.

The excursion was concluded by a visit to the Thomas Iron Works, which was particularly appropriate in view of the fact that the projector of this company, Mr. David Thomas, had been the first president of the American Institute of Mining Engineers, being elected on May 16, 1871, at the Wilkesbarre meeting, and being the presiding officer at the Bethlehem meeting in August of that year, now almost thirty-five years ago. At these works the large blowing engines, installed in 1905 by the Allis-Chalmers Co., were of special interest.

Souvenirs were given to the visitors by these three companies in form of a diary from the Atlas Cement Co., a match-box and a pen and pencil holder from the Lehigh Cement Co., and a paper weight in form of a little pig from the Thomas Iron Co.

In the evening there was an informal but most enjoyable reunion at the Northampton Club in South Bethlehem. For a happy change beer was served, and the refreshments were stated to be of the Pennsylvania Dutch type.

Excursion of Feb. 24.

Works of the Ingersoll-Rand Co. and New Jersey Zinc Co.

For the Saturday excursion a special train had been courteously provided by the Lehigh Valley Railroad. It left South Bethlehem at 9 o'clock, and took the party directly to the works of the Ingersoll-Rand Co., near Phillipsburg, N. J. The visitors were conducted by numerous guides through the works, where the manufacture of the company's well-known products were to be seen down to the smallest detail. The whole plant was thrown open to the party and the drills and compressors were to be seen in all stages of manufacture. Many were also shown in operation.

At the close of the inspection the party was taken to the company's private dining rooms, where a six-course dinner with wine and champagne was served. The whole party was in the best of humor, which found its expression in several after-dinner speeches, Major De Saulles acting as toast master.

For the photograph which is reproduced on page 87, and which was taken at the front of the Ingersoll-Rand Co.'s plant, we are indebted to that company.

The party was then taken back to South Bethlehem, where they were conducted through the plant of the New Jersey Zinc Co., which has already been described in our Vol. III., p. 380. This was the official conclusion of the ninetieth meeting of the Institute, which will long be remembered with pleasure by all who had the privilege of attending it.

The following is a complete alphabetical list of the members and guests who registered at the meeting:

E. O. C. Acker, Bethlehem, Pa.; William Afflich, Ellsworth, Pa.; Edgar A. Ashcroft, London, Eng.; W. S. Ayres and Mrs. Ayres, Hazelton, Pa.; F. E. Bachman and Mrs. Bachman, Port Henry, N. Y.; H. Foster Bain, Urbana, Ill.; Samuel Baker, Philadelphia, Pa.; Robert M. Bird, South Bethlehem, Pa.; Cyrus Borgner, Mrs. Borgner and Miss Borgner, Philadelphia, Pa.; Horace Boyd, Hokendauqua, Pa.; John Boyer, Bethlehem, Pa.; Albert Brodhead, Bethlehem, Pa.; Otto C. Burkhart, Bethlehem, Pa.; Robert E. Brooke, Birdsboro, Pa.; R. C. Carpenter, Ithaca, N. Y.; S. P. Christie and Mrs. Christie, High Bridge, N. J.; William H. Chandler, South Bethlehem, Pa.; F. H Clyner, Reading, Pa.; E. T. Clymer, Philadelphia, Pa.; Lee S. Clymer and Mrs. Clymer, Riegelsville, Pa.; W. B. Cogswell and Mrs. Cogswell, Syracuse, N. Y.; Frederick Conlin, Bethlehem, Pa.; Geo. G. Converse, South Bethlehem, Pa.; Edgar B. Cook, Pottstown, Pa.; Edgar S. Cook and Mrs. Cook, Pottstown, Pa.; C. B. Cook, Pottstown, Pa.; M. R. Cutter, Bethlehem, Pa.; Arthur B. DeSaullee, South Bethlehem, Pa.; W. A. Doble, San Francisco, Cal., Chas. M. Dodson, Bethlehem, Pa.; P. J. Donohue, Salt Lake City, Utah; B. I. Drake, New York City; Henry S. Drinker and Mrs. Drinker, South Bethlehem, Pa.; Mr. Dunn, Easton, Pa.; Howard Eckfeldt and Mrs. Eckfeldt, South Bethlehem, Pa.; Geo. R. Elder, South Bethlehem, Pa.; Lincoln Ellsworth, Ellsworth, Pa.; Wm. Eaty, Bethlehem, Pa.; Ernst F. Eurich, Montclair, N. J.; Harold G. Eynon, Bethlehem, Pa.; Thomas M. Eynon, Philadelphia, Pa.; B. F. Fackenthal, Jr., and Mrs. Fackenthal, Riegelsville, Pa.; Mrs. B. F. Fackenthal, Hereford, Pa.; A. B. Fichter, Bethlehem, Pa.; F. Firmstone, Easton, Pa.; W. S. Franklin, Bethlehem, Pa.; John Fritz, Bethlehem, Pa.; T. W. Fuller, Jr., and Mrs. Fuller, Catasauqua, Pa.; E. W. Frost, Cornwall; Chas. James Gayley, New York City; Fritz Glein, Portsmouth, Ind.; C. McM. Godley, New York City; H. H. Godshall, Lancaster, Pa.; Chas. Goodman, New York City; R. W. Gordon, Rhyolete, Nevada; Chas. B. Green, Easton, Pa.; N. V. Hansell, Lyon Mountain, N. Y.; Edward Hart, Easton, Pa.; C. W. Hayes, Washington, D. C.; J. R. Haze, Fairmount, W. Va.; J. E. Haverstick, Philadelphia, Pa.; Homer W. Hendricke, Bethlehem, Pa.; Hendry H. Henshaw, Albany, N. Y.; H. V. Hesse, Fairmount, W. Va.; L. Holbrook and Mrs. Holbrook, New York City; Garrett L. Hoppes, Bethlehem, Pa.; J. C. Holderness, South Bethlehem, Pa.; Henry M. Howe, New York City; G. A. Hoskins, New York City; Chas. E. Hulick, Hellertown, Pa.; C. S. Hurd, New York City; Edward S. Hutchinson and Mrs. Hutchinson, Newtown, Pa.; W. R. Ingalls, New York City; John D. Irving, South Bethlehem, Pa.; W. I. Johnson and Mrs. W. I. Johnson, New York City; W. McA. Johnson, New York City; Archibald Johnston, Bethlehem, Pa.; J. H. Jewett, New York City; Edward K. Judd, New York City; J. J. Kemp, New York City; Paul S. King, New York City; C. Kirchhoff, New York City; Joseph F. Klain, Bethlehem, Pa.; I. N. Knapp, Philadelphia, Pa.; S. F. Knapp, Cleveland, Ohio; Walter S. Landis, Bethlehem, Pa.; I. H. Lee and Mrs. Lee, Baltimore, Md.; N. Lilienberg, Philadelphia, Pa.; Stuart Lindeley and Mrs. Lindeley, Orange, N. J.; Garrett B. Lindermen, South Bethlehem, Pa.; C. P. Linville, State College, Pa.; I. P. Löhne, Cleveland, Ohio; E. J. McCausland, Ithaca, N. Y.; E. M. McIlvain and Mrs. McIlvain, South Bethlehem, Pa.; Wm. R. McIlvain and Mrs. McIlvain, Reading, Pa.; Henry Maddock, Bethlehem, Pa.; Chas. A. Matcham, Mrs. Matcham and Miss C. Matcham, Allentown, Pa.; Mr. Sydney H. Matcham, Allentown, Pa.; C. A. Meissner, New York City; Mansfield Merriman, South Bethlehem, Pa.; Harlan C. Miner, Gloucester, N. J.; H. S. Munroe, New York City; R. O. Norris, Wilkesbarre, Pa.; C. L. O'Neil, Allentown, Pa.; George Ormrod, Mrs. Ormrod and Miss Ormrod, Allentown, Pa.; I. P. Pardee and Mrs. Pardee, Hazelton, Pa.; E. W. Parker, Washington, D. C.; G. S. Patterson and Mrs. Patterson, Vivian, W Va.; Edmund C. Pechin, Buchanan, Va.; George Pettinos, Bethlehem, Pa.; Jos. S. Pendleton, Reading, Pa.; A. L. J. Queneau, South Bethlehem, Pa.; Rossiter W. Raymond, New York City; Thomas T. Read, New York City; Thomas Robins, Jr., New York City; M. K. Rodgers, Seattle, Wash.; E. F. Roeber, New York City; C. W. Roepper, Boston, Mass.; Jos. W. Richards and Mrs. Richards, Bethlehem, Pa.; Robert H. Richards, Boston, Mass.; John F. Saeger, Allentown, Pa.; James F. Sanborn, New York City; Robert H. Sayre, South Bethlehem, Pa.; Harry J. Seaman and Mrs. Seaman, Catasauqua, Pa.; Edward Schroeder, New York City; Abraham S. Schropp, Bethlehem, Pa.; J. M. Sherrerd and Mrs. Sherrerd, Easton, Pa.; F. F. Sharpless, New York City; P. W. Shimer, Easton, Pa.; E. M. Shipps, Newburgh, N. Y.; E. Bennett Smith, Wilkesbarre, Pa.; Oberlin Smith, Mrs Smith and Miss Smith, Bridgeton, N. J.; Baird Snyder, Jr., Lansford, Pa.; Morton Stevens, Philadelphia, Pa.; Joseph Struthers, New York City; H. H. Stock, Scranton, Pa.; L. S. Sullivan, Bethlehem, Pa.; Knox Taylor and Mrs. Taylor, Plainfield, N. J.; L. H. Taylor, Jr., Philadelphia, Pa.; John M. Thomas, Island Park, Pa.; J. B. Tyrrell, Dawson, Y. T.; R. S. Weaver, Catasauqua, Pa.; Walter Harvey Weed and Mrs. Weed, Washington, D. C.; Olaf Wenstrom, Boston, Mass.; Maunsel White, Bethlehem, Pa.; E. P. Wilbur, South Bethlehem, Pa.; W. A. Wilbur, South Bethlehem, Pa.; Eldridge Wilbur, South Bethlehem, Pa.; John F. Wilcox, Orange, N. J.; William S. Wintersteen, Bethlehem, Pa.

The Use of Chlorine Gas Under Moderate Pressures in the Chemical Arts.

By Edgar A. Ashcroft.

The tendency of the present day in electrochemical arts is all towards the cheaper and cheaper production of chlorine. This is largely due to the introduction of electrolytic caustic processes. The markets for bleaching powder have not kept pace with this development, and a modern branch of electrochemical industry, the direct electrolytic manufacture of chlorates, has also shut the door of outlet for chlorine gas long open in that direction.

Therefore, chlorine gas, once a main end product of the alkali industry, only to be reached after an expensive and complicated chain of operations, is to-day a mere by-product. It is, however, a by product which cannot, on account of its offensive properties, be thrown entirely away, and, therefore, great inducements are offered to the chemical engineer to find fresh outlets for this product. As a result many works are seeking such an outlet by manufacturing a number of chlorine

derivatives, amongst which may be named carbon-tetrachloride, stannous chloride, zinc chloride, sulphur chloride, etc.

Many of the operations necessary for the production of such products are greatly facilitated if the gas can be bubbled through a liquid or in some cases a fused solid substance, and so supplied under a moderate pressure sufficient to overcome the pressure head represented by a column of a few feet of the material under treatment.

For this purpose ordinary fans (ebonite coated or other-

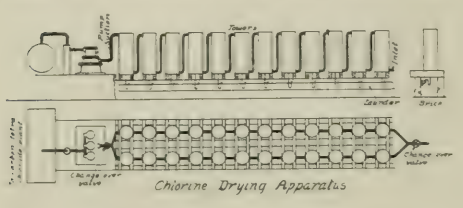


Fig. 1.—Sectional views of chlorine drying apparatus.

wise) are of no service, as by no practical device (such as multiple-series) can enough pressure be obtained.

Some years ago the writer had occasion to meet the above requirements in connection with some work for the Phœnix process for the production and electrolysis of zinc chloride from sulphide ores. Some account of the system he then worked out will possibly be found of interest to those engaged in finding outlets for chlorine gas.

This system is based upon the discovery that chlorine and hydrochloric acid gases at ordinary temperatures, *if perfectly free from moisture*, do not attack any of the ordinary metals. It is, therefore, *quite* practicable to pump the gas to *any* pressure by ordinary pumps, if certain precautions are taken.

That such a system is inherently possible is clearly evident when a few experiments are made with dried gas. The direct chlorine reactions, such as

$$Fe + Cl_2 = FeCl_2$$

like the allied oxygen reactions seldom, if ever, occur below a temperature of several hundred degrees centigrade. Even such active metals as melted zinc or metallic sodium below their critical temperatures of reaction are scarcely at all attacked by dry chlorine or oxygen (as is usually supposed), whilst good, even grained iron or steel working as valve faces to valves of ordinary engineers' bronze are absolutely unaffected by the gas for long periods.

The source of all corrosion, therefore, with chlorine gas at ordinary temperatures is found in the secondary reactions, due to the formation of HOCl(aq). This will take place either from moisture introduced with the gas, with air leaking in, or with the oil used to lubricate the pumps. Many complicated reactions also take place with the latter, which necessitates in practice special precautions. This then is the whole theory of the subject and the rest is merely a matter of mechanical design and practical experience.

The writer did a great deal of work with different kinds of fans, pressure blowers, cascade pumps, and even had a Parsons turbine air compressor constructed specially for this purpose, but all these were found defective in one or other particular, and he finally found the best solution of the problem to be in the use of ordinary metal reciprocating pumps with displacement plunger and oil seals using a special kind of oil for lubricating and for the seals which proved the escape of gas into the air.

The main principles to be borne in mind are summarized thus:

1. The gas (and all associated gas) must be carefully dried by passing and thorough drying great care must be taken that no moisture (water or leaks into the pumps or apparatus through suction glands, etc. Directions for desiccating the gas on a large scale are given below.

2. Owing to the pungent and corrosive properties of chlorine in ordinary moist air, leaks of even minute traces, such as would be negligible in pumping air, or even ammonia, are to be avoided, and, therefore, a certain specific design of pumps, with oil seals, etc., as shown in Figs. 4, 5 and 6 and described below, is requisite.

4. All ordinary kinds of oil being acted on by chlorine gas, a special oil is necessary, both for oil seals and for lubricating the pumps. The preparation of this oil, which is the result of empirical experiment, is described below.

5. It is to be observed that the rise of temperature of chlorine gas under compression, being much greater than that of air, it is necessary to use more stages to reach a given pressure, and it is generally also preferable to run the pumps slower for similar reasons. By observing these principles any desired pressure can be obtained successfully. It is thus quite practicable to manufacture liquid chlorine by this system. At present, however, we are only contemplating producing a maximum pressure of 20 pounds per square inch, which gives the pressure usually required for actual work and a margin of stored gas for flexibility. This can be successfully accomplished in a single stage. Anything beyond this can be met by any ordinary chemical engineer, who will apply the above principles to the existing practice for other gaseous compounds.

The whole apparatus employed by the writer is illustrated in

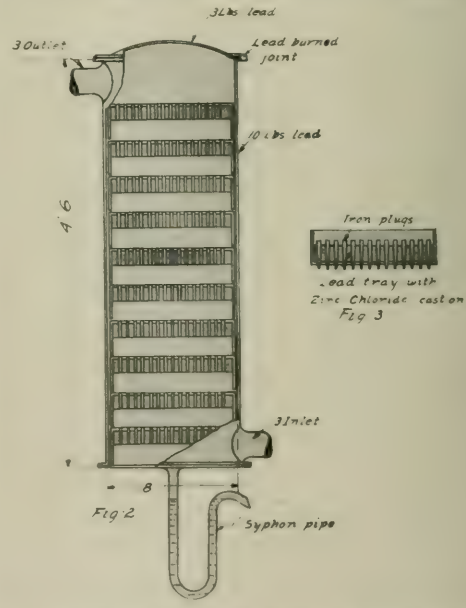


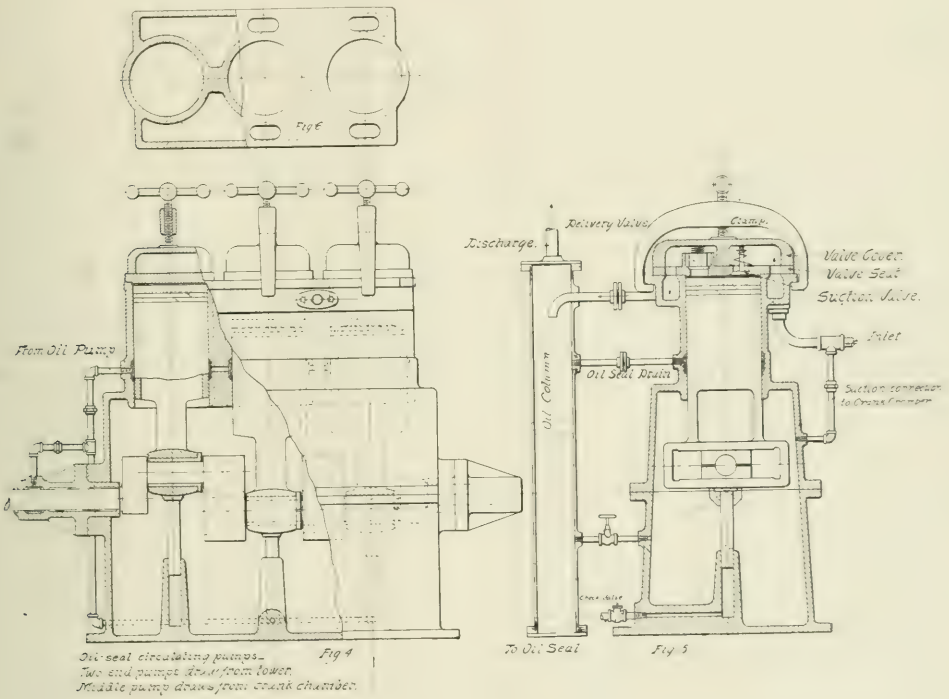
Fig. 2.—Section of lead-drying tower. Fig. 3.—Lead tray with zinc chloride cast on.

Fig. 1. There is a receiver which may conveniently consist of an old boiler or air receiver, having a good margin of safety. This is carefully tested and made tight in all joints and tarred inside and out. All useless joints should be eliminated, and all necessary joints made good and strong on a uniform plan, consisting of jointing by plain sheet lead, without any cement, between strong metal flanges. All valves should be of close grained iron or steel bodies and seats, and have gunmetal (bronze) spindles and valves, with strong threads and good (asbestos packed) glands. Always arrange the valve so that

the *permanent* pressure does not come on the gland and packing, as the latter is apt to leak and need repacking, which then can be done during temporary closure of the valve. Otherwise, the shutting down and removal of gas from the entire system becomes necessary; and as this in practice will be naturally avoided as long as possible a slight leak under such adverse conditions is apt to develop into a chronic nuisance.

This receiver is filled direct from the pumps through the standing oil column. The same remarks as above apply to all pipes, joints and valves on the pressure side. The details of this pump are further described below and illustrated in Figs. 4, 5 and 6.

On the suction side of the pumps all pipes may be most



FIGS. 4, 5, 6.—STURGESS TRIPLEX PUMP, ADAPTED FOR PUMPING DESICCATED CHLORINE BY THE ASHCROFT SYSTEM.

readily made of lead, with burned joints, and the same material has been found suitable for the drying towers.

(N. B.—It will be found advisable when very permanent work is contemplated to substitute special earthenware for lead in these towers and pipes; but the writer has used lead apparatus for several years of continuous work without difficulties arising.)

Figs. 2 and 3 illustrate a single drying tower. The desiccating medium found most useful is zinc chloride, all of which is recovered in the form of a strong solution running from the drain pipes and is periodically recast into the trays, as shown after concentration and expulsion of the moisture gathered. This is very easily and cheaply accomplished by evaporating and fusing the zinc chloride in an enamelled iron pan and casting while still fused into the trays, inverted for the purpose. During concentration the material passes through a "frothing stage," which, however, is not troublesome when working in an open pan. After this stage is passed the material settles down into quiet fusion at a temperature which need not exceed 350° C.

The packing of the towers in practice only requires to be done about once in three months, unless the gas is very wet. The lead top is removed for this purpose and the spent [illegible] out and refilled, the trays being inverted and plugs inserted in the holes whilst casting the chloride in, after which the trays are replaced and the lead top is again burned into place. These operations do not cut any big figure in costs. It has been found most convenient to use twenty-four such small towers, connected in two series of twelve each, and arranged with change-over lute valves, so that either series can be isolated at will for repacking. The connecting pipes throughout this series may be of 3-inch diameter. The "suction head," due to the friction of the towers and pipes will then be about 6 inches of water when pumping 40 cubic feet per minute in the apparatus illustrated.

Owing to the very considerable volume of mingled air always present with the gas leaving the electrolytic cell, due to the practical impossibility of completely sealing such apparatus, experimental data of actual conditions should be obtained for each plant and the apparatus proportioned accordingly. When hot gases are to be pumped this dilution of the gas by cooler air will be an advantage. In that case the first towers and pipes must be of more refractory material than lead, but the gas easily and rapidly cools to the temperature of the surrounding air in passing through the system of towers. In case of pure and very hot gases, a cooling device could be readily devised to eliminate this difficulty.

The special oil for the pumps, which plays rather an important part in this system, is prepared as follows:

Good sperm whale oil is treated with a jet of chlorine, the gas being bubbled through a cask of oil until no more chemical action (indicated by the rise in temperature of the oil) occurs. The oil will then be found to be strongly acid, and this must be neutralized by the addition of fine white zinc oxide, which is stirred well in and allowed to settle. After pouring off, the clear oil is found to be "chlorine proof," and it is the *only* oil in the writer's experience which is really satisfactory, although many lubricants only slightly affected by chlorine, such as glycerine, are partially satisfactory.

(N. B.—It has been suggested to use as a pump lubricant and in the seals, in place of this oil, strong oil of vitrol, which also has no action on ordinary metals at ordinary temperatures, and probably this would answer and might be cheaper; but so far it has not been tried. The fact that it is strongly hygroscopic is also against it, as in the case of glycerine.)

The special features of the pumps remain to be described. Figs. 4 and 5 illustrate in section a modified form of Sturgess air and oil pump, which fulfills all the necessary conditions admirably. The special features of this pump are: (1) Simplicity of design, eliminating all fancy work; (2) ease of access to valves; (3) complete enclosure of all working parts and the connection of the crank chamber to the main suction system, in order to take up any possible escape of gas; (4) a system of oil circulation and oil seals by means of which the double object of perfect lubrication and complete avoidance of leaks either outward or inward is obtained. The rest of the pump and apparatus will explain itself from a study of the drawing herewith—Figs. 4, 5 and 6.

It will thus be seen that there is nothing strikingly new or remarkable about this system of pumping chlorine; but the elaboration of small details, which have made it a practical success, will not be without interest and importance to those requiring to do similar work. Such matters of detail often take a considerable amount of experiment, involving both time and expense before they are overcome in practice.

The Zinc Industry in the Year 1905

By Franz Meyer, Ph. D.

The year 1904 was not a favorable one for the zinc smelting industry in this country. If we take the price for 60 per cent Joplin concentrates and the New York quotation for spelter as a basis, the ore price advanced about $3.00 per ton in 1904, while the price for spelter dropped 0.3 cents per pound during the same period, as compared with 1903, so that the smelters who, during the last decade paid about 50 per cent of the New York spelter price for the zinc in the ore, had, in 1904, to purchase their ore at a price of over 61 per cent of the spelter price.

These conditions have become still worse during the year 1905, as the price of Joplin ore has advanced about $10.00 per ton as compared with the previous year, while the increase in the spelter price was only 0.8 cents per pound, so that during the last year the smelters have paid almost 67 per cent of the New York spelter price for the zinc in the ore.

The situation in 1905 was, therefore, far from being satisfactory for the smelters, and an unduly large proportion of the high price for spelter which prevailed during the last year has gone to the miners of zinc ores, as will be seen from the following Table I, which was compiled from the *Mineral Industry*, and the *Engineering and Mining Journal* for the purpose of showing the relation between the price of 1 pound of zinc in the ore and in the spelter during the last two years.

Table I. shows at a glance that the past year has been the most unfavorable for the zinc smelting industry in this country during the last decade, and that the demand of the miners for a duty on zinc ores is not justified, as such a duty would have the effect of still aggravating the unprofitable conditions under which the zinc smelters have had to labor for the past two years.

The European zinc smelters have received a fair share of the high prices for spelter which also abroad governed during the last year, due to the fact that in Europe zinc ores are bought on a sliding scale, based on the London quotations for spelter. Such a system of price-making is fair to the miner as well as to the smelter, and there is no reason why it should not be adopted in the United States, more so, as in the lead and copper

TABLE I.

Year.	Average Price of 1 Ton 60% Joplin Con.	Average Price of 1 lb. Zinc in the Ore.	Average Price of 1 lb. Spelter in New York.	Difference between the two Prices.	Cost of 1 lb. Zinc in the Ore in per cents of the cost of 1 lb. Zinc in the Spelter.
1896	$22.33	1.86c.	3.94c.	2.08	47.2%
1897	22.28	1.86	4.12	2.26	45.2
1898	28.44	2.37	4.57	2.20	51.9
1899	38.54	3.21	5.75	2.54	55.8
1900	26.50	2.21	4.39	2.18	50.1
1901	24.21	2.02	4.07	2.05	49.7
1902	30.73	2.56	4.84	2.28	52.9
1903	34.44	2.87	5.40	2.53	53.1
1904	37.58	3.13	5.10	1.97	61.4
1905	47.00	3.92	5.88	1.96	66.7

smelting industry of this country the ore contracts are already based on the prices for lead, copper and silver which prevailed during the month, in which the ores were delivered to the smelters.

Ores.

Due to the conditions in the Joplin ore market, the zinc smelters have during the last year treated still larger quantities of low-grade ores than in previous years. The main supply of these ores comes from Colorado, New and Old Mexico, and British Columbia, where, before long, a zinc smelter will be in operation. There has also been an increase in the production of high-grade concentrates in Wisconsin, and there is every prospect that this State will become an important competitor to Missouri in the production of high-grade blende concentrates, as great activity is noted throughout the zinc producing district in Southwest Wisconsin, both in the prospecting for and the development of new deposits and in the construction of mills.

As the Wisconsin blende is not magnetic, the product from the wet separation, containing the zinc sulphide besides pyrites and markasite, must be partially roasted to render the iron sulphides magnetic, before it is subjected to magnetic separation. For this reason the Wetherill magnetic separator has found no application in this field, the less expensive magnetic separators designed for the treatment of highly magnetic materials being used. In Colorado, however, where the blende is weakly magnetic, the Wetherill is still the leading machine for the production of zinc concentrates without preliminary roasting, and also with British Columbian ores it has given good results.

A mill equipped with Blake-Morscher electrostatic separators was put into commission during 1905 in Salt Lake City, Utah, where it is said to do good work with ores from Park City, Utah. Of the floating processes which have been successfully introduced in Australia, none has found application in this country as far as is known.

Roasting.

Of the mechanically stirred muffle furnaces, the Hegeler kiln is still the only one which is used in actual practice in the United States. The two new zinc works now in the course of construction, namely, those of the Mineral Point Zinc Co., at De Pue, Ill., and the plant of Messrs. Hegeler Bros., at Danville, Ill., will also use this type of kiln. At the first named works the gases from the roasters will be utilized for the manufacture of sulphuric acid by the Schroeder contact process, while at the latter plant the chamber process will probably be used for this purpose. In this connection it may be mentioned that the General Chemical Co., who own the Her-

reshoff & Ferguson patents for the manufacture of sulphuric acid by catalysis, have recently acquired the exclusive United States rights for the use of the contact patents of the Badische Anilin & Soda-Fabrik, at Ludwigshafen, and of the Farbwerke, vormals Meister, Lucius & Bruening, at Hoechst, Germany, so that it may be expected that before long one of these processes, which are in operation on such an extensive scale in Europe, will also be applied to the manufacture of sulphuric acid from blende roasting gases in this country.

For improvements in mechanically-stirred muffle furnaces of the McDougal type, a number of patents have been granted during the last year, of which the following ones may be mentioned:

In A. R. Meyer's United States patents Nos. 780,115, 804,751 and 804,752, improvements in the construction of the shaft and rabble arms of the furnaces protected by United States patent No. 750,877, and means for cooling these parts of the furnace, are claimed. The designs as described in the above patents are rather complicated and expensive. In his United States patent No. 800,588, the same inventor proposes to reduce the high fuel consumption, due to the short passage to which the gases of combustion are compelled, in a muffled McDougal kiln, by a construction which very closely resembles the roasting furnace protected by F. Meyer's United States patents Nos. 731,114 and 761,691. The only difference between the two kilns lies in the fact, that in A. R. Meyer's design the walls separating the ore beds of the units arranged in a block from each other do not extend all the way up. This is no improvement on F. Meyer's construction, as for the replacement of a rabble arm and for other repairs in A. R. Meyer's furnace the whole block must be shut down, while in F. Meyer's design only a single unit is out of commission during repair work.

It is also more than doubtful whether A. R. Meyer's patent can be sustained in the courts, as it makes use of F. Meyer's prior invention to heat a muffle roasting furnace, consisting of a plurality of vertical chambers, each made up of a series of superposed hearths discharging from one to the other, and having rotary stirrers mounted upon a vertical shaft by a heating furnace common to the series.

W. R. Ingalls, in his United States patent 786,567, prolongs the course of the flames by placing a radial partially in the annular flue underneath the hearths of a McDougal kiln to be heated, whereby the fuel gas is compelled to make a circuit around the central partition protecting the shaft. He also proposes to convey the air by which the shaft and rabble arms are cooled, and which air is thereby preheated, to the lowest muffle, where the most extraneous heat must be supplied.

The same idea of transferring heat from parts where an excess may be generated to a point where all available heat is required, has been described by F. J. Falding in his United States patent 756,485.

In the United States patent 785,437, which was assigned to Frank Klepetko, who as the first one built the McDougal klins with a large diameter, now used so extensively in Montana and Utah for the roasting of cupriferous pyrites, C. H. Repath and F. E. Marcy show a furnace of this type with seven hearths, of which the two lower ones are heated directly, to effect the secondary roasting, which splits up the basic sulphates that are formed on the upper hearths. The gases from these two lower hearths are mixed with fuel gases and are, consequently, not suited for the manufacture of sulphuric acid, while the gases generated on the five upper hearths can be utilized for this purpose. To avoid the formation of large quantities of flue dust, the invention also embraces slides, over which the ore descends from one hearth to the next hearth below in a continuous stream. There is no doubt that in this kiln, the lower two hearths of which are operated like a reverberatory furnace, the fuel consumption will be lower than in kilns of the same type, of which the lower hearths are muffled. On the other hand, in furnaces of the latter description the whole of the gas developed can be utilized for the manufacture of sulphuric acid, besides the metallic losses are smaller in muffle kilns than in those of the reverberatory type.

F. J. Falding, in his United States patent 788,098, claims some improvements on the O'Brien furnace which belongs to the McDougal type. This furnace has been muffled, and will soon be tried by the United Coke & Gas Construction Co., who build it. If properly designed the kiln will, no doubt, roast blende down to a low percentage of sulphur, but the fuel consumption will be high, on account of the short passage of the heating gases.

A roasting furnace designed on a principle quite different from that underlying the above constructions is described by A. W. Chase in his United States patent 804,379. This kiln consists of a series of superposed screw conveyors, the trough of which is made from refractory clay, while the screw is built up from cast-iron wings and a drawn steel tube, which can be cooled by water. This kiln was tried as a reverberatory for pyrites and as a muffled furnace on blende, but in both cases the experiments have not led to its introduction.

Smelting.

No inventions of a revolutionizing character have been made during 1905 in the reduction of zinc ores. Of the improvements of minor importance the introduction of the Mehler hydraulic retort press in two works in the Kansas gas field and at one smelter in Virginia may be mentioned. The use of hydraulic-made retorts, which stand the corrosive action of ferruginous ores much better than the old-fashioned retorts, on account of their greater density, has enabled the zinc smelters to utilize large quantities of low-grade ores. While a few years ago blende with more than 2 per cent iron was regarded as almost unfit for the manufacture of spelter by the United States smelters, they have now, like their European brethren, learned to treat ores with as high as 20 per cent iron.

The experiments with the Lungwitz process at Warren, N. H., which I mentioned in my last year's report, have not led to a result, because of certain mechanical shortcomings in the construction of the furnace, which were only detected during the operation. These mechanical difficulties are, however, not unsurmountable, and the experiments, it is said, will be continued this year.

An editorial in this journal (Nov., 1905) and A. Lodin, in his recently published book, *La Metallurgie du Zinc*, draw attention to the fact that according to the law of the partial gas pressure the furnace must be placed under a considerably higher pressure than the inventor contemplates to use, in order to increase the pressure on the comparatively small volume of zinc vapor present in the mixture of the gases in the furnace to raise its boiling point sufficiently.

The experiments which were conducted in Hamborn-on-the-Rhine with O. Nagel's process, described in his United States patent 766,279, have been abandoned, according to information from the inventor, before conclusive results as to the merits of the process were reached.

P. Schurieder's shaft furnace, protected by United States patent 757,059, and also referred to in my last reports, is now being tried in Upper Silesia, Germany; the results have, however, not yet been made known.

F. Kellermann, in *Mining Magazine*, Vol. XII., 4, p. 310, recommends to arrange a series of his furnaces, which may be considered as vertical retorts, around one gas producer as heating source common to the whole series. According to his opinion, shaft furnaces heated internally promise very little success, as the dilution of the zinc vapors with [illegible] oxidizing gases is too large to allow their condensation. Kellermann figures that in a [illegible] the proportion of the volumes of zinc vapor and [illegible] is about 1 to 3, while in an internally heated shaft furnace this relation is 1 to 10.

Armstrong, in his British patent 20,543, 1904, shows an improved form of the [illegible]

132,139, and mentioned in my last year's report. As far as is known none of these constructions has been tried in practice.

Of the patents granted on improvements in the old-type retort furnace, the following may be mentioned:

H. and J. W. Hegeler, in their United States patent 792,773, in which a regenerative furnace with chambers for preheating both air and gas is shown, propose to introduce the entire amount of the air at the bottom of the retort chamber under the lower tier of retorts, and to have an excess of air up to the top of the combustion chamber and to introduce the producer gas at various heights of the chamber through channels in the center wall. If the size of the gas openings can be regulated with sufficient exactness, the arrangement will warrant great uniformity in heating the retorts, but the regulation of the numerous gas exits will be a very difficult, if not impossible, task.

E. C. Hegeler, in his United States patent 794,799, describes the long furnace as used by the Mathiessen & Hegeler Zinc Co., in La Salle, Ill. The waste heat from this furnace, which is very simple and inexpensive as compared with the Siemens regenerative furnace, is not utilized for preheating the gas or air, but it is made use of for generating steam in a set of boilers placed near the one short end of the furnace, the same as is done so successfully now with the waste heat from the long reverberatory copper smelting furnaces.

Although P. A. Mackay's United States patent 799,743 does not strictly refer to the smelting of zinc ores, it may be mentioned in this connection, as it has for its object the removal of cadmium from calcines containing same preparatory to their smelting. For this purpose the inventor leaches the calcines, which contain the cadmium in form of water-soluble $CdSO_4$, while all other oxides and sulphates formed in roasting are practically insoluble in water. This patent is of no great practical value, as a trace of cadmium in the spelter is in most cases not objectionable. Even the spelter used for brass making may contain up to 0.5 per cent Cd. Where the spelter is used for the manufacture of zinc oxide it should, of course, be free from cadmium, as its oxide is brown, and these discolor the zinc oxide. In this country, however, very little zinc oxide is made from spelter.

Of the electrolytic processes for the recovery of zinc from its solution, A. G. Betts' United States patent 701,401 may be briefly described. Betts proposes to leach ores with sulphuric acid, to electrolyze the solution with an insoluble anode and with a liquor metal cathode, for which he prefers to use mercury either pure or with zinc. The zinc-mercury-alloy formed, and which contains less than 5 per cent zinc, is removed to another cell, containing a solution of zinc chloride, into which a zinc cathode is introduced, the zinc-mercury-alloy forming the anode. The zinc dissolved from the alloy is deposited on the cathode. After the removal of a large part of the zinc the mercury is used over in the first cell. The process is rather complicated, and as losses of mercury, which is so much more expensive than zinc, cannot be entirely avoided, it is very doubtful whether zinc can be made economically by this method.

The same must be said about the process which O. W. Brown and W. F. Oesterle describe in ELECTROCHEMICAL AND METALLURGICAL INDUSTRY, Vol. III, p. 378. These authors propose to smelt a mixture of zinc sulphide, calcium oxide and coal in an electric resistance furnace. The results obtained with certain proportions of this mixture and described in the above publication are not very encouraging.

Conclusion.

Although the great number of patents taken out during the last year indicate that many efforts have been made during the last year to improve the present expensive method of recovering zinc from its ores, not much has been accomplished in this direction during 1903. It is, therefore, still true, as I stated in winding up my last year's report, that the zinc industry offers great opportunities to an inventive mind.

Two Tests of Electrolytic Hypochlorite Plants in England.

By W. Pollard Digby.

Engineering interests being liable to run in cycles—superheated steam, for instance, is not the discovery of the last decade or so—and interest having recently been aroused in regard to the properties of electrolytically-produced hypochlorites of soda, magnesium, etc., by means of some experiments at Guildford last year, together with some trials in the North of England in regard to the bleaching of paper pulps, and the fact that a London Borough Council is contemplating the erection of a municipal electrolizer, the writer has thought it opportune to describe a record of a test taken some six years ago, together with a record of a more recent test of a similar plant.

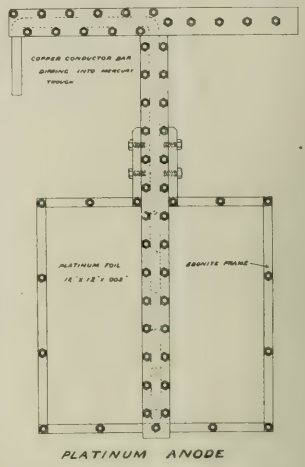


FIG. 1.—ELECTRODE CONSTRUCTION.

The first test was taken at Maidenhead, where, at the local sewage outfall, the effluents were being sterilized by the "electrozone process." As at first used in this country Woolf's form

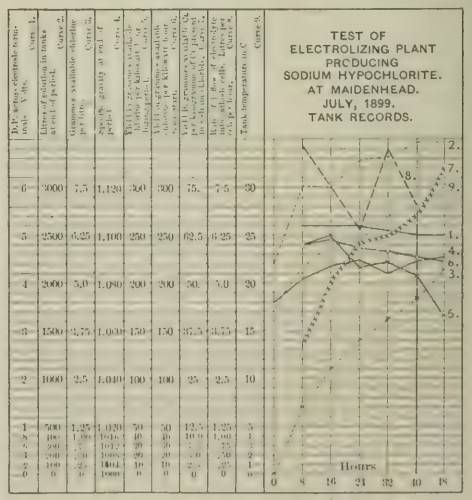


FIG. 2.—TEST OF HYPOCHLORITE PLANT.

of electrolizer was employed: the anodes consisting of platinum foil and the cathodes of carbon or zinc, but subsequently the Crawford modification was employed (British patent No. 14,852, of 1896). The construction of the anodes was slightly altered from the Woolf pattern; it is shown in Fig 1.

The essential features of the Crawford patent, as applied to

the manufacture of hypochlorite solutions, lay, firstly, in the use of air agitation, which maintained a circulation of the electrolyte, and also served to keep down the temperature, and, secondly, in the porous partition which surrounded the cathodes.

The function of this latter device was to enable the cathode hydrogen, which was given off in large quantities, to pass freely into the air without, as had previously happened, carrying with it a large quantity of chlorine gas, an action similar to that by which sulphurous acid fumes, which have such a corrosive action in badly supervised battery rooms, are carried into the air.

The cathode partition, or cell, must be extremely porous in order that the caustic soda liberated at the cathode may percolate back into the solution and reuniting there with the chlorine liberated at the anode, form an oxide of chlorine, usually regarded as hypochlorite of soda.

The particular test of which the writer gives particulars, showed, among other matters, the importance of the porosity of this cathode compartment. It had been the custom, in order to limit clogging of the partition, to lead the incoming unelectrolized saline liquor into the cathode compartments, whence, carrying with it a part of the caustic soda, a certain amount of fresh liquor was available for electrolysis, and caustic soda available for union with the chlorine to form the hypochlorite. The test in question was continued over 46

Table I.

TEST OF ELECTROLIZING PLANT PRODUCING SODIUM HYPOCHLORITE AT MAIDENHEAD, JULY, 1899.

TANK RECORDS

Hours since start	D. P. Across Electrode terminals. Volts.	Litres of solution in tanks at end of period.	Grammes available chlorine per litre.	Specific gravity at end of period.	Yield in grammes of available chlorine per kilowatt hour during period.	Yield in grammes of available chlorine per kilowatt hour single start.	Yield in grammes of available chlorine per kilogramme of chlorine present in sodium chloride	Rate of inflow of electrolyte into cathode cells, litres per cell per hour.	Tank temperature in C°.
0		2190	NIL	1.070		—			17.8
8	5.2	3070	1.78	1.087	240.4	240.4	34.1	8.56	20.6
16	5.2	2996	2.78	1.095	250.3	246.2	48.8	7.55	21.7
24	5.2	3276	3.58	1.098	217.7	236.8	60.9	6.24	22.5
32	5.1	3313	4.11	1.109	221.5	233.2	62.8	8.49	21.1
40	5.0	3482	4.60	1.114	208.2	228.1	67.3	6.92	22.2
48	5.0	3482	5.31	1.120	165.8	220.7	79.0	NIL	22.8

hours, during the last six of which no further liquid was fed into the cathode cells, with the result that the yield fell to a very marked degree.

In certain respects the first test is not a model one, but the exigencies of the situation are answerable for several of these, as large quantities of liquid had to be taken from the tank for immediate use for sewage sterilization. The initial voltage was rather in excess of that normally used, being 5.4 volts, while the specific gravity was at first somewhat low. Finally the voltage fell to 5.0 volts, and the specific gravity rose from 1.070 to 1.120.

Tests were taken at the end of each 2 hours, but for the purpose of conciseness only the values at the end of each 8 hours are reproduced in the accompanying diagram. The current was kept throughout at 500 amps. The available anode surface, counting each side of each anode, was 1,680 square inches, the cathode area being about 30 per cent greater, seven T-type platinum foil anodes being employed.

Table I. gives the chief results also as diagrammatically plotted in fig. 2.

As has been stated, considerable quantities of the liquid were withdrawn from the main tank for the purpose of treating the sewage effluent, these withdrawals taking place as follows:

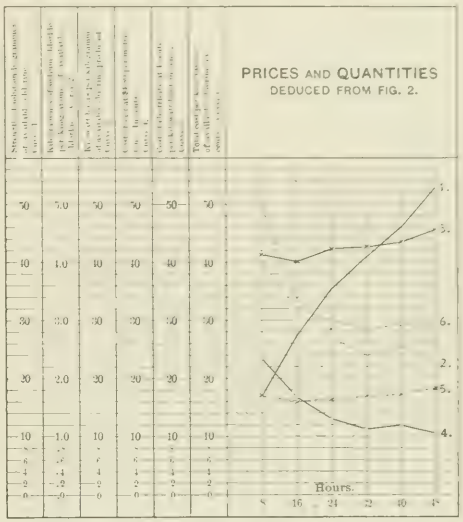


FIG. 3.—PRICES AND QUANTITIES DEDUCED FROM FIG. 2

Table II.

PRICES AND QUANTITIES DEDUCED FROM TABLE I.

Strength of solution in grammes of available chlorine	Kilogrammes of sodium chloride per kilogramme of available chlorine produced	Kilowatt hours per kilogramme of available chlorine produced	Specific gravity of solution.	Cost of salt at $4.80 per metric ton. Cents.	Cost of electricity at 4 cents per kilowatt hour. Cents.	Total cost per kilogramme of available chlorine. Cents
1.64	48.87	4.16	1.087	23.45	16.64	40.10
2.78	34.17	4.06	1.095	16.40	16.16	32.56
3.58	27.37	4.22	1.098	13.14	16.88	30.02
4.11	24.09	4.27	1.109	11.66	17.08	28.64
4.60	24.80	4.38	1.114	11.90	17.54	29.44
5.31	22.60	4.58	1.120	10.84	18.32	29.16

At the end of 8th hour, 760.5 liters containing 1.78 grammes average Cl per liter.

At the end of 18th hour, 688.5 liters containing 3.08 grammes average Cl per liter.

At the end of 26th hour, 580.5 liters containing 3.58 grammes average Cl per liter.

At the end of 34th hour, 418.5 liters containing 4.1 grammes average Cl per liter.

The test was not a model one, but, writing with the records before me of the working of a certain installation for a number of two years, a model test is hard to find. Being of an unstable nature, it is a matter of extreme difficulty when producing hypochlorite solution to keep all the factors constant and vary only one of these. The causes of variation in efficiency of production during the periods under review are included, in addition to variations in specific gravity, temper-

ture variations within the tank, the difference between the temperature of the tank and of the surrounding air, variations in in-flow through the cathode cells, condition of these cells militating against stability, and, therefore, against efficiency of production, lay in the temperatures attained. Tank temperatures to ensure stability should not exceed 18° C., and the more closely the tank and room temperatures approximate, the more stable the solution. During part of this run the engine room temperature was 29° C. However, within the actual range of the test the efficiency was, for a process of this character, very satisfactory, the current efficiency throughout being over 85 per cent. The figure given of the cost is a purely hypothetical one, based on a cost of electrical energy of 4 cents per kw-hour delivered at the tank terminals. To this would have to be added the cost of supervision, which would decrease with an increased plant capacity measured in kilogrammes of available chlorine per kw-hour per hour.

The final quantity of available chlorine present per liter was

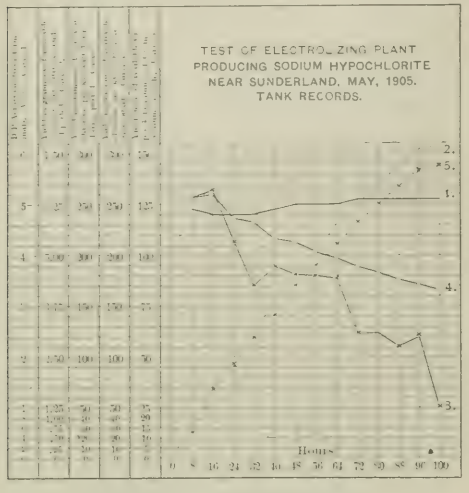


FIG. 4.—TEST OF HYPOCHLORITE PLANT.

TABLE III.

TEST OF ELECTROLIZING PLANT PRODUCING SODIUM HYPOCHLORITE NEAR SUNDERLAND, MAY, 1905. TANK RECORD

Hours since start.	D. P. Across electrode terminals. Volts.	Yield in grammes of available chlorine per litre at end of period.	Yield in grammes of available chlorine per kilowatt hour during period.	Yield in grammes of available chlorine per kilowatt hour since start.	Yield in grammes of available chlorine per kilogramme of chlorine present in sodium chloride.
0	—	0		—	—
8	4 9	0 88	258 4	258 4	12 7
16	4 8	1 80	262 6	260 5	35 0
24	4 8	2 53	214 5	238 7	47 4
32	4 8	3 15	172 9	232 6	59 0
40	4 9	3 84	188 5	215 8	71 9
48	5 0	4 52	180 8	212 7	84 6
56	5 0	5 10	180 8	202 0	95 5
64	5 0	5 72	178 5	197 2	107 1
72	5 1	6 28	172 8	189 9	116 6
80	5 1	6 70	125 4	182 9	125 5
88	5 1	7 12	110 2	176 0	133 3
96	5 1	7 58	120 4	171 2	141 9
100	5 1	7 65	47 2	166 0	144 2

[illegible] 1 080
[illegible] 2113 cubic
[illegible] 13 00
[illegible] 16 50

[illegible] is partially [illegible] impurities in the [illegible] to, and I am [illegible] [illegible] but not entirely [illegible] in the [illegible] [illegible] [illegible] test [illegible] [illegible] important item

TABLE IV.

PRICES AND QUANTITIFS DEDUCED FROM TABLE III.

Strength of solution in grammes avail able chlorine per litre.	Kilogrammes of sodium chloride per kilogramme of available chlorine produced.	Kilowatt hours per kilogramme of available chlorine produced.	Cost of salt at $4.80 per metric ton. Cents.	Cost of electricity at 4 cents per kilo watt hour. Cents.	Total cost per kilogramme of available chlorine.
1.86	47.9	3.84	23.00	15.36	38.36
3 15	28.2	4.34	11.22	17.36	28.58
3.84	23.2	4.64	10.14	18.56	28.70
4.52	17.5	4.70	8.40	18.80	27.20
5.72	15.6	5.07	7.49	20.28	27.77
6.23	14.3	5 27	6.86	21.08	27.94
7.12	12.2	5.68	5.86	22.72	28.58
7.65	11 7	6.02	5.62	24.08	29.70

5.31 grammes. The ratio between the available chlorine produced and the quantity of chlorine present in the form of sodium chloride at the end of the test was only 7.9 per cent. Reckoning the quantity of sodium chloride used per kilogramme of available chlorine produced, the final figure of 22.60 kilos. was secured, a figure too high to permit of the commercial application of electrolytically produced hypochlorite solution in a number of directions.

The second test was carried out in the neighborhood of Sunderland early in 1905. As will be seen, the tank capacity was considerably less, a smaller current being employed, but the net results were financially more favorable. The available anode surface was considerably less, only four T-type anodes being employed (three of which had been in continuous use since the preceding trial), having an aggregate net anode area of 900 square inches, the cathode area being 25 per cent greater.

In this case no circulation of an incoming electrolyte through the cathode cells could be secured, and trouble was therefore experienced through the clogging of the cells, a fact which reacted rather severely upon the electrochemical efficiency. It will be observed that the economical range of production is greater, thanks to a saving in the amount of salt used, the final quantity of available chlorine present per liter was 7.65 grammes. The ratio between the available chlorine produced and the quantity of chlorine present in the form of sodium chloride at the end of the test was 10.6 per cent. Reckoning the quantity of sodium chloride used per kilogramme of available chlorine produced, the final figure of 11.7 kilogrammes was obtained. The critical point, i. e., the point of minimum cost of production, is between 4 and 5 grammes per

liter, a point which agrees with that of the earlier test, save that the combined costs are about 8 per cent less.

In regard to this latter test, the writer found out during the investigation from which the test is quoted, that the figures which are given of the yield during the period represent the net amount of available chlorine produced within the electrolyzing tank. The gross amount actually produced is in excess of the net production, by reason of the depreciation which has

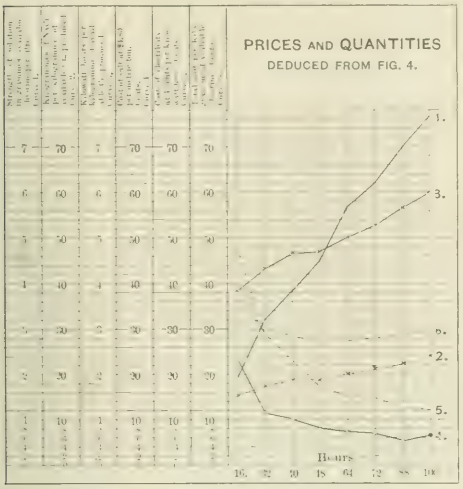


FIG. 5.—PRICES AND QUANTITIES DEDUCED FROM FIG. 4.

taken place. In a paper recently read before the Faraday Society, the writer has dealt with the effect of this during certain other tests.

Briefly, the whole question of the effect of depreciation upon efficiency at any given strength resolves itself into a matter of the ratio of tank contents to yield per hour. A depreciation at the rate of 1 per cent per hour is practically negligible when the capacity of the electrolyzing tank is equal to, say, 5 or 6 hours' output. When, however, the tank capacity represents several days' output, efficiencies, such as those indicated towards the close of the second test quoted, must, so long as hypochlorite solutions are even slightly unstable, be repeated. On other occasions, using a less volume of electrolyte, about 1,000 liters solution of 9 grammes of available chlorine were produced, with net efficiencies superior to those of the last 28 hours of the second test herein recorded. Details of these are not quoted, as they were not "start to stop" runs, but sequel runs with lessened volumes to other previous tests.

Reception to Prof. Ostwald.

A reception was tendered, on Feb. 2 by the New York Section of the American Electrochemical Society, to Prof. Wilhelm Ostwald, on the occasion of his return home to Leipzig. The hall of the Chemists' Club was beautifully decorated with evergreens and American and German flags, and there were present some 300 admirers of Prof Ostwald. The speakers were Dr. C. A. Doremus, Prof. J. W. Richards, Prof. C. F. Chandler and Prof. Morris Loeb. Prof. Loeb, who spoke for the many old and recent students of Ostwald, presented a souvenir album, containing portraits of a few distinguished American chemists, with the autographs of those who were present at the reception. Prof. Ostwald's reply was a plea for esperanto—a universal language. The reception was an extremely enjoyable affair. Prof. Ostwald sailed for Germany on Feb. 6.

Metallurgical Calculations.

By J. W. Richards, Ph. D.

CONDUCTION AND RADIATION OF HEAT.

These two factors are of the greatest practical importance to the metallurgist, yet they are also the subjects of all others upon which the practical metallurgist is usually the most poorly informed. It often happens that a furnace is built of twice the capacity of its predecessor or of its neighbors, and the manager unexpectedly and most agreeably discovers that it keeps up a more uniform heat and requires considerably less than twice the amount of fuel to keep it running. Two reasons were operative: first, the walls were made thicker to support the heavier structure, but that made them also poorer conductors of heat; second, the capacity increased as the cube of a linear dimension, while the radiating surface increased as its square, so that radiation was, therefore, less than twice the primary amount from the furnace of double capacity. Many practical metallurgists have learned the practical results, but are entirely ignorant of the principles upon which the results are obtained. While it is well to be successful, it is better to be intelligently so.

Principles of Heat Conduction.

It is well known that the ability of metals to conduct heat is very nearly the same as their ability to conduct electricity. The order of metals in these two series is almost identical. Further, the specific conductance for heat is closely analogous to specific conductance for electricity. Just as we say that the electrical resistance of a conductor is proportional to its length and inversely as its cross-section, so we can make the same statement concerning heat resistance. Just as we can add electrical *resistances* when the bodies are in series, and must add electrical *conductances* when they are in parallel, so we can add heat resistances when the bodies are in line, and must add heat conductances when they are hooked up together in parallel.

The unit of electrical resistance is an ohm, and a substance has unit resistivity when a cube of it, 1 centimeter on a side, has that resistance. In such a case a drop of potential of 1 volt from one side to the opposite one sends through the cube 1 coulomb of electricity per second. Since conductivity is merely the reciprocal quality, actually and mathematically, to resistivity, the unit of conductivity is defined in exactly the same way, and the conductivity of any substance may be expressed as so many reciprocal ohms.

If we read in the preceding paragraph, thermal resistance for electrical resistance, degrees temperature for volts, and gram-calories for coulombs, we have defined the corresponding unit of thermal resistivity. A substance has unit capacity for conducting heat when a cube 1 centimeter on a side transmits 1 gram-calorie of heat per second, with a drop of temperature from one surface to the other of 1° C. Many investigators have adopted slightly different units, such as 1 kilogram-calorie per cubic meter per hour per degree difference; but such are only simple multiples or fractions of the centimeter-gram-second unit above defined.

Illustration.—If the thermal conductivity of copper is 0.92 units, what would be the amount of heat passing per hour through a sheet 1 millimeter thick and 1 meter square, with a constant difference of 1° C. between the two sides?

Solution.—The 0.92 units means that a column of copper 1 centimeter long and 1 square centimeter in cross section, having a difference of temperature at its two surfaces of 1° C., would allow 0.92 gram-calories to pass through it per second. For a sheet one-tenth as thick, [illegible] and during 1 hour, the heat passing over 1° C. difference [illegible]

$$0.92 \times \frac{10}{1} \times \frac{10{,}000}{1} \times [\text{illegible}] = 331{,}200 \text{ Cal.}$$

Such calculations as the above are, of course, very useful for comparing different thicknesses of plates of different material, as to their relative heat-carrying capacity. With some slight modifications they are applicable to the actual conveyance of heat through such walls, from a fluid on one side to a fluid on the other. This introduces an idea analogous to transfer or contact resistance in electrolytic conduction. Thus, if we call R the thermal specific resistance (resistivity) in C. G. S. units, of the material of a partition having a thickness of d centimeters, and an area of S square centimeters, then the thermal resistance of the body of the partition will be

$$\frac{R \times d}{S}$$

and its thermal conductivity

$$\frac{S}{R \times d} = k$$

Besides this resistance in the body of the material there is transfer resistance at its two sides, or at its inner and outer surfaces, which depends on the materials which communicate heat and receive heat, and their velocity. These resistances may be expressed as so many units per square centimeter, the unit being of exactly the same nature as R, except that it connotes no thickness, but merely a surface effect. Calling R^1 the specific resistance of transfer from the inner fluid to the inner surface of the material, and R^2 the outside specific transfer resistance to the fluid outside, we can consider all three resistances as being in series, and the total thermal resistance to transfer from the inside fluid to the outside fluid to be

$$\frac{R \times d}{S} + \frac{R^1}{S} + \frac{R^2}{S}$$

or the thermal conductance of the system

$$\frac{S}{(R \times d) + R^1 + R^2} = k$$

The following table gives the thermal resistivity of various materials, in C. G. S. gram-Calorie units, also the thermal conductivity in units which are reciprocals of the resistance units:

Material.	*Conductivity. (in reciprocal resistance units.)*	*Resistivity. (in C. G. S. gram-Cal. units.)*
Silver (at 0°)	1.10	0.91
Copper (0° – 30°)	0.92	1.09
Copper, commercial	0.82	1.22
Copper, phosphorized	0.72	1.39
Magnesium	0.38	2.63
Aluminium (0°)	0.34	2.94
Aluminium (100°)	0.36	2.75
Zinc (15°)	0.30	3.33
Brass, yellow (0°)	0.20	5.00
Brass, yellow (100°)	0.25	4.00
Brass, red (0°)	0.25	4.00
Brass, red (100°)	0.28	3.57
Cadmium (0°)	0.20	5.00
Tin (15°)	0.15	6.67
Iron, wrought (0°)	0.21	4.76
Iron, wrought (100°)	0.16	6.25
Iron, wrought (1,200°)	0.14	7.14
Iron, cast (0°)	0.11	9.01
Iron, cast, hard	0.06	16.67
German silver (0°)	0.07	14.28
German silver (100°)	0.09	11.11
Lead (0°)	0.084	11.90
Lead (100°)	0.076	13.16
Antimony (0°)	0.044	22.73
Mercury (0°)	0.015	66.67
Mercury (50°)	0.019	52.63
Mercury (100°)	0.024	41.67
Bismuth (0°)	0.018	55.55
Wood's alloy (7°)	0.032	31.25
Alloy, 1 Sn. 99 Bi	0.008	125.00

The last alloy, bismuth, with only 1 per cent of tin, is remarkable for its extremely low heat conductivity, less than one hundredth as good as copper. It has also the lowest electrical conductivity of any alloy, about one two hundredth that of copper. These peculiar properties ought to make it of use for some particular purposes.

The above data enables one to calculate the rate at which heat will pass through a metallic partition or wall when the temperature of its two surfaces is known. This is a very useful calculation, for in many cases the temperature inside a partition or pipe is known, or can be easily determined, and when the temperature of the outside surface is determined, the calculation can be made. The temperature of the outside of a partition or pipe can be found by several methods: one is to lay a very flat bulb thermometer (made especially for this purpose) against it; another is to put the junction of a thermocouple against it, covering the couple with a little putty or clay; another is to take small pieces of metals, or alloys of known melting points, and see which melts against the hot metal. The only uncertainty then in the calculation is the question as to what difference in the conductivity may be caused by the higher temperature. The conductivities in most cases decrease as temperature rises, but in others increase. There is here a large field for metallurgical experiment, in determining the heat conductivities of metals, alloys and fire-resisting materials at high temperatures.

Illustration:—An iron pipe, 6 centimeters in diameter outside and 5 centimeters inside, is filled with water at 10° C., and surrounded by hot gases at 198° C. From the rise in temperature of the water it is known that for each square centimeter of heating surface 0.084-gram calories of heat pass per second. Assuming the thermal conductivity of the iron (k) = 0.14, what is the difference of temperature of the two surfaces, inside and outside, of the pipe?

Solution:—The thickness of the walls is (6 – 5) ÷ 2 = 0.5 centimeter. If the walls were 1 centimeter thick, 1° difference would transmit 0.14 calorie; but being only half that thick, 1° difference would transmit 0.21-gram calorie. The actual difference of temperature of the two surfaces must then be

$$\frac{0.084}{0.28} = 0.3° \text{ C.}$$

Of course, such calculations can be turned around, and if the temperature of the inside and outside surfaces is known, the heat being transmitted can be calculated; or if the temperature of these two surfaces is known and the heat being transmitted is measured, the thermal conductivity of the partition can be reckoned; or, again, if the temperature of the two surfaces is known, and also the thermal conductivity of the partition, and the temperature of either fluid on either side, the thermal resistance of the transfer from either of the fluids to the surface of the partition can be calculated.

Illustration.—In the previous illustration the temperature of the hot gases was 198°, while that of the pipe in contact with them was practically 10°. What was the thermal resistivity of the transfer from gas to metal?

Solution.—The thermal resistivity is the reciprocal of the thermal conductivity, and in the case of this transfer resistance is the reciprocal of the number of gram calories which will be transferred to 1 square centimeter of pipe surface from the hot gases per second per each degree centigrade of difference of temperature causing the flow. This is a surface or skin resistance, and, therefore no linear dimension representing

thickness enters into the calculation. There is 0.084 calorie per second being transferred to each square centimeter of pipe, with a difference of temperature acting as propelling force of 198 — 10 = 188°. The thermal conductivity of this transfer k is, therefore,

$$0.084 \div 188 = 0.00045$$

And the thermal resistivity, R, the reciprocal, viz.: 2222. Since the thermal resistivity of the iron wall is $1 \div 0.28 = 3.57$ units, it follows that the contact surface offers $2222 \div 3.57 = 622$ times as much resistance to the flow of heat as the metal itself. In this specific instance, we can conclude that the transfer of heat from the gases to the pipe is the principal item which conditions the flow of heat, the passage of the heat through the wall of the pipe itself taking place 622 times as readily.

The valuable practical conclusion is that the thermal resistance of the walls of the pipe is insignificant as compared with the whole thermal resistance, and the thickening or thinning of the walls of the pipe, or the substitution of copper for iron, because of its greater thermal conductivity, is practically unnecessary for thermal considerations, since such can be expected to make practically no change in the thermal resistance of the whole system.

If the pipe under discussion, however, acquires during use a layer of scale deposited from the water, then the thermal resistance of the pipe or of the system is materially affected. A study of the thermal resistivity of various boiler scales would be a very useful and practical subject, but has not been done, as far as the writer is aware. Assuming a deposit, 0.5 centimeter thick, of material having the thermal conductivity of plaster of paris, for which k = 0.0013, the specific resistance of this material is $0.14 \div 0.0013 = 108$ times that of iron, and therefore 0.5 centimeter of this represents 54 centimeters thickness of iron. The thermal resistances in this case, neglecting that of transfer from the water to the pipe or scale, are as follows:

Resistance of transfer, gases to pipe $= 2222.0 = 85$ per cent

Resistance of 0.5 c.m. iron $= \frac{0.5}{0.14} = 3.6 = 0$ "

Resistance of 0.5 c.m. scale $= \frac{0.5}{0.0013} = 384.4 = 15$ "

Total $= 2610.0$

It thus appears that a deposit of scale from the water to a thickness of 5 millimeters increases the total thermal resistance of the system some 18 per cent of its original amount, and would cut down the efficiency of this part of the heating surface of a boiler by this amount. These considerations are not only of vital importance to the steam boiler engineer, but they are the essential principles which condition the efficiency of steam-heating apparatus, feed-water heaters, hot-air stoves and ovens, air-cooling of parts of furnaces and the efficiency of water jackets.

Principles of Heat Transfer.

We have already had to speak of the transfer of heat from fluids to solids, or *vice versa*, and in one specific case we deduced the value 2222 for the transfer resistivity from hot gases to the surface of iron pipe, meaning thereby that for each degree of temperature difference between the gases and outside of the pipe 0.00045 gram-calorie passed per second through each square centimeter of contact surface. A consideration of the transfer of heat through such contact surfaces, from gases or liquids to solids and *vice versa*, has shown that the transfer resistivity varies with the solid and with the fluid concerned, but much more with the latter than with the former, and is very largely dependent upon the circulation of the fluid, that is, upon the rate at which it is renewed, and therefore upon its velocity. The conductivity or resistivity of such a transfer must, therefore, contain a term which includes the velocity of the fluid. Various tests by physicists have shown the specific conductance [illegible] to vary approximately as the [illegible] fluid.

From metal to air or similar gases, the mean velocity of flow being expressed in centimeters per second and the other units being square centimeters and gram calories, the transfer resistivity is approximately

$$R = \frac{36{,}000}{2 + \sqrt{v}}$$

and the transfer conductivity of the contact

$$k = 0.000028\,(2 + \sqrt{v})$$

From hot water to metal the relations are similar, but the conductivity is much better. Experiments show values as follows:

$$k = 0.000028\,(300 + 180\sqrt{v})$$

$$R = \frac{36{,}000}{300 + 180\sqrt{v}}$$

Illustration:—In the preceding case of the iron pipe, calculate the difference of temperature of the water in the pipe and the inner surface of the pipe, assuming the water to be passing through at a velocity of 4 centimeters per second.

Using the above given formula, the heat transfer per 1° difference would be

$$0.000028\,(300 + 180\sqrt{4}) = 0.0185 \text{ calories,}$$

and the difference to transfer 0.084 calories per second will be

$$\frac{0.084}{0.0185} = 4.6°$$

The inner surface of the iron pipe will be, therefore, continuously 4°.6 higher than the water, and, therefore at 14°.6; the outer surface will be continuously 0°.3 higher, or practically at 15°.

Illustration.—A steam radiator, surface at about 100° C., caused a current of hot air to rise having a velocity of about 10 centimeters per second, which was insufficient to keep the room warm. An electric fan was set to blow air against the radiator, which it did with a velocity of about 300 c.m. per second, and keeping the room comfortably warm. What were the relative quantities of heat taken from the radiator in the two cases?

The relative thermal conductivities of transfer were

$$2 + \sqrt{10} : 2 + \sqrt{300}$$

or

$$5 : 16$$

Showing over three times as much heat taken away per unit of time in the second instance.

This illustration proves the great efficiency which the metallurgist may attain in air cooling of exposed surfaces, by blowing the air against them instead of merely allowing it to be drawn away by its ascensive force.

Problem 21.

Hot air of the volume of 33,000 cubic meters per hour passes through an iron pipe exposed to the air, 30 meters long, 1.5 meters inside diameter, thickness of walls 1 centimeter, and lined inside with 5 centimeters of fire brick. Assume the hot air entering at 1000°, the outside air to be at 8°, the coefficient of internal transfer $0.000028\,(2 + \sqrt{v})$, the conductivity of the fire bricks 0.0014, of the iron 0.14, of external transfer $0.000028\,(2 + \sqrt{v})$, the velocity of the wind against the outside 6 kilometers per hour.

Required.—The temperature of the hot air leaving the pipe.

Solution.—The mean temperature of the iron is the [illegible] which conditions the mean velocity of the [illegible] flow of heat towards the outside. If, therefore, we let [illegible]

... temperature, the solution can be stated in the ... terms. We then have:

Volume of air per hour, at 0°, = 33,000 cubic m.

Volume of air per second, at 0°, = 9.167 "

Volume of air per second, at t, $= 9.167 \frac{t + 273}{273}$

$$\frac{1000}{3600} (19{,}998{,}000 - 16.414\,t - 3.564\,t^2) = \frac{1}{\frac{1}{0.000028\,(2 + \sqrt{595})\,(1 + \alpha t)} \div \frac{1}{0.00028} + \frac{1}{0.14} \div \frac{1}{0.000515}} \times 1{,}423{,}000 \times (t - 3)$$

Cross-section of inside of tube

$(1.5 - 0.1)^2 \times 0.7854 = 1.54$ sq. m.

Velocity of air at t, in pipe

$$9.167 \frac{t + 273}{273} \div 1.54 = 5.95 \left(\frac{t + 273}{273}\right) \text{ m. p. s.}$$

$$= 5.95\,(1 + \alpha t) \text{ m. p. s.}$$

$$= 595\,(1 + \alpha t) \text{ c.m. p. s.}$$

Coefficient of internal transfer =

$$0.000028\,(2 + \sqrt{595})\,(1 + \alpha t)$$

Coefficient of conductance of 5 c.m. of fire-brick lining =

$$0.0014 \div 5 = 0.00028$$

Coefficient of conductance of 1 c.m. of iron =

$$0.14 \div 1 = 0.14$$

Coefficient of external conductance ($v = 278$ c.m. per sec.) =

$$0.000028\,(2 + \sqrt{278}) = 0.000515$$

Total fall of temperature of air $= 2\,(1000 - t)$

End temperature of the air $= 1000 - 2\,(1000 - t)$

$$= 2t - 1000$$

Mean specific heat of air per cubic meter between 1000 and end temperature =

$$0.303 + 0.000027\,(2t)$$

Heat given out by the air per hour =

$$33{,}000 \times 2(1000 - t) \times (0.303 + 0.000027\,(2t)) =$$

$$19{,}998{,}000 - 16.414\,t - 3.564\,t^2$$

This heat is the quantity transferred through the pipe in kilogram calories. The outside surface of the pipe may be taken as the conducting surface, because the larger part of the resistance to the flow of heat takes place there. If we desired to be more exact, the mean area of iron, fire-bricks and the inner surface could be each used separately. The outside ... being 152 centimeters, and the length 30 meters, the ... surface is $1.52 \times 3.1416 \times 30 = 142.3$ square meters $= 1{,}423{,}000$ square centimeters, the total driving force is the ... temperature minus that outside, or $t - 3$, and the total thermal resistance is the sum of the four thermal resistances in ..., that is, the sum of thermal resistance gas to fire-bricks =

$$\frac{1}{0.000028\,(2 + \sqrt{595})\,(1 + \alpha t)}$$

thermal resistance of fire-brick lining =

$$\frac{1}{0.00028}$$

... =

$$\frac{1}{0.14}$$

... =

$$\frac{1}{\ldots}$$

The ... of the sum of these four resistances is the thermal conductance of the system per square centimeter, which, multiplied by the conducting surface, 1,423,000 square c.m., and by the total difference of temperature, $t - 3$, gives the heat transferred per second in gram-calories.

We, therefore, have the final equality expressed as the heat given up by the air, per second, equals the heat transmitted to the outside air per second; *i. e.*,

Whence $t = 965°.5$

And the temperature of the air at the end of the tube is

$$2t - 1000 = 931° \qquad (1)$$

It would be interesting to compare the temperatures of the inside and outer surfaces of the tube. The air inside is at a mean temperature of 965°, the heat transmitted per second is 0.1574-gram calories per square centimeter of outer surface, whose thermal conductivity is 0.00515, and, therefore, the outside surface must be

$$\frac{0.1574}{0.000515} = 306°$$

hotter than the surrounding air; that is, must be at 309° C.

The extra loss of heat by radiation from this outside surface would be small at that low temperature, but has not been allowed for in the working of the problem.

For such metallurgical problems we need the data as to the thermal conductivity or resistivity of ordinary furnace materials. These have been determined in but few instances, and in most cases not at the high temperatures at which they are practically used. A whole series of physico-metallurgical experiments is needed just upon this point. The following are probably nearly all that have been determined and the values published. The unit k is the C. G. S.-gram calories unit, the same as used for the metals.

	k.
Ice (datum useful in refrigerating plants, where pipes become coated with ice, as in Gayley's method of drying blast)	0.00500
Snow	0.00050
Glass (10° — 15°)	0.00150
Water	0.00120
Quartz sand (18° — 98°)	0.00060
Carborundum sand (18° — 98°)	0.00050
Silicate enamel (20° — 98°)	0.00040
(Explains the small conductance of enamelled iron ware.)	
Fire-brick, dust (20° — 98°)	0.00028
Retort graphite dust (20° — 100°)	0.00040
(Datum useful where articles are packed in this poorly conducting material.)	
Lime (20° — 98°)	0.00020
(Datum would be highly useful for oxyhydrogen platinum furnaces, if it were only known at high temperatures.)	
Magnesia brick, dust (20° — 100°)	0.00050
Magnesia calcined, Grecian, granular (20° — 100°)	0.00045
Magnesia calcined, Styrian, granular (20° — 100°)	0.00034
Magnesia calcined, light, porous (20° — 100°)	0.00016
Infusorial earth (Kieselguhr) (17° — 98°)	0.00013
Infusorial earth (0° — 650°)	0.00038
Clinker, in small grains (0° — 700°)	0.00110
Coarse ordinary brick dust (0° — 100°)	0.00030
Chalk (0° — 100°)	0.00028
Wood ashes (0° — 100°)	0.00017

Powdered charcoal (0° — 100°)	0.00022
Powdered coke (0° — 100°)	0.00044
Gas retort carbon, solid (0° — 100°)	0.01477
Cement (0° — 700°)	0.00017
Alumina bricks (0° — 700°)	0.00204
Magnesia bricks (0° — 1300°)	0.00020
Fire-bricks (0° — 1300°)	0.00310
Fire-bricks (0° — 500°)	0.00140
Marble, white (0°)	0.0017
Pumice	0.0006
Plaster of paris	0.0013
Felt	0.000087
Paper	0.00040
Cotton	0.000040
Wool	0.000035
Slate	0.00081
Lava	0.00008
Pumice	0.00060
Cork	0.00072
Pine wood	0.00047
Oak wood	0.00060
Rubber	0.00047

A study of the above table will in many cases show the metallurgist how conduction of heat can be checked, and to what degree. The substance chosen must be able to stand the temperature without being destroyed; but it is in many cases possible to use one kind of material inside, where the heat is greatest, and a material of much poorer conductivity outside, where the heat will not destroy it. Such compound linings or coverings may be very advantageous. Infusorial earth is one of the very best insulators for moderate temperature; above a bright red it loses efficiency greatly, and is then hardly better than powdered fire-brick.

Radiation.

A body placed in a vacuum, with no ponderable substance in contact with it, radiates heat to its surroundings. As far as experiments have gone they show the reliability of Stefan's law, that the amount of energy radiated is proportional to the difference between the fourth process of the absolute temperature ($C° + 273$) of the hot body and of its surroundings. With the surroundings at 0° and the hot body at 100°, Peclet determined the following amounts of radiation which we give here in C. S.-gram calorie units, that is, the number of gram calories radiated per square centimeter of surface, per 0° to 100° difference of temperature, with the understanding that this quantity gives the heat radiated in the range 100° and 0°. For other ranges of temperature Stefan's law would have to be used, starting with these figures as a basis; that is, the quantity of heat radiated from 100° to 0° represents a difference of 13.8×10^8 between the fourth powers of the absolute temperatures 273 and 373, and for any other numerical difference between the fourth powers of the two absolute temperatures concerned, a corresponding value for the heat radiated in that range could be calculated.

Polished silver	0.00054
Silvered paper	0.00177
Polished brass	0.00108
Copper	0.00068
Zinc	0.00102
Tin	0.00090
Polished sheet iron	0.00189
Leaded sheet iron	0.00273
Ordinary sheet iron	0.01164
Russian sheet iron	0.01410
New cast iron	0.01332
Oxidized cast iron	0.01410
Glass	0.01222
Paper	0.01583
Lamp-black	0.01684
Building stone	0.01500
Plaster	[illegible]
Wood	[illegible]

In order to [illegible] experiments that the heat radiated per square centimeter of surface, per degree difference of temperature, is not a constant, but varies with the temperature, as follows for the range from 0° to [illegible]:

100° to 0°	[illegible]
150° "	[illegible]
200° "	3.3
300° "	[illegible]
400° "	[illegible]
500° "	18.3
600° "	[illegible]
700° "	[illegible]
800° "	45.3
900° "	[illegible]
1000° "	[illegible]

It it is thus possible to calculate the heat radiated from a hot surface, and to add it to the heat transferred to the air by contact. In practice the total heat loss from the outside surface equals the total heat coming through the walls, which at the outer surface splits into two parts. The one is transferred to air by contact, calculated from the laws of transfer already discussed, and the other radiated freely without contact, and calculated from the data just given. If the temperature of the outer solid surface is carefully taken, both of these amounts of heat can be calculated, and the sum is the total heat being lost by transmission through the wall of the furnace.

Containers for Alkaline and Lead Accumulators.

By M. U. Schoop and C. Liagre.

For accumulators with caustic potash or soda solutions as electrolyte it is absolutely necessary that the container be absolutely dense and not attacked by the electrolyte.

Materials which satisfy these requirements are hard rubber and iron or steel. Glass is slowly attacked with the formation of silicate, wood is attacked with the formation of oxalate.

The following notes are intended to sum up the present status of the art of manufacture of containers of hard rubber and sheet steel:

Hard-Rubber Containers.

Hard rubber (ebonite, vulcanized rubber) is produced by treatment of a mixture of rubber and sulphur at a high temperature. According to the duration of the vulcanizing process, and according to the proportions of rubber and sulphur in the mixture, it is possible to produce any desired degree of elasticity, so that a very soft and elastic material or a hard and brittle one may be produced at will.

It is now possible to produce a material, perfect in quality in every respect, while most hard rubber boxes which were on the market five or seven years ago, left much to be desired. The density of hard rubber is about 1.2. Pure hard rubber is insoluble in caustic solutions. For containers for alkaline accumulators the use of this material, therefore, suggests itself. But even hard rubber receptacles of best quality have some disadvantages: they are breakable and are sensible to strong variations of temperature. For instance, at low temperatures cracks may occur spontaneously, while with [illegible] the hard rubber becomes plastic, so that changes of form take place.

The thickness of hard rubber walls [illegible] millimeters (about [illegible] inch), while with large [illegible] the vessel the thickness [illegible] 100 to 200 mm. On the other hand with [illegible] it is possible to make the walls [illegible] meter thick [illegible] ter of forty [illegible] in volume. Moreover, by [illegible]

[illegible] of weight is realized which may amount to 50 per cent. Finally, it should be taken into consideration that hard-rubber containers are quite expensive, and that storage battery factories are usually not in a position to undertake the manufacture, so that they must depend to some extent on the manufacturers of hard-rubber boxes. It will be shown below that the manufacture of metallic boxes offers no special difficulties, so that they may be made in the storage battery factory itself.

However, hard-rubber containers will undoubtedly maintain, for some time to come, their prominent position for all those purposes in which transportable lead accumulators are used. Besides, hard rubber is very suitable for insulating purposes within the alkaline accumulator. For these reasons a discussion of the properties peculiar to hard-rubber material should not be without interest, especially as the following notes are based on practical experience.

Pure hard rubber, which would contain no other substances but rubber and sulphur, would cost from 15 to 20 francs per kilogram (about $1.40 to $1.80 per pound). This is a price which would be out of all proportions to the present prices of storage batteries.

It is, therefore, easily understood that the commercial hard rubber contains all possible substitutes and is very often almost anything but a vulcanized mixture of rubber and sulphur. Nevertheless, it is possible to make fairly good boxes with a minimum of rubber, in spite of the use of foreign organic materials, of which the following are mostly used: Resin, pitch, fat oils, bitumen, and especially artificial rubber, so-called "factis." Factis is obtained by vulcanizing for some time various vegetable oils with minium or magnesium peroxide, or sometimes with a mixture of boiled linseed oil and sulphur. Factis is not attacked by sulphuric acid of the usual density, but is attacked by caustic solutions. The use of "factis" in hard rubber is, therefore, of no account as long as the containers are intended for lead accumulators, but should not be permitted for containers of alkaline accumulators. The presence of "factis" may be easily proven by treating a sample of the hard-rubber material with alcoholic caustic soda.

There is also a large number of mineral substitutes. Among them are barium sulphate, talcum, asbestos, in form of a powder, kaolin, lime, magnesium carbonate, silicon, zinc oxide, lead carbonate, lead sulphide and minium. In many cases hard-rubber scrap is also added. If the scrap is obtained from different hard-rubber makes, this method is extremely dangerous, since in this way all possible foreign materials may get into the final product, and in the end no longer any control is possible.

In Table I we give the results of analysis of hard-rubber containers made by different companies.

TABLE I

	Soldiers Free Pipes	Different Samples of Hard Rubber.				
[illegible]	1.20	1.15	1.55	1.05	1.65	1.58
W[illegible]	0.50	1.00	0.10	1.00	0.10	0.65
A[illegible]	1.50	1.25	0.25	31.00	27.50	16.25
[illegible]	5.50	11.00	13.50	12.50	8.25	5.50
[illegible]	6.00	5.00	12.00	traces	5.25	0.50
[illegible]						
[illegible]	2.00	4.00	5.00	9.50		1.50
[illegible]	81.10	34.50	28.15	19.6	32.85	42.85

(1) Aluminium silicate, chalk, iron oxide.
(2) Magnesium silicate.
(3) Magnesium silicate, chalk, silicon, iron oxide
(4) Barium sulphate, chalk, lead carbonate
(5) Magnesium silicate, chalk
(6) Magnesium silicate, iron oxide

Besides the use of substitutes, which may cause trouble (though this does not necessarily follow), certain flaws, due to [illegible] account which shall now be briefly discussed.

Hard-rubber containers are made as follows: Rubber is first [illegible] after adding the other materials which [illegible] the mixture in a degree of consistence [illegible]

The dough is rolled by calenders to [illegible] sheets of proper size and placed on the pattern, which may be made of cast iron or of an easily fusible metal (lead-antimony alloy), and which has the inside dimensions of the container to be made. Metallic plates, mostly of cast iron, are then pressed on from the outside; they determine the outside dimensions of the hard-rubber box. These outer cast-iron plates are maintained in position by screws. The whole block (containing the hard-rubber pieces between the inside metal pattern and the outside metal plates) is now placed in the vulcanizing vessel.

The flaws, which occur most frequently, are due to air bubbles enclosed in the hard rubber or to conducting foreign substances, like graphite, pieces of metal, etc.

In order to determine the reliability of the finished boxes and to test the homogeneity of the hard rubber, the following method of testing the boxes with the induction coil is to be recommended:

The container *A* to be tested (Fig. 1) is partly filled with water and placed in a vessel *a*, which is also filled with water. The two liquids are, therefore, separated from each other by the hard-rubber walls of the box *A*. Connections are then made with the secondary poles of an induction coil (or transformer) *T*.

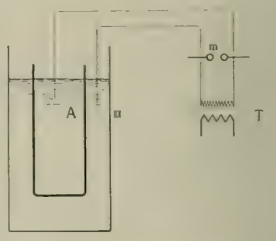


FIG. 1.—TESTING HARD-RUBBER CONTAINERS.

Care must be taken that the upper part of the hard-rubber box, which is above the water level, is well dried, since the sparks would otherwise jump above the top. The spark length *m* is now adjusted and the induction coil is set into operation. The length of the spark gap *m* depends, of course, on both the thickness and the quality of the hard rubber, as well as on the induction coil. If the latter gives, for instance, in air a spark length of 30 centimeters, the distance between the brass globes is diminished until a continuous spark discharge takes place. One will find in this way a spark length of about 5 to 6 mm. per mm. of wall thickness, on the supposition that the hard rubber is perfect in every respect. With too great a spark length sparks pass no longer, but the so-called silent discharge takes place. If the spark length is too short, it is also impossible to detect any flaws in the hard rubber, since the sparks, of course, pass along the path of minimum resistance.

With some experience and with proper adjustment of the distance between the brass knobs, it is easy to detect small flaws of manufacture. One has to observe whether the sparks stop playing or a visible discharge passes through the hard-rubber wall. If, in this way, a flaw is detected in the hard rubber this flaw is suitably burned out, and any conducting foreign bodies are thereby definitely removed. The box is then pasted at the place of the flaw with gutta percha or with a thick solution of para and bisulphide of carbon, and is vulcanized a second time.

In most cases repairs do not offer good prospects of permanent success, so that it is better not to use at all containers having flaws.

When hard rubber, which contains other substances, especially fat oils, mixed with the rubber and sulphur, is brought in contact with a caustic solution (as used in the Edison accumulator), troublesome formation of foam occurs.

(To be concluded.)

SMOKE ABATEMENT.—This subject was discussed by the New York Section of the Society Chemical Industry on Feb. 23. The speakers were Health Commissioner F. Darlington, Mr. A. de Frampe, Mr. I. H. White, on down draught furnaces; Mr. Walker Bowman, on filtering smoke; Mr. E. H. Foster and Mr. A. A. Cary, on furnace construction.

Notes on Electrochemistry and Metallurgy in Great Britain

(From Our Special Correspondent.)

Metallurgical Papers Read During January.

Those attending the January meeting of the Institution of Mining and Metallurgy had four papers for their consideration, which were discussed in the following order:

(1) "Notes on Metals and Their Ferro-Alloys Used in the Manufacture of Alloy Steels," by O. J. Steinhart.

(2) "Pyritic Smelting," by R. C. Alabaster and F. H. Wintle.

(3) "The Acme Combined Concentrating Table," by L. H. L. Huddart.

(4) "Stadia in Careful Work," by A. H. Webb.

Ferro-Alloys.

Dr. Steinhart's paper can be dealt with only in abstract.

"There are many people who, having found some of those rarer minerals which are used as raw material for the production of metals in question, have formed erroneous and exalted ideas as to their consumption and value. These mistakes have in many cases been the outcome of the want of reliable information, which is frequently most difficult to obtain, the steel makers and manufacturers of these metals being, not unnaturally, somewhat reticient as to details. Before dealing more fully with the various metals and alloys forming the subject of these notes, it is well to state at the outset that the present-day requirements of the steel makers with respect to the purity of the elements they add to the various steels are most stringent and exacting, and those intending to evolve a new method, or wishing to add another metal or alloy to the already not inconsiderable list of the latter, should bear this in mind. The fatal effects which very small quantities of such elements as sulphur, phosphorus, etc., have upon steels form such elementary knowledge that it goes without saying that none of these should be present to any appreciable extent in the materials added to the steel." Another point often overlooked by novices is the presence of several per cents of carbon, so often found with electric furnace products, which are as undesirable as the other impurities just referred to. Therefore, generally speaking, the freer the metals and alloys are from these interfering substances the more welcome they will be to the steel maker, and the better will be the prices paid by him. The great changes brought about in the recent industrial development and application of the electrical furnace, based chiefly upon the early laboratory experiments of H. Moissan, have had a most important bearing upon the manufacture of our alloys and metals, but experience has shown that quite a number of these, as for instance, nickel and tungsten, are still best prepared by the older chemico-metallurgical methods, chiefly on account of the difficulty of producing pure low percentage carbon alloys in the electrical furnace. The Goldschmidt aluminium reduction process has likewise yielded a ready method for preparing the pure metals and alloys, although they are somewhat costly. The metals and their ferro-alloys used by steel makers in the order of their consumption and importance are probably as follows: Manganese, nickel, chrome, tungsten, molybdenum, vanadium, and also, experimentally, cobalt, titanium, uranium, and recently tantalum.

On the purely electrometallurgical side Dr. Steinhart had much to say. With regard to nickel, reference was made to Mr. Ulke's contribution to *Electrochemical Industry* in 1903, and the following remarks were added: Lately semi-metallic granules and scraps easily produced by reducing nickel-oxide were substituted for cast-nickel anodes, and thus the cost of refining was reduced to less than 1 cent per pound of refined nickel. Quite a number of processes have been evolved by Frasch, Ulke, Hopfner and others, with the object of dealing with copper-nickel mattes. The matte was generally either electrolyzed in a solution, such as ferricsulphate, cupric, chloride, sodium chloride, sulphuric or other acids and the metals deposited from the mixed electrolyte by fractional electrolysis, or the matte was [illegible] and the metals thrown down by the current. On the [illegible] these processes are very complicated, and have not led to practical results, although some nickel has been produced at large experimental installations. (See, however, the description of the successful Browne process in our Vol. I., pp. 48 et seq.) The Lake Superior Power Co., in 1902, attempted, in conjunction with E. A. Sjostedt, to produce ferronickel in the electrical furnace from dead-roasted Canadian ores, but this process never came into practical use, although a large plant was put down. On the other hand, the companies, "La Societe Electrometallurgique de Froges" and "Le Ferronickel," have produced ferronickel commercially. Dr. Steinhart regretted, however, to say that he was unable to obtain any information from these concerns.

With regard to chromium, some new information—new at least in England—was supplied concerning the Willson Aluminium Co., of New York, who have electric furnaces at Kanawha Falls and also at Holcomb Rock, both in Virginia. Kanawha Falls uses three dynamos of 800 kw. each, and those at Holcomb Rock one-third of that power. Alternating current of 110 volts pressure is used at Kanawha Falls, the current of 22,000 amps. being divided over seven furnaces, all crucible furnaces of partly circular, partly square and rectangular cross-section. The combined works can turn out about 200-250 tons of ferrochrome per month. The quality of the ferrochrome seems to be improving every year. At one time the alloy was rather soft and high in carbon. They can now turn out ferrochrome of good solid quality, testing 5-6 per cent maximum carbon.

Where special low carbon steels are required alloys containing as little as 1.271 per cent, or even 0.555 per cent, are obtainable. Passing reference was also made to Dr. Goldschmidt's aluminothermic process, chromium produced by it having considerable purity (from 0.741 to 0.055 per cent in the case of two analyses cited), but being rather expensive, at from 2s. 3d. to 2s. 6d. per pound.

As regards tungsten, the chemistry of various methods of treating wolfram ores was discussed, and the samples of ferrotungsten from the electrical furnace which had come under Dr. Steinhart's notice were characterized as "not very pure or suitable for tungsten steels." Mention was also made of the occurrence, treatment and applications of molybdenum and tungsten. As Dr. Steinhart confessed, it was impossible, even in a remote manner, to touch upon the special characteristic properties which the metals under consideration imparted to steel and iron. Much of the descriptive matter had been purposely omitted as to the properties of the metals and minerals, facts which could be found in any good text-book on chemistry and metallurgy. Finally, it might be said that with the many pure metals and alloys now at their disposal, the number of useful alloy steels which may be produced from these offers a field of such gigantic magnitude that it required a master mind and ceaseless investigation to select those which should give the desired result with the least expenditure.

Expiring British Electrochemical and Electrometallurgical Patents.

Compared with the United States or Germany the number of patents which have run their full term and are about to expire during any year, are relatively very few in England. The system of intermittent payments of fees, and the immediate lapsing of any patent on the fees being unpaid, weeds out a large number of patents which would remain in force where a single outright payment is made at the outset. Therefore, in chronicling the survivors expiring this year (unless specially extended by recommendation of the judicial committee of the Privy Council), only a few are of interest to readers of this journal, and may be conveniently under [illegible] headings.

Firstly, of these connected with electrolytic alkali industry, mention must be made of No. 9,347, of 1892, issued to Carl Kellner, which is concerned with the separation of the undecomposed sodium chloride from electrolytically produced caustic soda. The methods in use consist in mechanical separation of the sodium chloride crystals by sieves from a heated vessel, the washing of the crystals and evaporation of the resultant liquor. It is claimed that this arrangement, by separating the alkali contained in the solution by repeated and systematic displacement by means of brine, will allow of the alkali being obtained in an almost pure condition, and that the whole of the residual undecomposed electrolyte is separated and returned into the cycle (of evaporators, displacement vessels, etc.). Another patent, by H. Young Castner, No. 16,042, of 1892, which invention is described as consisting of the employment of a mechanically moving body of Hg, or other liquid metal or alloy between the anode and cathode departments of a decomposing cell, so that the current must flow through the moving metal. This is claimed to reduce the counter-electromotive force, and render it possible to carry on the action with a very high current efficiency. Another patent, Charles Kellner's, No. 17,169, of 1892, in which the cathode is formed of a thin film of mercury contained in partitions permitting a free passage to the current. Various methods of making the troughs are described, such as a wooden trough lined with paraffined linen and provided with a groove in which is acid-proof cement. Sodium forms an amalgam with the mercury and is converted by water into caustic soda. The patent claims that this apparatus can also be used for separating other metals forming an amalgam with mercury. As a contrast to the mercury cathode cells, one of the Hargreaves-Bird patents also expires (No. 18,039, of 1892). This describes a cathode formed of wire gauze or perforated metal, on the anode side of which is material constituting a porous diaphragm. The porous diaphragm consists of a layer of fibrous material next the cathode, and a layer of stone-like material on the fibrous material. Briefly described as an improved electrode, made by depositing the materials which form it (asbestos or Portland cement, or plaster of paris) in the form of pulp, either direct on the gauze or on paper or felt laid on the gauze. A mixture of clay and silicate of soda is also used.

Of the metallurgical patents, No. 18,516, of 1892, is concerned with the obtaining of homogeneous deposits of metals. It was granted to E. X. A. Picard and J. V. Lamere, and covers the continuous hammering of the metal being deposited by means of an electromagnetic trembler device. Two patents standing in the name of Mr. T. L. Willson also expire; of these, No. 21,696, of 1892, covers a process bringing the metallic compound in a state of extremely fine subdivision into most intimate contact with a reducing agent before subjecting it to an electrical smelting process. It is applied to porous alumina or other metallic oxides. The inventor steeps it in a hydrocarbon until all the liquid is absorbed. Coal-tar, or heavy hydrocarbon oil or paraffin, is used, also salt. The other, No. 21,791, of 1892, relates to the treatment of refractory metallic compounds by mixing with carbon in sufficient proportion to prevent the formation of a bath of molten metal boiling up, and so [illegible] the process.

Two of Paul Héroult's [illegible] patents expire, both being concerned with the details of the electrodes. The earlier, No. 9,011, of 1892, relates to the production of ozone by silent electrical discharges from point-bearing metallic electrodes. The electrodes here described are made in the form of a [illegible] sheet or double metal plates set between two metal [illegible] from which the points are separated by glass, mica, etc. The points [illegible] connected to one terminal of a [illegible] and [illegible] may be used as the separating substance instead of glass. Various forms of utilizing these electrodes are described, either with a natural draught or with air forced in by a pump. The other patent, No. 21,794, of 1892, is a development of the former patent, and relates to the use of plates of metal cast, cut or stamped, so as to form sharp edges, tongues or angles to act as points for the discharges. One showed in appearance like fret-saws strung on a frame.

For the electrolytic production of oxygen and hydrogen only one patent, P. Garuti's, calls for mention, being No. 16,588, of 1902. It provides for a lead-lined bath divided into sections by metallic longitudinal partitions, each containing one anode and two cathodes. Two large cup-like chambers receive the discharging gases, the cells giving off the gas separately. The use of metal diaphragms for separating the cells is claimed as offering small resistance.

In regard to secondary batteries mention may be made of H. W. Headland (No. 15,120, of 1892), entitled "Improvements in Accumulator Plates," which are to be built up of hollow tubes or bars, pierced with numerous holes, and filled with paste, which projects through the holes. There are also No. 22,987, of 1892 (W. W. Griscom), for lead plates first pitted by the action of an alkaline solution, and then formed in the familiar Plante manner, and No. 24,127, of 1892 (F. King), for special plates having projections or pins covered with an insulator touching the opposite plates, the object being to support the plates and to limit buckling.

The Council of the Faraday Society.

At the meeing of the Society on Jan. 30, the nominations for the Council for 1906-07 were announced. Lord Kelvin retains the presidential chair for another twelve months, and the new vice-presidents (in place of Mr. Swinburne and Alexander Siemens) are Prof. A. Crum Brown and Prof. Hittorf. From the Council, Mr. Cowper-Coles and Mr. G. T. Beilly retire by rotation, and are to be replaced by Dr. T. M. Lowry and Mr. W. R. Cooper.

The January Meeting of the Faraday Society.

Again a "three-paper evening." Virtually, but not literally, for doubts were expressed as to whether the first contribution was a paper at all, and the second was frankly a demonstration.

We began with the third instalment of M. Adolphe Minet's compilation of notes on electric furnaces. This was the third paper presented to the Faraday Society. How many more there will be one does not know; perhaps the grandchildren of the present Council may closure the series. Whatever be the number of succeeding papers one thing is manifest, and that is their potential archæological interest.

Monsieur Minet being absent, Mr. Murray Morrison acted in *loco parentis* in the first instance. Later on, expressing his pleasure at having acted as intermediary, he personally disclaimed identity with many of M. Minet's views and historical data. Free from the trammels of deputyship, Mr. Morrison was interestingly retrospective. For instance, one of the Heroult's patents, described by Minet as dating back to 1887, really dated back to early in 1886 or late in 1885, a dishonest patent agent annexing the fees first sent. As to the author's figures of the aluminium output and power used, they "were not at all accurate." As director of the British Aluminium Co. he regretted he was not at liberty to controvert certain figures. Referring to the early history of aluminium, the speaker observed that, initially, aluminium had been regarded as only useful for alloys, and when Heroult first offered to produce cheaper aluminium he was advised to confine his attention to alloys.

Dr Burns then pertinently asked whether the paper should have been printed in extenso, in view of Mr Morrison's repudiation of certain of its facts? There was nothing really new. Mr Bertram Blount said that aluminium was a so entrancing, indeed, almost per technic, subject, that the object of the paper had been overlooked. Its aim was mainly historical. For his own part he would summarize the history of aluminium as belonging, firstly, to the experimenter, and then to the academical chemist, whose work was equally notable for

its originality and its impracticability. There was no reason why the discussion should be confined to the aluminium furnace. There was that closely guarded secret—the phosphorous furnace. Did anyone know how phosphorous was prepared electrically? Returning to aluminium, Mr. Blount, after some further remarks, concluded by observing that the aluminium industry was now settled and lucrative, but there was still room for great scientific development.

Another speaker asked about the purposes for which calcium could be actually used. Prof. F. W. Harbord observed that he had had no practical experience of aluminium manufacture. Prof. Huntington (who was in the chair) appealed to the previous speaker for his opinion as to a metallurgist on the resultant effect of using calcium in place of aluminium in steel work, and received the reply that a similar effect could be expected. Only small traces would be necessary, otherwise the metal would be very spongy if cheaper calcium might be used, but at present only about 4 ounces of aluminium was required per ton.

I must defer until next month my account of the rest of the meeting, as my space is restricted.

MARKET QUOTATIONS DURING JANUARY.

During January prices [illegible] the closing prices for some of the [illegible] ments best serve to indicate the general [illegible] Ammonia sulphate, £12 11 3 per ton, [illegible] per cent), £4.10 per ton; crude carbolic acid liquid (62½ per cent), 1s. 2d. per gallon; bicarbonate of soda, £6.15 per ton; caustic soda (77 per cent), £10.12.6 per ton; copper sulphate, £25 per ton, and zinc sulphate, £6.15 per ton.

In the metal market, the chief feature is a decline in Cleveland warrants to £2.12.4, and hematite to £3.9. In regard to these small declines, the general election has been in part responsible. There is still a heavy demand, and tariff reform being apparently relegated to the Greek Kalends, a source of apprehension and unsettlement has been removed. Tin has fluctuated rather wildly, reaching £167.10 by the middle of the month, falling to £162 in three days, then recovering and closing at £165.5, an uncomfortably high price. Copper closed at £78.12.6, not falling to less than £77.5 during the month. English lead closed at practically £17, a decline from the high price of £19, which was touched on Jan. 5.

London, Feb. 7.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

Industrial Electrochemistry.

Bleaching Liquor by Electrolysis.—In our Vol. I., p. 439, a concise summary was given by W. H. Walker of the various processes and apparatus in use for the production of hypochlorite by electrolysis of sodium chloride in cells without diaphragm. Interesting results obtained with one of the apparatus described there, namely, the Hass & Oettel cell, in a New England textile mill, were given in a paper by H. S. Duckworth, before the New England section of the Society of Chemical Industry. The following table is given concerning the work of the cell:

Capacity of tank = 126 U. S. gallons = 477 litres.
Brine at 21½° Twaddell = 169 pounds salt = 76.66 kilos.

	Hours Worked					
	1	2	3	4	5	5½
Grams Cl per litre.....	4.65	7.90	10.44	12.58	13.91	14.38
Temperature °C.....	17	21	21	20	21	21
Amperes at 110 volts..	79	74	73	70	65	64
Horse-power hours....	11.7	22.6	33.4	43.7	53.3	58.0
Total Cl in grams.....	2,218	3,768	4,980	6,000	6,635	6,859
Salt per kilo. of Cl....	34	20.3	15.4	12.8	11.5	11.2

The following calculations of cost are made: The salt is valued at $6.10 per ton delivered. Thus 169 pounds of salt worked for 5½ hours give 6.859 kg. Cl = 15.12 pound. Three pounds of electrolytic chlorine are stated to do the work of 4 pounds of chlorine from bleaching powder, and, therefore, the 15.12 pound of active electrolytic chlorine is equal to about 60.48 pound of bleaching powder. The cost of the power is calculated first on the basis of water-power and then on the basis of steam-power. For water-power the horse-power-year is assumed to cost $15.00. The time of one run = to 5½ hours in the above table has been chosen, since it allows of two runs per day of 12 hours for five days in the week and one run on Saturday, which is in accordance with the usual hours of working of the textile industry in the East. The cost of the power is based on an assumption of 58 hours per week. The cost of steam-power is assumed to be $25.00 per horse-power year.

Under these assumptions the author gets the following comparative costs of the hypochlorite process and for the use of bleaching powder. First for water-power he gets in cents:

	Cents.
Cost salt	51.54
Cost power	27.84
Total	79.38
Cost of equivalent amount (60.48 pounds) bleaching powder at 1¼c. =	75.00
Freight on same	7.00
Total	82.60

Secondly for setam-power he gets—

Cost salt	51.54
Cost power	46.40
Total	97.94
Cost of equivalent amount (60.48 pounds) bleaching powder at 1¼c. =	75.00
Freight on same	7.00
Total	82.00

The author discusses the advantages of the electrolytic production of hypochlorite, and points out that the process is not only applicable to the whitening of cotton in any form, but is especially adapted to the bleaching of flax. It is also applicable to the fibres used in paper making, also to the bleaching of jute, oil, artificial silk, viscose compounds, etc. The process is also used largely in linen bleacheries in Germany, and the author recommends it strongly for laundries. The full paper and the discussion which followed, are published in the *Journal of the Society of Chemical Industry*, Nov. 30, 1905.

Dissolved Acetylene.—Two papers, by Knoch and by E. Bournonville, read before the International Acetylene Association, are abstracted in the London *Engineering* of Dec. 29, 1905, and in *Science Abstracts* of Jan. 30. Acetylene is dissolved at a pressure of about 10 atmospheres in acetone, which is ab-

sorbed within some porous material held in steel cylinders. In America the porous material is made of a paste of asbestos in fine fibers and sodium silicate, moulded into discs fitting the steel cylinders, and dried at a gentle heat. In Austria the paste is made of charcoal and a cementing substance, run into the cylinders as a fluid, and then dried. Acetone expands as it dissolves the acetylene, and when saturated at 12 atmospheres, unit volume has become about 1.5 volumes. When the cylinder is charged to 12 atmospheres, according to Kuchel, the solution of acetylene in acetone expands till it fills 64.3 per cent of the space within the cylinder, 15.7 per cent being occupied by compressed gas, and the remaining 20 per cent by the porous material. The acetone vapor passes off and is burnt with the acetylene. In practice 1,000 litres of acetylene carry off the vapor of 0.06 litre of liquid acetone. This is no disadvantage; the acetone vapor burns well and is cheaper per unit volume than acetylene. Experiments have also been made with a view to finding the extent to which the pressure is raised by increased temperature. From these, it appears that if a cylinder is charged to a pressure of 12 atmospheres at a temperature of 20° C., it might develop a pressure of about 24 atmospheres if exposed to the hottest rays of the sun. A cylinder charged at 12 atmospheres will contain about 100 times its apparent volume of available acetylene. The papers of both authors close with figures relating to cost, particularly with a view to the lighting of railway carriages.

Ozone by Silent Discharges.—Before the last meeting of the American Phys. Society, A. W. Ewell described experiments on the electric production of ozone by silent discharges, using an alternating current and plate electrodes separated by dielectric. According to the *Phys. Review* of February, the chief results are as follows: The yield per coulomb was found quite independent of the dielectric, and greater the higher the gas velocity. The yield increases with the current to a maximum and then decreases. Narrow, bare electrodes are most efficient. The yield per kilowatt-hour is greatest for a polar distance of a few millimeters. This type of electrodes is far more efficient than the point and plate and than other minor types.

Ozone by Electrolysis of Alkali Fluorides.—A brief paper recently presented by E. B. R. Prideaux before the Faraday Society (and abstracted here from advance sheets) refers to the production of ozone from aqueous solutions of KF and of HF, due to the action of nascent fluorine on water. The electrolysis of potassium fluoride seems more promising commercially than that of hydrofluoric acid, since potassium fluoride solutions are both easier to deal with and do not lose strength so rapidly by evaporation and attack of the containing vessel as solutions of hydrofluoric acid. The author's experiments show, however, that the yield of ozone from saturated potassium fluoride solutions at 0° C. is in all cases quite low, certainly not exceeding 1 per cent.

Electrolytic Preparation of Spongy Tin.—In the *Comptes Rendus* of Jan. 8, D. Tommasi describes an industrial method of preparing spongy tin, and gives figures with regard to the efficiency of the method. The following notes are taken from the London *Electrician*, Jan. 29. "The electrolytic cell contains two anodes of tin, and the liquid consists of fifty parts water, ten parts stannous chloride, and one part hydrochloric acid. The cathode is a large copper disc, capable of rotating on a vertical plane, and used for this purpose with its center to a bronze shaft. The disc is half submerged in the liquid. The upper half passes between two brass scrapers, which scrape off the tin deposit and depolarize the disc. The spongy tin is collected automatically and washed, the liquid being evaporated to the concentration of the bath and used over again. In one experiment, with a disc 30 cm. in diameter, an e. m. f. of 3 volts and a power of 120 watts, the amount of tin deposited per horse-power hour was 380 grams, the theoretical amount being 440 grams. The efficiency is therefore 86 per cent. But there is no reason why discs as large as 3 meters in diameter should not be used. With a disc of that size the amount of tin deposited would be 4,400 grams per horse-power-hour, or 105 kg. per day of 24 hours."

Metallurgy.

Copper.

Losses in Bessemerizing Matte.—Dr. E. Günther, in *Metallurgie* for Nov. 22, discusses the German patent 160,046, of Oct. 5, 1904, which contemplates stopping the Bessemerizing of matte when all the iron is slagged, and using the copper sulphide as anode in electrolytic refining vats, thus avoiding the losses of copper, silver, etc., occurring in the blowing to blister copper. Dr. Günther thought it timely to find what the losses of copper and silver are in this second half of the Bessemerizing, and to this end run a small converter in the laboratory of the Aachen Technical School. The lining was an aluminious sand brick, air pressures up to 350 mm. of mercury, temperatures determined by the Wanner optical pyrometer, time of blow to rich matte 25 to 30 minutes, to blister copper 45 to 50 minutes, temperatures in the first case 1,150° to 1,200°, in the second 1,250° to 1,280°. With a matte carrying 36.72 per cent copper, 0.1913 per cent silver, 2.78 per cent of silver, blown up to 71.15 per cent copper, the losses were 1.77 per cent of the copper and 2.78 per cent of the silver present; blowing the same matte to blister copper, the losses in fume were 20.14 per cent of the copper and 22.12 per cent of the silver present. Using a richer matte, carrying 46.42 per cent copper and 0.2443 per cent silver, the losses, when blowing it up to 77.06 per cent matte were 0.37 per cent of the copper and 1.59 per cent of the silver; but when blowing it to blister copper 21.11 per cent of the copper and 23.42 per cent of the silver.

These results prove the extraordinary large losses by volatilization occurring in the second stage of the Bessemerizing operation, and show that if it is possible to blow to rich matte and then to electrolytically treat it, these high losses can be avoided. The patentees of the German patent have, therefore, the correct idea of where great saving may be made, whether or not they have found the successful and specific remedy.

Bessemerizing Matte.—F. W. Hinrichsen and T. Watanabe made some very interesting experiments in Dr. Borchers' laboratory, at Aachen, on blowing matte with air containing above the normal proportion of oxygen. Their results are reported in *Metallurgie* for July 8. Using a small experimental converter, preheated by a blast flame, 25 kilograms of matte was blown in 35 to 39 minutes, using ordinary air, with a rise of temperature of 180° to 250° for low-grade matte and 15° to 70° for high-grade matte. When exactly similar amounts of the same mattes were blown with air containing 28.5 to 29.0 per cent of oxygen, the blows lasted 16 to 21 minutes, with rises of temperature of 335° to 390° for low-grade matte, and 85° to 105° for high-grade matte. The difficulties of blowing rich matte to blister copper are by this device largely reduced, and calculations are made as to a possible commercial application, but the present price of oxygen seems to still stand in the way. An important point in favor of running in this manner is that silica sand charged with the matte was melted down by the higher temperature to a homogeneous slag, thus saving corrosion of the lining to a considerable extent; a reaction which has been always found impracticable when blowing with ordinary air.

Copper and Arsenic.—The Königlich Sächische Bergakademie, at Freiberg, has recently, under the direction of Prof. K. Friedrich, taken up actively the subject of metallography, and a very comprehensive article on the copper-arsenic compounds may be regarded as the first fruits of this activity. The eighteen large pages which it occupies in *Metallurgie* for Oct. 22 are worthily filled, and the article is sure to attract attention from all directions. The twenty tests made were approximately every 2 per cent from pure copper to 50 per cent

copper. It was found impracticable to make a richer alloy than 44 per cent arsenic. The melting point curve falls with great regularity at the nearly constant rate of 18° for each per cent of arsenic, to the eutectic containing 21.5 per cent of arsenic, and melting at 682°. Rising sharply, the melting point rises at the rate of 19° per cent of arsenic to a maximum of 830° for the compound Cu^3As, containing 70.88 per cent of copper. Thence up to 44 per cent of arsenic the melting point again fell almost linearly, at the rate of 14° for each per cent of arsenic, to 651°. Within the range studied there is, therefore, only one true chemical compound, Cu^3As, and one eutectic at 21.5 per cent arsenic. Two other eutectics, however, were indicated by halts in the cooling curve after setting had taken place, one at 711°, corresponding to a transformation of the alloy containing 33 per cent arsenic, and another at 600°, corresponding to a possible eutectic at 48 per cent arsenic. Another halt, at 305°, found in alloys with 32 to 44 per cent arsenic, corresponds to some transformation in the solid alloy, the nature of which is unknown. The progressive constitution, starting with pure copper, is therefore,

	Per Cent As	*Melting Point.*
Cu	0	1084
Cu + eutectic of Cu and Cu^3As		
Eutectic of Cu and Cu^3As	21.5	682
Cu^3As + eutectic of Cu and Cu^3As		
Cu^3As	29.1	830
Cu^3As and Cu^5As^2 (isomorphous mixtures)		
Cu^5As^2	32.0	822
Cu^5As^2 + eutectic of Cu^5As^2 and Cu_xAs_y		
Eutectic of Cu^5As^2 and Cu_xAs_y (48 per cent arsenic)	48.0	600

Twenty-three fine photomicrographs accompany and illustrate the paper, and prove the conclusions from the cooling curves. The minerals, Cu^6As (Algodonite) and Cu^6As (Whitneyite) do not form, and their existence as compounds is doubtful. The mineral *Ledouxite*, identified by Richards, corresponds very closely to the first eutectic. The alloys with arsenic are comparatively feeble; those containing much arsenic lose some by heating a long time, powdered, at 300° C.; even Cu^3As is not fixed at a red heat, as is known also from the Plattner slagging test for cobalt, nickel and copper when combined with arsenic, in which copper is known to stay as Cu^3As as long as Ni^2As is present, but to lose some arsenic as soon as all the nickel is removed.

Lead.

Treatment of Galena.—Carmichael's British patent, 17,580, of Jan. 30, 1902, states that a mixture of PbS and $CaSO^4$ react at a low red heat, about 400° C., to form $PbSO^4$ and CaS. F. O. Doeltz considered the reaction to be *a priori* unlikely, and therefore made some very decisive laboratory tests, which settled the question. The results are reported in *Metallurgie* for October 8, and will interest all concerned with the treatment of lead ores. An intimate mixture of PbS and $CaSO^4$ was kept 1.5 hours at 400° C., in a current of CO^2, but no trace of reaction was to be detected. Heated to 850° for 1 hour a considerable amount of PbS distilled off into the cooler part of the tube, condensing there unaltered, but no change, even a microscopic one, could be detected in the mixture. By heating above the melting point of PbS (which is 930° to 940° C.) the mass was melted down at 1,100°, but still did not react. These results were completely substantiated by heating together an intimate mixture of $PbSO^4$ and CaS. Even at 400° this mixture turns gray from formation of PbS. $PbSO^4$ and Na^2S react on each other even cold when ground together in a mortar, and $PbSO^4$ and CaS show evidences of the same. From these tests the author concludes that Carmichel's process is based upon a wholly suppositious reaction, which, in fact, works only in the reverse direction to what is desired.

Lead and Galena.—Prof. Friedrich, of the Freiberg [illegible] and his assistant, A. Leroux, publish in *Metallurgie* for Nov. 22 [illegible] tests of PbS to Pb. The prepared [illegible] two substances in different proportions from pure lead to pure PbS (13.5 per cent. sulphur), and studied the mixtures [illegible] were an alloy of Pb and PbS. The melting point of PbS was determined as 1,092° C. (2,000° F.); the first 0.5 per cent of lead reduced the melting point 30°, but subsequent additions very little, 75 per cent lead to 25 per cent sulphur still melting at 1,000°. From this point the melting points fell off more quickly, being for 80 per cent of lead 950°, 85 per cent 920°, 90 per cent 880°, and 95 per cent 860°. It thus appears that the first 5 per cent of PbS present, equal to only 0.67 per cent of sulphur, raises the beginning of the setting point of lead 480°. This is one immensely important fact for lead smelters; another is that the two substances melt together in all proportions, showing a perfectly uniform transition, in the molten state, from Pb to PbS, with every proportion of sulphur from 0 to 13.5; a third is that no sign of any such compounds as Pb^2S, or Pb^3S, could be detached. The solution characteristic of the mixture was the fact that every mixture tested showed the presence of lead in it by a halt in the cooling curve at 327°. The photomicrographs confirm these conclusions, by showing a regular gradation from lead with scattered crystals of PbS to pure PbS.

Mercury.

Roasting of Ore.—Several Western papers (*Mining Reporter, Mining World, Pacific Miner*) have recently contained rather roseate descriptions of the new Dennis furnace, used at the Black Butte Mines, Lane County, Oregon, but if half what is claimed for the furnace is true, it is a great improvement on the best modern mercury furnace—the Scott-Huttner. It is stated that patents are pending, and for this reason detailed descriptions cannot yet be given. We are told, however, that the coal is first gasified, and then the ore heated by combustion of the gas, thus avoiding ash and soot in the gases; also that the ore is dried in one chamber, heated to about 200° C. (400° F.), then passes into a second compartment heated to 300° C. (600° F), then to a third, and finally to a fourth, heated to 800 to 900° C. (1,400 to 1,600° F.). The ore passes through the furnace in 4 hours, instead of 24, the capacity is 240 tons daily (against 40 tons), and the condensers are kept free from ore dust and soot, so that the mercury condenses clear. We will present to our readers as soon as it can be gotten an official report of the achievements of this Western wonder.

Zinc.

Review. In *Chemiker-Zeitung* of Dec. 2, 9, 16, 1905, F. Peters gives a concise summary of actual developments and tentative proposals which have been made since 1901 on use of electrochemical methods in the metallurgy of zinc. The summary does not seem to give much that is new, but should be useful for the many references which it contains. The author discusses electrolysis with aqueous solutions, with fused electrolytes and electric furnace methods.

Zinc Shavings.—London "Engineering" of Nov. 17 contains drawings and illustrations of a lathe specially designed for the manufacture of zinc shavings.

Iron and Steel.

Blast Furnace Practice. A very interesting note for blast furnace managers and students of blast furnace matters is that of E. de Loss, in the *Revue de Metallurgie* for November. This engineer had charge, for two years, of a blast furnace north of the Urals, in 60° north latitude, where in Summer the temperature marked 40° to 45° C. (104° to 113° F.) in the shade, and in winter, once for two consecutive days, stood at 51° C. (60° F.). Under such conditions [illegible] ing engine running the same speed, the weight of [illegible] and

consequently of oxygen) blown into the furnace per unit of time was inversely as the absolute temperatures, viz.:

$$\frac{45 + 273}{51 + 273} = \frac{318}{222} = 1.43 \text{ to } 1.00$$

or 43 per cent more oxygen was blown in, in the coldest weather than in the hottest. Unfortunately, Monsieur de Loisy does not point the moral to adorn his tale, by telling whether the furnace really ran anything like that much faster in the cold weather. Undoubtedly it did, however, make much more iron in the Winter; and this recalls the predictions made when hot blast was first introduced, which were to the effect that since a furnace made more iron in cold weather, when it was fed with cold air, than it did in summer when it was fed with warm air, that, therefore, heating the blast must result in lessening the production of the furnace.

Fusibility of Blast Furnace Slags.—O. Boudouard, an assistant of Prof. Le Chatelier, published in *Revue de Metallurgie* for June, 1905, an exhaustive account of his laboratory researches on this subject. The same material was incorporated in a paper read later before the Iron and Steel Institute. A great deal of careful work was done, but the experimenter made the capital mistake of determining the temperature by Seger cones, thus removing from the outset the possibility of scientifically reliable temperatures being obtained; for, no matter how careful the calibration of the cones, the conditions under which they were calibrated can never be exactly reproduced during an experiment—leaving out of consideration the exactness of the comparison with the cone of substance being tested. It is, therefore, probable that the temperatures given in the paper are throughout unreliable. One conclusion is that when the ingredients of the slag are finely ground and intimately mixed, the temperature of formation of the slag is practically identical with its melting point; but although this may be so, it must be borne in mind that in a blast furnace the slag-forming ingredients are neither finely ground nor intimately mixed, so that, in practice, the temperature at which a slag forms is almost inevitably far above its melting point.

Hardening and Tempering.—Demozay, in *Revue de Metallurgie* for October, records some illuminating tests made on tempering. He measured the tempering by the resulting hardness, as given by the Brinell steel-ball test. Taking a piece of hardened steel, it was found that equal degrees of tempering were produced by heating 5 minutes at 775°, 10 minutes at 766°, 15 minutes at 755°, 20 minutes at 745°, or 25 minutes at 735°; while at 730° no tempering took place in over 30 minutes heating. For this piece of steel, heated between 775° and 735°, each extra 2° of temperature above 735° compensated for 1 minute shorter time of heating. From this series of tests the author concludes that if the temperature of transformation is *a*, it is better to heat the sample to a higher temperature than this, in order to cause the transformations to take place quickly, then to cool down to the temperature *a* and to then temper. By proceeding thus, the steel has ample time and opportunity to be transformed, the latter operation not being an instantaneous change at *a*, but a change which takes considerable time at that temperature but a shorter time at a higher temperature. Records of many series of tests are then given, using pieces of different sizes, heated at different rates and cooled at different rates in different menstrua. The following important conclusions are given in the words of the author himself:

HEATING.

A.—The transformation point of a steel varies in practice between two extremes, the maximum being the temperature of transformation, measured at the center of a very small sample heated rapidly to a high temperature; the minimum being that of the periphery of a large specimen heated very slowly to a lower temperature.

B.—(*a*) To heat a piece of steel rapidly to a given temperature it is advantageous to have continuously the greatest possible difference of temperature between the piece and its surroundings. This advantage is greater the larger the piece.

(*b*) The time required to heat a piece increases in much greater proportion than the size of the piece.

C.—Aside from the size of the piece being heated, and all other factors being constant, as the temperature of the heating bath increases.

(*a*) The maximum speed of heating increases proportionately.

(*b*) The transformation effect becomes evident sooner.

(*c*) The temperature of commencement of the transformation decreases, that of the middle point and end of the transformation increase.

(*d*) The range of temperature covered by the transformation increases.

(*e*) The total duration of the transformation period decreases.

(*f*) When the transformation is once provoked it will progress at the same temperature, and if the rate of heating is changed at this time, it appears to influence only the time necessary for the transformation.

D.—For a given temperature of the heating bath, at points at regular intervals between the outside and the center of the piece, as the center is approached.

(*a*) The maximum speed of heating decreases.

(*b*) The commencement of the transformation is retarded more and more.

(*c*) The temperature of beginning of transformation rises.

(*d*) The duration of the transformation increases.

(*e*) Between any two given points the difference between the temperatures of commencement and end of the transformation increases with the temperature of the bath.

E.—At homologous points of similar pieces the temperature of the bath being alike.

(*a*) The temperature of the transformation varies inversely as the size of the piece.

(*b*) The duration of the transformation period varies directly as the size of the piece.

(*c*) The duration of the transformation period varies inversely as the range of temperature embraced in that change, independently of the size of the piece.

(*d*) The product of a linear dimension of the piece by the maximum velocity of heating varies proportionately to the temperature of the bath.

COOLING.

A.—The transformation point during cooling of a steel varies between two extreme temperatures; the maximum is that of the transformation of a point on the periphery of a large specimen cooled very slowly; the minimum is that of the transformation of the center of a small specimen cooled very slowly.

B.—(*a*) The maximum temperature attained during the heating conditions the maximum possible rate of cooling, which is proportional to this temperature.

(*b*) All other factors being equal, the energy for tempering bath may be represented by the value of the ratio between the maximum rate of cooling of any two baths.

C.—The fall of temperature during tempering commences to produce effects in a time which is shorter as:

(*a*) The speed of cooling is increased.

(*b*) The smaller the specimen.

D.—The temperature of transformation when cooling depends both on the speed of the cooling and the duration of the heating after the transformation point was passed going upwards. All other things being equal the transformation point when cooling will be lower as:

(*a*) The speed of cooling is greater.

(*b*) The duration of heating has been prolonged.

(*c*) A given variation, the maximum speed of cooling displaces the transformation the more the shorter the time of heating.

E.—With the same tempering bath and at homologous points of similar pieces.

(*a*) The maximum speed of cooling decreases as the dimensions increase.

(*b*) The temperature of transformation increases with the size of the piece.

F.—All other things equal, the hardness at any point varies with the mean velocity of cooling at the period of transformation.

(*a*) This velocity varies in the same manner as the maximum velocity of cooling, but diminishes much more quickly.

(*b*) With a given tempering bath this velocity varies inversely as the size of the piece.

(*c*) For a given hardness, other factors being equal, the time during which a piece should be held at a given temperature varies inversely as the temperature.

G.—For a given heating temperature and tempering bath, at points equally spaced from periphery to center.

(*a*) The maximum speed of cooling decreases.

(*b*) The speed of cooling during the transformation falls still more rapidly.

(*c*) The transformation is more and more delayed.

(*d*) The temperature of transformation decreases.

(*e*) The period of transformation lasts longer.

Nickel-Alloy Steels.—The seventh report of the Alloys Research Committee of the (British) Institute of Mechanical Engineers deals with the effect of varying percentages of nickel in steels containing about 0.45 per cent carbon and 0.9 per cent manganese. The contents of carbon and manganese were maintained constant, while the nickel was varied. From the report, which is signed by Carpenter, Hadfield and Longmuir, the following points, among many others, are especially interesting. With the increase in nickel content up to 4 per cent, the change in properties of forged bars is gradual; the material, when under elastic stress yields more to the stress; at the same time, after the apparent yield point is passed, the maximum stress increases. Although the change in properties is gradual, there is in nearly every case a more or less pronounced kick in the curves between 0 and 4 per cent nickel. At some point between the percentages of 4.25 and 4.95 nickel, there is a very sudden change in nearly all of the properties, evidenced by a rapid increase of maximum stress, which reaches the highest value at 0.42 nickel, a fall of ductility, and an increase of brittleness, as shown by the bending, tension, torsion and shock tests. Thus, as far as industrial products are concerned, a danger limit for nickel content is found at 4.25 per cent, when carbon and manganese are present to the extent of 0.44 per cent and 0.88 per cent, respectively. The results of the tests of the cast material followed generally the same order as those of the forged material, the maximum tensile strength corresponding to a content of 7.95 per cent nickel. It is peculiar that this maximum is associated with an elongation of 4.5 per cent, and the three steels which in the forged condition are distinctly brittle, show in the cast-normalized state elongations of 6.2 per cent, 4.5 per cent and 6.2 per cent. Tests made at the temperature of liquid air show that these alloys are well adapted to be used at low temperatures. In every case the tenacity is greater at the low temperature than at ordinary temperatures. The alloy containing 20 per cent of nickel has at ordinary temperatures a tensile strength of 43.9 tons per square inch, which at the temperature of liquid air—182° C. rises to the remarkable figure of 157.2 tons, with an elongation of 15.5 per cent.

Nickel-Manganese Steels.—Guillet, in *Revue de Metallurgie* for November, discusses very instructively the limiting proportions of nickel, manganese and carbon found in these steels, and the essential nature of the alloys. Steels were studied with 0.15 per cent carbon, 5, 7 and 15 per cent of manganese, and 2, 12 and 30 per cent of nickel; with 0.75 carbon, and similar quantities of manganese and nickel. From an extended mechanical and micrographic study of these eighteen steels, together with some of other compositions outside those limits, the author concludes that there are three varieties of these steels, corresponding to the values of the following equations:

$$P = \frac{\text{per cent nickel}}{1.3} + \frac{\text{per cent manganese}}{6} + \frac{\text{per cent carbon}}{1.65}$$

$$P' = \frac{\text{per cent nickel}}{21} + \frac{\text{per cent manganese}}{13.5} + \frac{\text{per cent carbon}}{1.65}$$

If P is less than O, the steel is a perlitic steel.

If P is greater than O } the steel is martensitic.
and P' is less than O }

If P' is greater than O, the steel contains *gamma* iron.

The above explains the microscopic appearance and some of the consequent thermal and mechanical qualities. These martensitic steels can in many cases replace nickel steels, with the advantage of being much cheaper.

Chromium Steel.—The engineer for Jacob Holtzer & Co., Mr. A. Brustlein, relates in *Revue de Metallurgie* for July last, how he commenced to make chromium steel in 1874. By mixing powdered potassium bi-chromate, wood-charcoal, ground glass and fine steel turnings, and melting down in a crucible, he obtained ingots of ferro-chromium containing about 55 per cent of chromium. To make this more cheaply, recourse was had later to the natural chromite, which was pulverized, made into a soup with bran and powdered charcoal, and then boiled down to a dry mixture. The mass was mixed with more charcoal powder, ground glass and a little lime, ground to a uniform tint, and then charged into old crucibles which were unfit for further manufacture of steel. Giving these crucibles a very strong heat, which frequently softened their upper parts completely, there was obtained from each an ingot weighing 1 to 2 kilograms, containing about 55 per cent of chromium, and which cost about 2 francs (40 cents) per kilogram of contained chromium. Ferrochromium was thus made for a dozen years, but later the steel works of St. Chamond succeeded, at their works at Broucau, in making regularly a product rich in chromium in a cupola, at a lower price, and the manufacture in crucibles was abandoned.

Influence of Nitrogen on Iron and Steel.—Hjalmar Braune, in *Revue de Metallurgie*, Vol. II., No. 7, describes some very important results obtained by heating iron and steel to 800° in ammonia gas, for various lengths of time, and analyzing and making microscopic study of the specimens. Tests were made on soft iron with 0.06 per cent of carbon, medium steel with 0.5 per cent carbon, and hard steel with 1.15 per cent of carbon. The soft iron increased regularly in tenacity, but decreased in elongation, the results being:

Per Cent Nitrogen.	*Elongation*	*Tensile Strength (K. per sq. mm.)*
0.000	39.0	33.5
0.015	34.0	34.0
0.030	32.0	35.1
0.045	30.0	30.0
0.060	29.5	36.8
0.090	27.0	38.0
0.105	24.0	38.4
0.120	18.0	38.8

The effect on steel is still more marked, malleability disappearing when the contents of nitrogen are the following:

Steel with 0.20 per cent carbon	0.050 to 0.060 per cent.
Steel with 0.50 per cent carbon	0.040 to 0.045 "
Steel with 1.15 per cent carbon	0.030 to 0.035 "

The nitrogen seems to dissolve in the ferrite or pure iron

present, imparting to it a characteristic striated appearance under the microscope. The iron is saturated when it contains 0.20 per cent nitrogen, and then shows little rods or spikes on the polished section and is very brittle. However, commercial metal rarely contains 0.06 per cent, but amounts of 0.030 to 0.040 are frequent, and frequently account for the unexpected brittleness of high carbon steels. Direct contact with air does not favor the entrance of nitrogen into the steel, unless a basic slag and a reducing atmosphere are present. Such conditions are found in a blast furnace and during recarburization in a basic Bessemer converter.

The observations of Mr. Braune are clearly presented, and if confirmed by other reliable observers will do much to clear up some of the so-called *"mysteries"* of steel.

Electric Conductivity of Steel.—In view of the increasing use of the "third rail" as a means of conducting electric power to moving cars, the problem of the effect of chemical composition and physical treatment on the electric conductivity of steel is of importance. Some experiments made in this line by S. S. Wales are described in the December issue of the "Rose Technic." It appears that the electrical resistance increases with the total amount of dissolved impurities, or what is known as the "total content not Fe." Of all the elements contained in steel, manganese has the most powerful effect in increasing the electric resistance.

Strength of Steel at Different Temperatures.—The Swedish metallurgist, Brinell, has determined (*Iron and Steel Magazine*) the strength of acid and basic open-hearth steel at temperatures up to 1200° C. The chemical composition of the two specimens was nearly identical (carbon 0.17, silicon 0.014, manganese 0.35, sulphur 0.015, phosphorus 0.028 in acid steel and 0.009 in basic). The results were:

Temperature		*Strength in Kilos.*	*Strength in Lbs.*
C°	*F°*	*Per Sq. Millimeter.*	*Per Sq. Inch.*
0°	32	35.8	51,100
100°	212	35.8	51,100
200°	392	34.4	49,100
300°	572	42.2	60,300
400°	752	41.1	58,700
500°	932	30.5	50,400
600°	1112	26.9	38,400
700°	1292	13.8	19,700
800°	1472	9.8	14,000
900°	1652	8.4	13,400
1000°	1832	6.7	9,600
1100°	2012	4.6	6,600
1200°	2192	2.7	3,900

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

Electric Furnaces.

Electric Tube Furnace.—J. E. Ober, 812,801, Feb. 13 (assigned to General Electric Co.). Application filed July 15, 1904.

The essential feature is the covering of the heating tube of carbon with a uniform layer of titanium carbide to prevent deterioration of the tube. The construction is shown in Fig. 1, where 1 are massive end pieces of carbon or graphite, and 2 is the tube. The latter comprises two cylindrical portions 3 and an intermediate concentric cylindrical portion 4, the external diameter of which is less than that of the end portions. A uniform layer 5 of titanium carbide surrounds the portion 4. The whole is embedded in coke. As soon as current passes through the tube 2, the surrounding layer of powdered titanium carbide sinters together, to form a shell of considerable mechanical strength. This shell is a good conductor of electricity and is non-hygroscopic and is refractory.

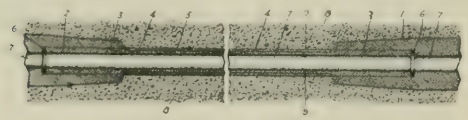


FIG. 1.—TUBE FURNACE.

Electrolytic Processes.

Platinum Plating with Alternating Current.—W. C. Arsem, 811,729, Feb. 6 (assigned to General Electric Co.). Application filed July 25, 1904.

In most cases electrodeposition of a metallic salt solution with an alternating current permits the deposition of the metal during one half cycle, as one electrode (which is a cathode) and the passing back into solution during the next half cycle from the same electrode (which is now anode). The total net result is therefore zero. But this will not be the case with certain metals which are not readily attacked by the acid radical of the plating bath. In such cases both electrodes will be plated with the metal. The inventor uses this application of alternating current for plating purposes, for depositing heavy coatings of platinum on copper, iron or other metallic plates. As electrolyte he uses a solution of ammonium chloroplatinate in sodium citrate.

Production of Hydrated Oxide of Lead.—J. H. Bridge and C. Ellis, 811,552, Feb. 6. Application filed March 14, 1904.

An electrolyte of sodium chloride of 10° Baume is used with lead as anode and an inert cathode of graphite, platinum or lead. The anode should be of a good quality of lead, quite free from iron; ordinary pig-lead is usually suitable. The electrolyte is heated to prevent the precipitation of lead chloride during electrolysis. An anodic current density of 20 amps. per square foot is used. The anodic reaction is the formation of lead chloride, which passes into the hot electrolyte and reacts with the sodium hydroxide formed at the cathode, giving hydrated lead oxide (which appears as a white precipitate on the bottom of the cell) and regenerating sodium chloride. The electrolyte is, therefore, continually regenerated.

Anode. L. Levett, 813,048, Feb. 20. Application filed May 9, 1905.

The object is to provide an anode for electroplating which may be entirely submerged in the electrolyte, and which will always present a smooth surface, so that any oxide which forms on it may be readily scraped off. The construction is shown in Fig. 2, both in perspective view and in vertical cross-section. The anode consists of an in-

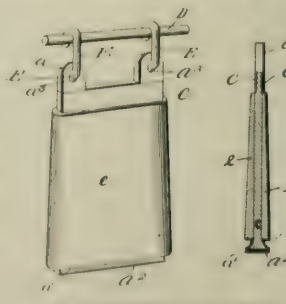


FIG. 2.—ELECTROPLATING ANODE.

ner carbon plate or core *c*, with the ears *aa* for suspension, and an outer sheath *e* of metal which may be readily slipped on or off the carbon plate. The lower enlarged portion of the carbon plate is covered with insulation *d*."

Wired Plating and Dipping Rack.—J. P. Clark, 811,375, Jan 30. Application filed June 26, 1905.

To a vertical wooden center bar a series of transverse wooden bars are rigidly secured. To each of these transverse bars are attached a series of metallic wire hooks, from which the articles to be plated are suspended. These wire hooks are electrically connected to wires placed in longitudinal grooves in the transverse bars, and these wires are again in electrical connection with a wire placed in a groove in the vertical center bar and connected with the outside terminal.

Electrolysis of Water.—W. F. M. McCarty, 813,105, Feb. 20. Application filed Nov. 8, 1904.

The object is "to render the water, which is normally not an electrolyte (!), more susceptible to the action of an electric current by adding to the water a substance which will disturb or break up the molecular arrangement of the hydrogen and oxygen of the water, and thus facilitate its ready decomposition." For this purpose the inventor adds metallic sodium (!), resulting in formation of sodium hydroxide and evolution of hydrogen. "At the moment of ebullition" he supplies current to the electrodes.

Batteries.

Storage Battery.—William Fennell and William P. Perry, 810,929, Jan. 30. Application filed March 17, 1904.

The third claim reads: "In electric accumulators a honeycombed mass, an active material on the exposed surfaces of the mass, and unobstructed passageways formed therein, besides the interstices of the mass, certain of said passageways being inclined for the egress of gas."

Storage Battery.—William Fennell and William P. Perry, 810,930, Jan 30. Application filed March 6, 1905.

A hard, porous, non-conducting substance, insoluble in the electrolyte, is used in the shape of a network, honeycomb or framework, so as to contain the electrolyte in the plate itself, and at the same time to act as a support for the active material which is spread over the surface in the form of a thin film. Pumice is the most effective material for the honeycomb, but earthenware or other burnt clay, asbestos, slate, soapstone or the like may also be used. One special construction consists of a number of rough burnt-clay rods of irregular shape, each covered with a film of active matter, and held together in a bundle by india rubber rings, yarn or the like. A piece of metallic wire is placed in the center of the bundle to serve as conductor.

End-Celf Switch.—L. Lyndon, 810,958, Jan. 30 (assigned to General Storage Battery Co.). Application filed Oct. 17, 1904.

The object is to make the operation of the end-cell switch automatic. Thus the end-cell-charging switch will automatically set the contact brush of the switch upon such a cell contact that all the cells on one side of the contact are fully charged, while all on the other side have not yet been brought to full charge. The switch-actuating device is of electrical nature, and is controlled by the voltage of one or more of the end-cells, as is described in great detail in the specification.

Primary Battery.—C. E. Lockwood and G. A. Lutz, 812,504, Feb. 13. Application filed July 13, 1904.

Claim 1 reads: "A battery element comprising a perforated retainer containing a depolarizer and having a binder soluble in a battery liquid or solution for preventing such depolarizer from passing through the perforations in the retainer." The object is to replace a retainer when its depolarizer has been used up by a new retainer with fresh depolarizer. The arrangement appears to be intended specially for use of cupric oxide as depolarizer in an alkaline cell, with sugar or starch saccharine matter as binder.

Primary Battery.—C. E. Lockwood and G. A. Lutz, 812,505. Feb. 13. Application filed Jul. 13, 1904.

Compare the preceding patent. Claim 1 reads: "A battery element comprising a perforated retainer containing a depolarizer and having a binder soluble in a battery liquid or solution applied to the wall of the retainer closing the perforations therein to prevent such depolarizer from passing through the perforations in the retainer, said binder being of a character to not materially affect the electrical efficiency of the battery."

Binding Post.—E. C. Henn, 813,093, Feb. 20. Application filed May 9, 1905.

Details of construction of a binding post of a galvanic cell. Claim 1 reads: "An element for an electric battery-cell consisting of a rod of non-metallic substance having a bore in its outer end portion, a binding-post comprising conductor-attaching means at one end, and having at its other end portion longitudinally-extending ribs, said ribbed portion of the binding-post being inserted in said bore in the rod, the bore being of a diameter less than that of the engaging portion of the binding-post, whereby the ribs will have a retaining engagement with the wall of the bore."

Miscellaneous.

Ozone Generator.—C. F. Birtman, 811,364, Jan. 30. Application filed June 9, 1905.

Mechanical details of construction of a chamber for producing ozone by silent discharges through air. The chamber contains vertical glass plates which act as dielectrics and form passages for the flow of air. On each side of such a glass plate there are two other glass plates, around which coils of wire are wound. These coils are alternately connected to the positive and to the negative terminal of the electrical supply

RECENT METALLURGICAL PATENTS.

Iron and Steel.

Reduction of Iron Ore with Water-Gas.—G. M. Westman (812,247, Feb. 13) emphasizes that if iron ore is to be reduced by gases containing carbon monoxide and hydrogen, the gas should be free from oxidizing agents (like carbonic acid), and should be used at a temperature of about 1,100° C. The gas is produced as follows: Water is sprayed into a current of air, which is afterward raised to a temperature which must be at least high enough to convert the water into steam, and should be in practice "as high as possible." It is desirable to introduce into the air as large a proportion of water as possible; but the proportion of air must be high enough to maintain the gases at a proper reducing temperature, which for iron ore is 1,100° C. The mixture of air and steam is then brought into contact with a mass of glowing coke, which yields hydrogen and carbon monoxide. The gas is kept long enough in contact with the glowing coke until any carbonic acid which may be formed takes up enough carbon to form carbon monoxide. The gas is then introduced into the reducing furnace containing the ore mixed with a small quantity of solid carbon, the ore and carbon being preferably formed into briquets. The use of this small percentage of carbon in reducing the ore is to practically remove all traces of oxygen from the ore. But the amount of carbon must be small, only about 2 or 3 per cent. because the object of the process is to produce wrought iron. If the producer gas contains sulphur (from the coke) this is removed by mixing limestone with the coke in the chambers of the gas producer. The process "can be carried on at a low cost and is of high efficiency. Furthermore, ore which cannot be reduced in ordinary blast furnaces, because it contains

titanic acid, can be reduced by this process. This process can also be advantageously used in reducing ores containing phosphoric acid. In blast furnaces, on account of the high temperature, the phosphoric acid is converted into phosphorus, which enters the iron. At the comparatively low temperatures employed in this process, however, the phosphoric acid is only slightly reduced, so that the resulting iron contains but little phosphorus."

Manufacturing Iron Sponge.—G. Gröndal (812,174, Feb. 6) patents a furnace for the production of iron sponge from pulverized iron ore and coal, the necessary heat being obtained by feeding a mixture of combustible gas and air in a number of streams repeatedly through the charge. For this purpose the furnace is provided with a large number of tubes arranged in zigzag across the shaft. To cool the produced iron sponge, the inventor uses either the combustible gas or the atmospheric air, which, when introduced into the furnace, is forced through passages to and fro until the iron sponge is sufficiently cooled to be discharged in a closed chamber connected with a supply of an indifferent or reducing gas, which is absorbed by the iron sponge, so that the latter is thereby rendered indifferent to the oxidizing action of the air.

Titanium Steel.—F. E. Canda (813,278, Feb. 20) calls attention to the difficulty of introducing titanium into steel. He has found that when ferro-titanium, containing 20 or more per cent of titanium, is introduced into molten steel, it is not melted and does not combine appreciably with the steel, but remains substantially unchanged. The difficulty is due to the fact that the temperatures available in ordinary metallurgical furnaces are below the melting point of high percentage ferro-titanium. Canda, therefore, melts the ferro-titanium in a small special electric furnace and mixes the molten alloy with the molten steel while the latter is run into the ladle.

Zinc

Charging Machine.—In connection with the report of the visit of the American Institute of Mining Engineers to the Hazard works of the New Jersey Zinc Co., an account of which will be found elsewhere in this issue, three patents,

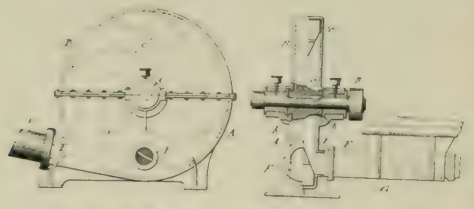


FIG. 1.—CHARGING MACHINE FOR ZINC RETORTS

which have just been granted to A. L. J. Queneau (813,021, 813,022, 813,023, Feb. 20) are interesting. The first two patents refer to the method and apparatus for charging a retort or muffle. The construction of it may bee seen from Fig. 1, showing a side elevation and front elevation. The charge is fed from a supply-hopper, and by means of a feed-screw *G* through the feed-inlet *I*, into the casing *A*. Upon the shaft *C* in the casing is kept the impeller disc, consisting of a single casting having a hub-portion *a* with end flanges *b*, a flat main body portion, constituting a circular plate of a diameter corresponding to the internal diameter of the casing, and a series of blades *c* of a curvature indicated in the left-hand diagram. At their outer ends these blades are of a width corresponding to the width of the casing, and thus continue of this width to a radial distance corresponding to the lower portion of the feed-inlet *I*, from which point they decrease in width until they merge into the surface of the disc. The charge forced into the casing through *I* will, therefore, receive the full impact blow of the extreme outer portion of the blade approaching at that particular instant. This impact blow is sufficient to give the charge a very considerable momentum and force it through the outlet spot *F* in a substantially cylindrical stream, which retains its shape and form without material deviation for a distance varying from 3 to 10 feet and more according to the velocity of movement of the impact blades. This practically solid stream may, therefore, be directed with great accuracy into the muffle. Patent 813,023 refers to a charging machine for charging a bench of retorts made up of horizontal and vertical rows. In present practice the retorts are charged by hand, and although the workingmen receive the wages of high-skilled labor the charging is often uneven and uncertain.

Roasting Furnace.—Frank Klepetko (811,643, Feb. 6) patents a modification of his system of cooling the rabble-shaft and arms in a McDougall roasting furnace. The object is to constrain the circulating cooling medium to remain in contact with the walls of the rabble-shaft for a maximum length of time. For this purpose the rabble-shaft is divided into two longitudinal contiguous intercommunicating conduits, the cooling medium passing through the shaft in one conduit in one direction and through the other conduit in the opposite direction.

Tin.

Production of Stannic Chloride Solutions.—Like other electrolytic alkali and bleaching powder plants, the Acker Process Co., of Niagara Falls, has made for several years extended experiments with a view of utilizing their chlorine to better advantage than is possible in the manufacture of bleaching powder at the present market price. Chas. E. Acker (810,897, Jan. 30) proposes to prepare stannic chloride solutions as follows: Certain tin salts, in which the tin is in the tetravalent state, are extremely energetic solvents for metallic tin, whereby the solvent liquid is reduced to the divalent state and looses its solvent power for tin. By bringing the liquid into contact with chlorine gas, it is raised again to the tetravalent condition, with corresponding restoration of its solvent power. This regenerated solvent is again brought into contact with metallic tin and again regenerated, the operation proceeding in this manner until the required concentration is reached. In this way stannic chloride solutions of high concentration and uncontaminated by impurities of any kind may be produced. This is one process of Acker. A second process starts with a stannic chloride solution produced in this way and having a specific gravity of at least 1.8. If this is converted to a stannous chloride solution by contact with metallic tin, the stannous chloride will separate out in form of crystals of a high degree of purity, and if the stannous chloride solution is formed at a temperature of about 50° or 60° C., a large yield of the crystals will be obtained when the solution is permitted to cool. Acker proposes not to use pure electrolytic chlorine gas, but to mix it with a large proportion of air (even as much as 95 to 96 per cent of air), and to inject steam into the chlorine for the production of hydrochloric acid. Since the reaction evolves considerable heat, a portion of the heat is absorbed in raising the temperature of the excess of air, and it is, therefore, possible to utilize larger absolute quantities of chlorine in a given time. The addition of hydrochloric acid results in the acceleration of the solvent action of stannic chloride upon metallic tin and also prevents the formation of oxychlorides of tin. The patent above referred to describes apparatus for carrying out these processes.

Aluminum

Soldering Aluminium.—A. W. King (811,725, Feb. 6) patents a special solder by means of which it is possible to "unite aluminium coated therewith to either aluminium similarly 'tinned' or to another metal coated with ordinary solder, the connection between the two coated surfaces in either case being then practicable by means of ordinary commercial soft solder." The preparation of the special solder is as follows: Soft

solder, composed of one part of tin and two parts of lead, is melted together with zinc and aluminium in the following proportions by weight: Ten parts of soft solder, five of pure zinc and five to eight of aluminium. This mixture is melted in an iron pot to a low red-heat, and then poured into a mold. For soldering aluminium to aluminium the surfaces are first cleaned by scraping and heated. The special solder is then rubbed thereon until it adheres, and the coated surfaces are then cleaned. The two pieces can then be united by using ordinary soft solder, no spirits or resin being required to enable the ordinary solder to adhere to the above special solder. To solder any other metal, f. i. brass, to aluminium, the aluminium piece is coated with the special solder, as above described, while the brass piece is coated in the usual way with soft solder and either hydrochloric acid, neutralized with zinc or resin, and then carefully cleaned and washed. The two pieces are then soldered by ordinary commercial soft solder with application of heat. The special solder, when prepared according to the above recipe, must be kept clean and free from grease, and must not come in contact with mercury.

BOOK REVIEW.

SECOND-YEAR CHEMISTRY. By Edward Hart, Easton, Pa.: The Chemical Publishing Co. Price $1.25.

Prof. Hart has given the student of chemistry an excellent aid in his recent volume, although it seems to us that good manuals of qualitative analysis are already sufficiently numerous. This book is to be praised because it co-ordinates some of the methods of quantitative work with second-year work in the college laboratory. Such is undoubtedly the proper method of instruction.

A Low-Resistance Thermo-Electric Pyrometer.

At the reunion of the American Society of Mechanical Engineers, held on Jan. 30, Prof. William H. Bristol, of the Stevens Institute of Technology, described and exhibited his new and very interesting thermoelectric pyrometer, suitable for temperatures not to exceed 2,000° F.

The pyrometer is of the Le Chatelier type, but differs from the instrument of the French scientist in the nature of the alloys employed for the couple and in several mechanical details which were devised to increase the convenience of the use of the instrument for practical use. With the platinum and platinum-rhodium couple of La Chatelier an extremely delicate high-resistance galvanometer is practically indispensable. Le Chatelier considers 200 ohms as the minimum resistance allowable in the indicating instrument. This amount of resistance is necessary to eliminate the atmospheric temperature influence upon the whole resistance. Contrarywise with the Bristol pyrometer a low-resistance instrument is used. The British pyrometer consists of three parts, the couple, the indicator and the leads, connecting couple and indicator. The leads between the couple and the indicator may be of almost any desired length to meet the special requirements; the combined resistance of the leads, couples and indicator are fixed to suit the total range of the instrument, and varies from 3 ohms as a minimum to 10 as a maximum. The indicator is a low-resistance instrument of special design, and is made especially for this purpose by the Western Electrical Instrument Co.

FIG. 1.—SWITCHBOARD TYPE OF PYROMETER.

Fig. 1 shows a wall or switchboard form of the pyrometer. Fig. 2 illustrates a portable form.

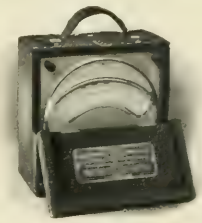


FIG. 2.—PORTABLE TYPE OF PYROMETER.

Prof. Bristol made a long series of experiments with different metals and alloys so as to find a couple which would withstand a high temperature and at the same time give a high electromotive force. The couples finally adopted in him consists of alloys of tungsten, steel, nickel, iron and copper, different alloys being employed to suit the total ranges of temperature that it is desired to have the scale cover.

Since no rare metals are used for these couples this part of the pyrometer is inexpensive, and it is possible to employ elements of large cross-section which will not be affected in their resistance any appreciable amount by the variation of temperature along the lengths of the elements forming the couple.

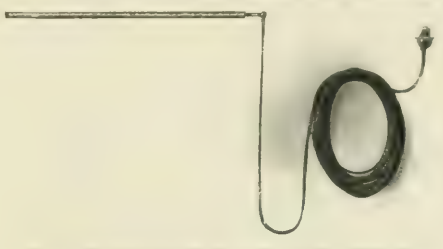


FIG. 3. THERMO-ELEMENT, JOINT, LEADS AND PLUG.

The leads to the indicator are made of flexible insulated copper duplex cable and of ample cross-section to practically eliminate the influence of variations of atmospheric temperature.

The cross-section of the elements of the couple is reduced at their junction, thus rendering it sensitive to sudden changes of the temperature to be measured.

FIG. 4. DETAILS OF JOINT.

A novel feature of the couple is that it is made separable at the point where it passes through the wall of the space within which the temperature is to be measured.

The object of the joint is two-fold: first, to make it possible to renew the "fire-end" whenever it may be necessary; and, secondly, to permit carrying the cold ends of the elements to a point toward the floor, where the atmospheric temperature will be constant and not influenced by the temperature that is being measured.

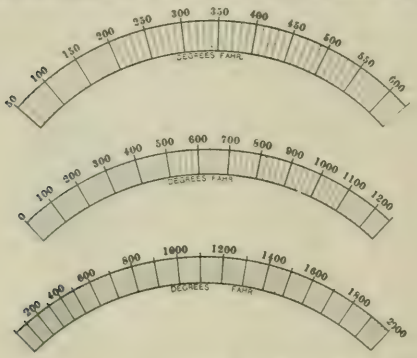
Fig. 3 shows a complete element with separable joint, coiled leads and lamp plug on the end of the cable forming the leads for convenient connection to the indicating instrument shown in Fig. 1.

A special feature of the joint is that it is provided with large bearing surfaces to prevent possibilities of variations of resistance at the connection, and is constructed to allow for easily breaking and making the connection. The details of the joint are shown clearly in Fig. 4, and from this it will be seen that it is impossible to make connection incorrectly, as is usually done when there is nothing to guard against it.

The low cost of the couples makes it practicable in the use of the instrument to keep an extra fire-end in reserve, which may at any time be quickly substituted for the one that has been in continual service, thus affording an economical and positive check upon the accuracy of the instrument.

The elements of the couple are independently insulated in a novel and effective manner by winding each with asbestos cord and then coating the surface with carborundum paint, a solution of silicate of soda being used as a binder. This makes a clean, compact and smooth insulation.

Couples thus insulated are flexible, and can either be applied to the heated space directly or they may be insulated in a piece of ordinary pipe as a protection. This protection has proven itself to be effective and economical. For continuous applica-



FIGS. 5, 6 & 7.—SCALES OF INSTRUMENT.

tions of the couples to temperatures in the neighborhood of 2,000° F., or over special protecting tubes of nickel, plumbago or porcelain are employed.

Figs. 5, 6 and 7 show the scales up to 600°, 1,200° and 2,000° F. For very open scales over shorter ranges several couples may be placed in series, thus making it possible to read to small fractions of a degree.

A novel application of the thermo-electric couple is that of determining the temperature of molten metals, as cast iron, copper, brass and bronze. It consists in leaving the ends of the elements disconnected and without insulation.

When these ends are slightly immersed in the molten metal it makes a junction between the elements, and the reading will be the same as if the elements of the couple had been originally joined. The advantage of this plan is that the reduced cross-section at the ends of the couple allows it to almost instantaneously attain the temperature of the molten metal, and consequently there is no time lag or error, a most important advantage. As the couple is used over and over again, the ends become worn away, but the couple is nevertheless, always ready for use by inserting at a fresh portion, which will not be changed to any appreciable extent by continued use, and will give the same reading for a given temperature as if the couple had not been worn away. A joint is provided near the end that is exposed so that a fresh tip can be applied to the couple before enough of the end has worn away to appreciably affect the temperature of the protecting system.

Compensators.

For several reasons it is necessary to allow for changes in temperature at the cold end of the couple unless some means is provided to maintain it at a constant temperature.

This is done by Prof. Bristol in a most ingenious and simple way by the compensating device shown in Fig. 8. It consists of a glass bulb with a short stem, similar to an ordinary mercurial thermometer. Two platinum terminal wires are fused into the stem near its top. These are connected within the bore of the stem by a loop of fine platinum wire, thus completing the circuit as indicated in the diagram.

The size of the bulb, the cross-section of the bore of the stem and the cross-section of platinum wire loop are proportioned to suit the case in hand.

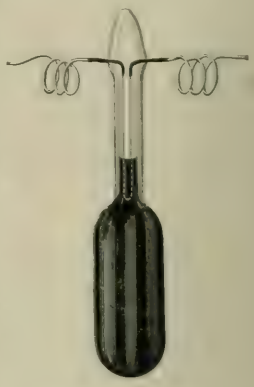


FIG. 8.—COMPENSATOR.

The compensator will perfectly compensate for any particular point on the scale, as, for instance, the working point where it may be desired that the reading shall be absolutely independent of changes of temperature at the cold ends. It will readily be seen that if the temperature rises at the cold end the mercury rising in the stem will short-circuit a certain portion of the platinum loop, thus reducing the resistance of the entire circuit by exactly the necessary amount, so that the diminished electromotive force of the couple, due to the rise of temperature of the cold end, will send the same amount of current through the circuit and instrument, and consequently give the same reading as if there had been no change of temperature at the cold end.

Fig. 9 is a diagrammatic sketch showing the compensator connected in circuit.

In summing up the features of his new pyrometer, Prof. Bristol emphasized the following chief advantages:

First, the commercial switchboard type of portable dead-

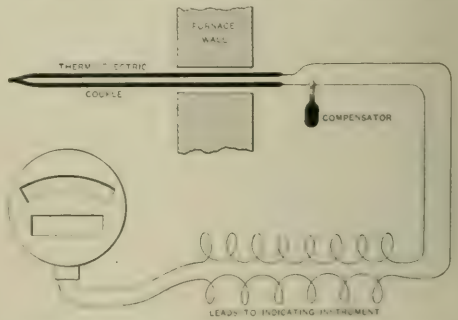


FIG. 9.—DIAGRAM OF CONNECTIONS WITH COMPENSATOR IN CIRCUIT.

beat indicating instrument may be employed, instead of the extremely delicate suspension galvanometer required for use with a single platinum-rhodium couple. This advantage is gained by the fact already stated, that the thermo-electric couples employed give several times as much electromotive force as the platinum-rhodium couples, which is ample to successfully operate a pivot instrument if of sufficiently low resistance.

Secondly, it affords a practical method for automatically

compensating for the changes of temperature at the cold ends of the couple, as already described.

Thirdly, it makes it practicable to use the same indicating instrument and the same couple for different total ranges of temperature by using different binding posts and having several scales drawn, the proper resistances being inserted for each individual total scale.

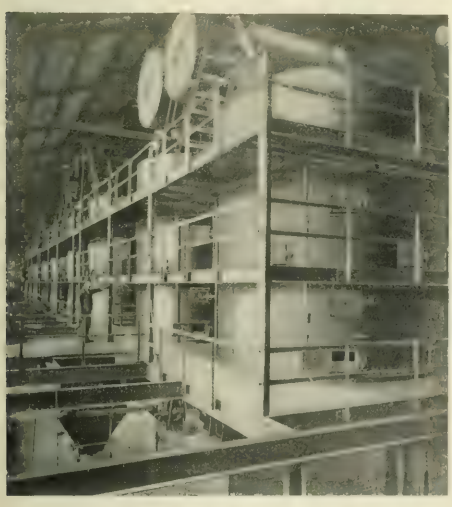
This new pyrometer is made by William H. Bristol, 41 Dey Street, New York City.

A New 1000-Ton Concentrating, Roasting and Smelting Plant.

The Nevada Consolidated Copper Co., San Francisco, whose properties are located near Ely, Nev., where a new 1,000-ton concentrating, roasting and smelting plant is in process of construction, has recently let the contracts for the entire equipment, including power and electrical apparatus, to the Allis-Chalmers Co., of Milwaukee.

The new works, when completed, will constitute one of the most carefully equipped and thoroughly modern plants in the United States. Up to the present time the work on the company's properties has been largely that of development, awaiting the completion of a plant with adequate capacity to handle the output. The process of treatment will consist in the concentration of ores, which concentrates will be passed to the McDougal roasters, and thence to the reverberatory furnaces and converters in the usual manner.

In laying out the plant the arrangement of equipment of various kinds follows closely the process referred to. It is divided into departments under the following heads: 1. Concentrating machinery. 2. Roasting machinery. 3. Reverberatory smelting machinery. 4. Converter machinery.



ROASTING PLANT OF ANACONDA COPPER CO.

Under these divisions the ore-treating machinery, much of which is of special and unusually heavy design, aggregating 2,800,000 pounds in weight, will be installed.

One of the features of the new plant's equipment will be the battery of six McDougal roasting furnaces, each standing 19 feet, 6 inches high and 18 feet in diameter. The main shafting for the entire group of six furnaces will be driven from a 20-horse-power Bullock induction motor, running at a speed of 850 r. p. m. The operator will be enabled to control the operation of each furnace from [illegible] stages about them.

The McDougal roasting furnaces are [illegible] struction shown in the adjoining illustration, which is given here on account of its inherent interest, although it has no further connection with the plant of the Nevada Consolidated Co., The illustration is an interior view of the Anaconda Copper Mfg. Co.'s roasting plant—the largest in the world—where fifty-six McDougal furnaces are installed.

The list of concentrating machinery to be installed at the Ely plant will include, among other appliances, six [illegible] Huntington mills, specially heavy in design, of the Anaconda type. They will be equipped with special step-box and continuous shaft for three mills, dividing the six mills into two groups, also two heavy-design Blake crushers, eight Overstrom tables, ninety-one vanners, and twenty-four "Richards" classifiers, designed under the direction of the inventor, Prof. R. H. Richards, of the Massachusetts Institute of Technology.

The engine equipment consists of two 22-inch and 48-inch x 48-inch compound condensing heavy duty type, a blowing engine, with cylinders 10 inches x 32 inches, to deliver [illegible] cubic feet of free air per minute, and electrical machinery as follows: Two 800-kw. Allis-Chalmers "Bullock" engine-type alternators, two exciters and seven motors, ranging from 20 to 100 horse-power for the operation of the various ore reducing devices, furnaces, etc.

Mr. Thomas W. Cox is engineer of the company. It is expected that the new plant will be ready to begin operations by early summer.

Water Gas in the Manufacture of Incandescent Lamps and Storage Batteries.

Water gas has been used for several years with an enormous economy as compared to illuminating gas in European lamp factories for glass-blowing and carbonizing filaments and in storage battery factories for lead soldering. The most prominent European manufacturers of lamps and storage batteries use the latest German type of water-gas producers, by which a far greater yield of gas is claimed to be obtained than with the water-gas plants that are used in this country, which are similar to the older German type.

Water gas is made by alternately blowing steam and air through glowing coke. By blowing air through the coke (hot blowing period) the latter gets sufficiently hot so as to decompose the steam during the gas making period.

In the decomposition of steam hydrogen is liberated, the oxygen forming carbon-monoxide with the carbon of the coke. Such a mixture of equal volumes of hydrogen and carbon-monoxide with a small amount of impurities is the commercial water gas. The decomposition of steam is absorbing heat, so that after a certain length of time a cooling off of the coke is taking place. The coke is now not hot enough to decompose the steam entirely, and at last the steam will go through the fuel undecomposed.

The object of a good and efficient water gas process is to increase the time of gas making to a maximum, and to decrease the time of hot blowing to a minimum.

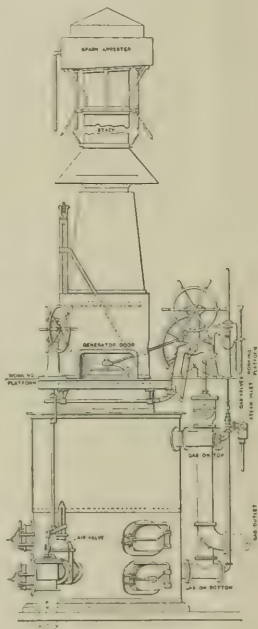
In the producers in use in this country and in the older German processes a high layer of fuel is used in the producers and a low air pressure is applied during hot blowing. The best results obtained by these systems was a gas making of 4 to 5 minutes. After 4 to 5 minutes of gas making the heat of the coke decreases to such an extent that gas making has to be stopped and "hot blowing" done for 4 or 5 minutes, after which hot blowing the gas making is started again.

The product of combustion of the hot blowing period is low grade producer gas which has to go to waste in most places, and can be utilized only under certain local conditions. The problem to be solved by the water gas engineers was

generate, during the hot blowing, a gas practically free of carbon-monoxide and high in carbon-dioxide, as by burning the coal to carbon-dioxide four times as much heat is generated as by burning same to monoxide. By burning to dioxide, in a certain time, four times the heat is generated as by burning to monoxide, and the hot blowing reduced to one-fourth or less. This has been successfully accomplished in Germany by using a low layer of fuel and high-pressure blast, and there are now over 200 plants working in Europe successfully for the last five years.

In these plants gas is made for 8 minutes and air blown for 1 minute. It is apparent that with such a producer of certain size the yield is largely increased as compared with the old type, and is in fact 35 to 40 cubic feet of gas of 275 to 285 B. T. U. per pound of coke.

By decreasing the period of hot blowing, as above stated, only a small holder is required so as to hold sufficient gas to help over the hot-blowing period, while in the older systems a very large holder had to be used, the cost of which, as a rule, amounted to more than the cost of the water-gas plant proper.



WATER-GAS GENERATOR.

Since the development of this system water-gas plants have been erected and are in successful use not only for electrical industries, but also for welding, annealing, brazing, forging, chemical factories, etc., in fact, wherever high temperatures or good regulation of same are required.

The working of the water-gas generator is as follows: The generator, which is a cylindrical shell, lined with fire-brick and provided with the necessary valves for entrance of air, etc., is filled with coke to a certain height, and blown hot by means of a positive blower. During this period the air inlet valve and the charging door leading to the flue are kept open; the gas valves are closed. The air enters under the grate and goes through the fuel to the stack. When the generator is sufficiently hot the air inlet and the charging door are closed, and at the same time the upper and lower gas valves are opened, depending where the gas is to leave the generator. Then steam is blown into the fuel until the gas gets inferior, which can be seen from a test flame.

When the quality of the gas deteriorates the steam valve is closed, the air valve and charging door are opened, the gas outlet closed and the blower started. This way air and steam is blown in from the top and from the bottom, for making the distribution of the heat in the generator more uniform. Fuel is charged through the charging door after three times of gas making. The generator is provided with tight doors, through which clinkers and ashes are removed.

The air and gas valve are interlocked, so that they cannot be opened at the same time, when the air valve is opened the generator is locked and vice versa. This insures absolute safety. The generator and all the valves are worked from the platform.

The water gas, after leaving the generator, goes to the scrubber, which is filled with coke and sprinkled with water; from here the gas goes to a small holder, which serves as an equalizer.

In this country such water-gas plants are built by Dr. Oskar Nagel, 90 Wall Street, New York City.

Notes.

Bethlehem Steel Co.—The Bethlehem Steel Corporation now employs 11,000 men. It is proposed to devote the proceeds of the sale of $12,000,000 of bonds to enlargement of the plant to a capacity that will employ double the number of men now on the pay roll. A year and a half is the estimated period within which the new work will be completed.

Metallurgie.—Prof. W. Borchers' well-known journal entitled *Metallurgie*, which is now in its third year, and which in the first two volumes dealt essentially with metals other than iron, has now, beginning with the third volume, extended its scope so as to include iron and steel. The editors are now Prof. W. Borchers, for all metals other than iron, and Prof. F. Wuest for iron and steel.

Protective Paint.—According to recent reports the Carbondale Machine Co. are successfully applying crysolite for painting brine and ammonia pipes, steam and ammonia condensers, brine and water tanks. It is said to meet in every respect the purpose of an elastic and strongly adhesive paint which will not become loosened when exposed to extreme ranges in temperature or to the action of acids or alkalies. Fuerst Bros. & Co., of New York, are distributing agents for the manufacturers, the Semet-Solvay Co.

Battery Exhibit.—At the Chicago Electrical Trades Exhibition, held from Jan. 15 to 27, the General Storage Battery Co. showed, in addition to a full line of Bijur "high-duty" plates and batteries, a working booster system handling violently fluctuating loads while maintaining a steady load on the generator. This is accomplished by a Bijur automatic regulator working on a new principle.

Electric Power in the South.—The Westinghouse Electric & Mfg. Co. have recently closed a contract with the Southern Power Co., at Charlotte, N. C., for a large and complete power equipment, consisting of eight 3,000-kw. water-wheel type alternators, with a complement of transformers, exciters and switchboards. This is the largest contract for electrical apparatus placed in the South for some years. The Southern Power Co.'s plant being situated in the midst of the cotton mill district will open up opportunities for the use of electric drive by the surrounding cotton mills and other industrial plants, which have heretofore been impossible, because of the lack of cheap electric power in that vicinity. The Southern Power Co. has full assurance that the cotton mills and other industries in the adjoining territory will make liberal use of the cheap power thus placed at their disposal, it being recognized that electric drive not only increases both the reliability and efficiency of a plant, but at the same time decreases the cost of production, and insures a better quality and a larger output of material.

Water-Wheels.—We have received from the Abner Doble Co., of San Francisco, their new catalogue entitled "Doble Tangential Water-Wheels." The first portion contains a general description of the essential features and parts entering into the construction of the water wheels built by the firm, special attention being paid to the Doble needle regulating nozzles, ellipsoidal buckets, ring oiling bearing, nickel steel shafts, etc. Illustrations of several of their different types of water-wheels are followed by descriptions of some of the typical hydroelectric power plants in which Doble water-wheels are operating. Some of these are very interesting,

thus the 8,000-hp. wheels which have been constructed by this firm for the de Sabla and Electra power houses and the 9,000-hp. wheel now being built. The Cornell plant is interesting on account of the high efficiency, 84.5 per cent. obtained on the Doble wheels. The latter portion of the book contains water-wheel tables covering all the conditions of water-power up to a head of 2,500 feet, and in capacities up to 5,000 hp. Other tables relate to the loss of head in pipe friction, riveted steel pipe, etc. The last seven pages are devoted to a series of ready conversion factors, most of which have been compiled from a recent book of Mr. Carl Hering. On account of the large amount of information contained in it the catalogue really represents a handy hydraulic engineering reference book.

Chemistry and Rats.—On the editorial pages of a recent issue of the London *Electrician* the following note appears: "It has sometimes been said that we have too much experiment now, and that accumulating piles and piles of undigested facts does not help science much. But there are exceptions which are quite outside any carping criticism. Who would deny the utility of this formula: $\text{Log } N = \frac{1}{4}[0.233 + (3 \times \log B.W.)]$?

It is hardly necessary to add that B. W. is the body weight of a white rat, and N is the total nitrogen in milligrammes per diem excreted by white rats differing in age and weight—and, perhaps, in religious opinions. All the information, but the last item, is in an abstract to be found in the November *Journal* of the (British) Chemical Society, p. 740. We have not seen the original paper; doubtless it will be out of print by this time. The compilers of technological pocket-books will have swooped down upon it and bought all the available copies."

Engines.—The Allis-Chalmers Co. has in preparation for its power department bulletin No. 1504, on heavy duty engines. Although the heavy-duty horizontal engines, manufactured by Allis-Chalmers Co., may be seen in operation in so many public and private power plants that few who desire to investigate their construction need have recourse to a descriptive bulletin, yet this is now to be issued for the convenience of prospective customers, and it will naturally embody the latest features of design. The bulletin is one which will appeal not only to engineers in charge of power plants, but also to the owners of such, who may not have much information concerning technical details of operation, yet can understand "results."

Tube Mills for Wet Crushing.—The Allis-Chalmers Co. has in preparation for its mining and crushing machinery department a new bulletin (No. 1,410) on tube mills for wet crushing in mining works. This is a publication which should be of considerable value to mine owners, particularly those who have properties yielding low-grade ore, for it gives the results of tests made by some of the leading companies in the South African fields to determine how far they can benefit by the introduction of tube mills as fine pulverizers working upon the product of their stamp batteries. The figures given are not only interesting but will probably surprise many mine owners in this country who have not paid much attention to the possibilities of tube mills in ore reduction work. Tests at some of the mines on the South African Rand showed, for instance, that a yearly increase in profit of from 15 to 40 per cent could be effected by the use of tube mills as auxiliary crushers. The data given in the bulletin may be regarded as absolutely authentic, inasmuch as they have been taken from official reports made at directors' meetings of the mining companies referred to. We have repeatedly referred in our columns to the introduction of the tube mill into practical gold metallurgy.

Belted Alternators.—The Allis-Chalmers Co. will issue, shortly, for its electrical department, bulletin No. 1,051, on alternating-current generators of the belted type. While direct-connected alternators have largely displaced belted machines in modern power stations, there are many places, particularly in comparatively small [illegible] units are desired. Belted machines are easily connected to the existing source of power in, for example, a [illegible] driving other machinery, and for those installations of [illegible] size they are lower in first cost than [illegible] apparatus. The Bullock machines described in this bulletin are adapted to a wide range of output.

Machinery for Cement Making.—When at the recent Bethlehem meeting of the American Institute of Mining Engineers an excursion was made to the Atlas Portland Cement Co., and later to the Lehigh Portland Cement Co., those visitors who were not familiar with the advances of the cement industry of the Lehigh Valley during the last fifteen years, were struck by the enormity of the plants and the skill in design of the machinery employed in the various steps of cement making. It is now recognized that the phenomenal progress in the Portland cement manufacture set in when it was learned how to turn out a uniform product. Of course, this knowledge came hand in hand with the improvement in machinery. The visit to the above named two plants was also instructive in showing quite essential differences in the mechanical equipment of both, while the product of both of them is excellent. A useful and profusely illustrated description of up-to-date cement-making machinery has just been issued as a pamphlet of the Allis-Chalmers Co., of Milwaukee (catalogue No. 1,291), dealing concisely, yet fully, with rock breakers, crushers, dyrers, kilns, ball mills, tube mills, elevators and conveyors, and in general with all kinds of machinery for every stage in cement making by either the wet or dry process. The Allis-Chalmers Co. is the largest manufacturer of cement-making machinery. During the year 1903, when the total production of Portland cement in the United States was a little more than 20,000,000 barrels, the Allis-Chalmers Co. sold cement-making machinery capable in the aggregate of turning out 5,000,000 barrels of finished cement per year.

Silver.—According to the Mint figures, the supply of silver from all available sources in 1904 was 176,840,014 ounces, as against 173,222,088 in 1903. This was an increase of 3,617,926 ounces, or a gain of 2.1 per cent. Mexico was the heaviest contributor, with an output of 60,868,879 ounces, although this shows a decrease of more than 7,000,000 ounces from the output in 1903. The United States was second with a total of 57,786,100 ounces, against 54,300,000 a year earlier, a gain of 3,486,100 ounces, which almost equals the entire world's gain for the year. Australia was third, with a record of 14,558,802 ounces, against 11,909,040 ounces in 1903.

Fixation of Atmospheric Nitrogen.—In our January issue we referred on page 24 to commercial developments in connection with the Birkeland-Eyde process at Arundal, Norway. Concerning the principal features of the process see our Vol. II., pp. 300 and 507, and Vol. III., p. 33. O. N. Witt gives some additional information in *Chemical Industry*, 1905, 28, 1031, which is reported in the *Journal of the Society of Chemical Industry*, 1906, Jan. 15, as follows: The nitrogen compounds are absorbed in towers and the weak acid is used over and over again in the towers until it attains a strength of 50 per cent. The gases leaving the absorbing towers are next passed through milk of lime and then over dry lime, calcium nitrite being chiefly produced. This is treated with nitric acid, the nitrous acid evolved is oxidized, the nitric acid formed being absorbed in the absorbing towers. At present the waste of the nitric acid is fixed as calcium nitrate. In accordance with a suggestion by R. Messel, the [illegible] is converted by addition of quicklime [illegible] calcium sulphate into base nitrate, which forms a practically neutral [illegible] and one of great value as fertilizer. The [illegible]

generate electric power at $3 per horse-power-year. (With respect to this point it may be said that as a matter of fact there are some places in Norway where very cheap water-power is available, although the figure of $3 seems rather too low.)

Electric Laboratory Furnace.—A new laboratory furnace recently placed on the market by the Vereinigte Fabriken für Laboratoriums-Bedarf, of Berlin, uses platinum wires as heating resistances, the same being entirely enclosed within the refractory linings. The arrangement shows a very uniform temperature, and wear and tear or destruction of the platinum crucibles put into the furnace is impossible. When wet deposits are to be heated in the crucible, it is preferable to first place a triangle of fused silicate glass into the furnace, so that the crucible does not rest directly on the bottom of the furnace. There is sufficient space for crucibles up to 30 cc. volume. The furnace is built for voltages between 65 and 220. For very accurate quantities of work the inside walls of the furnace are covered with platinum, in order to prevent small pieces of the refractory linings from dropping down. The Laboratory & School Supply Co., which has been a New York branch for the Vereinigte Fabriken für Laboratoriums-Bedarf, of Berlin, Germany, have recently combined their business with that of the Kny-Scheerer Co., department of laboratory supplies, 225–233 Fourth Avenue, New York.

Industrial Chemistry in the South.—At the New Orleans meeting of Section C of the American Association for the Advancement of Science and of the American Chemical Society, Mr. C. A. Browne, Jr., spoke of recent developments of industrial chemistry in the South, and especially in Louisiana. The developments in the fertilizer industry, manufacture of ice, preservation of wood, distillation of turpentine, tar, methyl alcohol, etc., from wood and the utilization of wood waste for the manufacture of methyl alcohol by Classen's process were briefly discussed. The utilization of cotton-seed by-products in a number of new ways was alluded to, and reference was made to a new process for extracting oil from rice and the improvement of the rice by-products for cattle feeds. The remainder of the paper was devoted to a discussion of recent developments in the sugar cane industry, particular stress being laid upon the recent work in improving the varieties of sugar cane and in the utilization of the by-products of the sugar house—the bagasse and molasses.

Pressed and Seamless Steel Specialties.—Messrs. Janney, Steinmetz & Co., of Philadelphia, have recently issued a catalogue showing various applications of seamless cold-drawn steel, high-pressure reservoirs, boiler shells, domes, tanks, tubes, cylinders, special drawn and forged shapes in seamless steel, etc. This is an art that has lately developed tremendously.

Gas Power.—The Westinghouse Co. has recently issued very neat and fully illustrated reprints of the following papers, which contain much useful information on gas power; two papers, by A. West and J. R. Bibbins, on gas power in electric railway work, and a paper by J. R. Bibbins on gas power for high pressure city fire service.

Illinois Water-Power.—The famous case of the State of Missouri *vs.* the State of Illinois, involving the right of the city of Chicago to divert its sewage into the Mississippi River through the Chicago sanitary canal and the Illinois River, has not been decided by the Supreme Court of the United States in favor of Illinois. This decision leaves the way clear for the power development project, which is well in hand, for using the power in water power at Lockport, Ill., the lower end of the sanitary and ship canal. At this point will be installed four 5000-kilo-volt-amp., three-phase, 60-cycle, 6,600-volt alternating-current generators, now under construction in the shops of the Crocker Wheeler Co., Ampere, N. J.

Personal.

Mr. Benjamin Magnus, former superintendent of the Anaconda electrolytic refinery, and for the past two years with the Calumet & Hecla Co., at their Buffalo smelting works, has resigned his position on Feb. 15, when he left immediately for California, to take charge of special work for the Mountain Copper Co., Ltd., at their new smelters at Martinez, Contra Costa County, Cal.

Mr. Albert Ladd Colby has just sent out cards announcing that he has opened an office as consulting and inspecting engineer and iron and steel metallurgist. He has postponed the announcement until after the annual meeting of the Association of American Steel Manufacturers, when he resigned as secretary, being ineligible, as he was no longer connected with the steel manufacturers. Mr. Colby has had over twenty years experience in the steel business, for eighteen years of which time he was connected with the Bethlehem Steel Co. He has visited all the prominent works abroad, and was a juror in metallurgy at the Paris Exposition in 1900. For the past three years he was nickel-steel expert for the International Nickel Co. He has written a book on steel specifications, and is a frequent contributor to the technical press and the many scientific societies of which he is a member.

Digest of U. S. Patents.

Prior to July, 1902.

Compiled by Byrnes & Townsend,
Patent Lawyers,
National Union Building, Washington, D. C.

No. 657,937, Sept. 18, 1900, Charles B. Jacobs, East Orange, N. J.

Metallic cyanides are produced by the reaction of nitrogen upon carbides in a porous condition. In one form of the process barium carbide, melted in presence of carbon and distributed over the carbon surfaces, is then permitted to cool below the fusing point of the carbide, and is subjected to the action of nitrogen, which is directly absorbed with production of the corresponding cyanide. In another and preferred form barium carbonate or hydrate is mixed with sufficient carbon to produce the carbide and to leave an excess of carbon, and is subjected to such heat as to produce the carbide; the porous mass is then permitted to cool and subjected to the action of nitrogen as above. In either case an electric furnace of the rotary type is preferably used, tuyeres being provided for injecting nitrogen into the porous mass at some distance from the electrodes. It is stated to be essential to the best efficiency that the fused carbide shall be permitted to solidify by cooling before being subjected to the action of nitrogen. The process is applicable to various cyanides, but the reaction occurs with especial facility with barium carbide. The barium cyanide produced may be separated from the excess of carbon by solution, crystallized, and either used directly or converted by known methods into the alkali metal cyanides.

No. 657,938, Sept. 18, 1900, Charles B. Jacobs, East Orange, N. J.

An improvement on the method of 657,937, relating to the steps whereby the barium carbonate is obtained in more intimate mixture with the coke. To this end barium carbonate is mixed and finely ground with an amount of coking coal sufficient to furnish an excess of carbon over that required for the manufacture of barium carbide, the preferred proportions being three parts by weight of barium carbonate to two parts of soft coal. This mixture is coked in an ordinary retort oven, and is thereafter transferred to the electric furnace for conversion into cyanide in the manner described in patent 657,937. It is stated that a furnace producing 1 ton of product per day requires a current of about 1500 amps. at 100 volts.

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The Problem of the Combustion of Atmospheric Nitrogen.

For the undisturbed maintenance of our civilization nothing is more important than continuity. The supply of the raw material for the work of to-morrow is one of the foremost problems of to-day, and among the most pressing problems which now confront mankind is that of opening new sources of supply of fixed nitrogen—indispensable alike in peace and in war—in the industries of fertilizers and of explosives. What the fertilizer industry means to our civilization may be judged from the fact that the production of fertilizers in the United States alone has increased during the last ten years from 2,000,000 tons to over 3,000,000—and this with our abundance of virgin soil. Our chief source of ammonia is now from the destructive distillation of coal, and the supply is steadily increasing with the introduction of by-product coke ovens, though the whole production in this country is not more than 40,000 tons a year, according to the latest statistics. On the other hand, the supply of Chili saltpeter is bound to run out within the next twenty or thirty years. It is, therefore, not only a problem promising large returns to the successful inventor—no, it is a moral obligation to future generations, to render available in useful form the practically unlimited quantities of elementary nitrogen in the atmosphere. As important as the bound nitrogen is in many industries, so useless is it in the elementary state *per se*. Which, then, are the possibilities of fixing the atmospheric nitrogen in useful compounds?

* * *

If we observe how nature forms nitrogen compounds, we may study the work done by certain kinds of bacteria which bind the atmospheric nitrogen by some secret of their own. This voluntary process of Nature was made use of some hundred years ago in various European countries for agricultural purposes, but has been abandoned on account of the expense, and is now almost forgotten. More recently the United States Department of Agriculture has made interesting experiments on the artificial inoculation of the soil with cultures of these forms of bacteria. It would be most interesting to know exactly what happens. According to A. Fischer, it seems that the energy required for the formation of the nitrogen compounds is furnished by fermenting organic substances; the hydrogen given off combines with the nitrogen and the first assimilation product is presumably ammonia. Besides this process, which is at present only of biological interest, there are a number of chemical methods available.

* * *

First, it is a fact that under certain conditions oxidation of nitrogen will occur even at low temperatures, if simultaneously another substance is oxidized. An example is the observation which Schoenbein made in 1801, namely, that nitric acid is formed when phosphorus oxidizes in humid air. Schoenbein, in describing this fact, referred to the older observations of

Priestley and Cavendish on the effect of an arc discharge through air, and expressed frankly his inability to explain the peculiar ("merkwürdig") process of nitric acid formation; "our explanations leave much to be desired." With respect to the phenomenon described by Schoenbein we are not much further at the present day, though we now classify it under a general class for which we have a pretty name—antoxidation. A second possibility of fixing atmospheric nitrogen could be derived from the phenomenon that nitrogen combines comparatively easily with some "non-noble" metals, like lithium, forming a nitride, according to the equation $3Li + N = Li_3N$. By treating the nitride with water it would then be possible to get ammonia with a simultaneous formation of lithium hydroxide. To base a practical process thereon, it would be necessary to regenerate metallic lithium, which could be done by electrolysis. This method has recently been the subject of some remarks by J. Brode, who thinks, however, that the process would be far too complicated and expensive for industrial operation. The problem to prepare ammonia synthetically from the elements is very attractive. It has recently been studied in detail by F. Haber. The formation of 1-gram molecule of ammonia at ordinary temperature and at constant pressure sets 12,000 calories free. But the reaction velocity is so very slow that for practical purposes it may be considered zero. If it was possible to find a suitable catalytic agent which would accelerate the reaction, the outlook would be very good. But the best catalyzer known at present —iron—is so inefficient as to make the process practically impossible.

* * *

There remain two processes which are most interesting, since they now are becoming of commercial importance. First, the fixation of atmospheric nitrogen in form of cyanamide, and, second, the direct combustion of atmospheric nitrogen by the electric arc discharge. We hope much of interest will be found in the two articles in our present issue dealing with these subjects, not only on account of the report of progress made on a commercial scale, but also in view of Prof. Guye's excellent summary of the application of modern principles of physical chemistry to the combustion of nitrogen. A number of investigators have contributed to the evolution of this theory, the fundamental hypothesis of which is extremely simple. It is, in fact, negative rather than positive, and states that there is no specifically electric effect whatever of the arc discharge on the combination of the nitrogen and oxygen. If the atmospheric pressure is constant, it is simply a question of the proper temperature, and the electric arc discharge is nothing but the most suitable means for producing a high temperature within the gas mixture and for allowing at the same time the gas mixture to be cooled very quickly afterwards, so as to prevent the reversal of the reaction. Another method of obtaining the same result is by means of explosions of mixtures of hydrogen and oxygen gases with additions of atmospheric air; as a matter of fact, Bunsen's old experimental researches in this field have recently been made use of by Nernst in calculations concerning the combustion of atmospheric nitrogen. In this connection we may mention that it is now usual to classify very different things under the name of chemical effects of electric discharges. By electric discharges through air we can produce ozone or nitric acid, but the production of ozone requires the silent discharge, and the production of nitric acid requires the arc discharge. In the latter case we may be reasonably sure that the effect is purely thermal. In the production of ozone we may have really an electric effect of the silent discharge, but it is also quite possible that the ozone is produced by the ultraviolet light which accompanies the discharge.

* * *

With respect to the commercial production of nitric acid from air by electric discharges, we have to distinguish two stages: first, the production of a mixture of gases containing small quantities of oxides of nitrogen, and, second, the transformation of this gas mixture into nitric acid of merchantable strength or into nitrates. It has repeatedly been pointed out that the cost of the second stage will be quite high, but no definite figures are available. For the first stage of the process we have as raw material air, which costs nothing, and the whole cost of this first stage of the process consists of capital charges and cost of power and attendance. It is evident that in this case the cost of power is of fundamental importance, and in considering the commercial success of the Norwegian Co. great stress must be laid on the very low cost of power at the plant—which is three or four times less than at Niagara. The problem is so important that the success obtained in Norway should encourage American enterprise again in this field, although most careful procedure is to be recommended.

An Interesting New By-Product Development.

We have often noticed in our editorial columns interesting phases in the evolution and in the diversification of industry in the United States. A notable difference between this country and our great industrial rival, Germany, is that we pay usually much attention to labor-saving devices to meet American high-wage conditions, but that Germany pays more attention to working up by-products to meet its high price of raw materials. Necessity has been the mother of invention in each case. And the remarkable result has occurred that hard endeavor has reduced the labor charge per ton product in the United States often to a figure lower than in Germany. Likewise, profit from by-products in Germany sometimes pays the entire cost of raw materials. A seeming hardship has thus been the spur to success. An example of this Teutonic line of progress, worked to a conclusion here in the land of "unlimited possibilities," is the recent installation of a 600-ton acid plant at Ducktown, Tenn., by the Tennessee Copper Co. This concern is working a large low-grade cupriferous pyrrhotite deposit by means of monster copper blast furnaces. The smelting is now real pyritic smelting, for less than 1.5 per cent of coke is used in the charge. The gases from the copper furnace are exceedingly rich in sulphur dioxide: 6 or 7 per cent is an average figure. Moreover, the long column of charge acts like a return filter, and cleans the gas of all dust, though a dust collector has been provided for reasons of conservatism.

* * *

The utilization of the sulphur from copper ores for the production of sulphuric acid has long been used by the Nichols Chemical Co. at Laurel Hill, and from zinc ores by three zinc

works in the West. But the interesting feature of the Tennessee proposition is that the gas comes not from a roasting furnace but from a copper blast furnace. The high grade of gas made in pyritic smelting, of course, increases the capacity of the acid plant, for the "reaction velocity" of sulphuric acid-making is a maximum at about 7 per cent. Irregularities in the composition of the gas will be averaged by use of several furnaces, just as is the case in utilization of blast-furnace gas in gas engines. Another favorable fact, is the large acid market existing in the South in the phosphate fertilizer trade. The probable developments in sheet steel and wire making in Birmingham, Ala., will be another favorable future outlet for acid. This whole Tennessee development is an extremely stimulating instance of American wide-awake enterprise, and new processes of this kind are sure to be profitable where once the difficulties incident to their evolution are "met, seen and conquered."

Sliding Scale Purchases of Zinc Ore.

Dr. Franz Meyer called attention in our last issue to the extremely unsatisfactory manner in which zinc ores are sold by the miner to the spelter works in the United States. While the lead smelting industry has had the advantage of purchases of ores on a sliding scale for years, the zinc industry has been subject to distressing fluctuations in the price of ore and metal, due to lack of proper understanding between the producer and consumer of ores. This had to effect the unpleasant result of mutual suspicion and distrust between the producer and consumer of zinc ores. Such a state of affairs is unfair to both parties, and does not improve the conditions of the zinc industry in the United States. Such a remedy as Dr. Meyer has proposed is entirely in line with modern business methods and should be adopted as soon as possible. In any business at all, stability is required above all else. Indeed, the great and essential good of a "trust" is founded on the assurance of settled and permanent conditions. Improvement and advance is much more easily effected when it is known to a certainty that no great change will disturb economic conditions. The competition among the spelter workers for certain classes of ore, especially concentrates from the Joplin district, has reduced the margin of profit to a small figure, too small to be regarded as a legitimate return on capital invested in the reduction works. Such a condition is unhealthy. In the broadest sense, the interests of the mine owner and spelter company are identical, and no business can exist on a satisfactory basis if a policy of "dog-eat-dog" is pursued. If the time, worry and care expended by the management of the several zinc companies on ore purchases were expended in improvements in the plants and process, and in developing new markets for the metal and enlarging the known markets, the ultimate effect would redound greatly to the welfare of the entire industry.

* * *

A policy of settlements on zinc contents, less a certain loss of metal in the process, with a further deduction for "treatment charge," the average spelter quotation being the base, of course, for payment, has been employed for some time by the Belgian and German zinc companies. It has had exactly the result described in the previous paragraphs. The purchases of ore were made on a basis fair to both parties. The profit in the business was divided equitably. And as things go in this unsatisfactory world, both parties were satisfied with events. Nothing is truer than the statement that nowadays an enlightened self-interest produces superior results than does short-sighted selfishness. The policy of fair dealing applies universally and is retroactive. We believe that the future of the zinc industry in this country is in a measure dependent upon a mutual "get-together" attitude of all interested parties, and the adoption of the sliding scale for ore purchases, recommended by Dr. Meyer would be thus conductive to the best interests of all concerned.

The Present Scarcity of Lead Ore.

The rapid expansion of American industry in the past ten years has increased largely the consumption of all metals. This increase is figured at from 7 to 10 per cent yearly. As a consequence of such phenomenal growth, the miners of the country are called up for an enormous tonnage. In the lead business the result has been that the mines cannot meet the unprecedented demand, and naturally there has arisen a shortage of lead ores in the West and Mexico amounting to almost a famine. This condition has been clearly foreseen for some time by important interests in the metal business. As we mentioned in a recent editorial upon "copper as a collector for gold and silver," there has been apparent a decided tendency to abandon lead smelting for the recovery of the precious metals in favor of copper smelting.

* * *

As a general rule lead deposits do not extend to the depths to which copper deposits extend. There have been encountered at lower levels of Leadville, Park City and elsewhere low-grade deposits in which pyrites and blende predominate. The lead plant is thus forced to use a charge in which the lead content is continually growing smaller. And the consumer of lead is forced to pay a price for lead which is continually growing higher. Naturally, the use of zinc white as a substitute for white lead is increasing as also iron for lead pipe. Further, the advantage of the lead chamber sulphuric acid processes over the "contact" processes for 60° acid is lessening, owing to increased interest charges on lead tied up in the chambers. While all metal markets are evidencing this same pressure, in none is the question of the ore supply more serious and immediate than in the lead industry.

* * *

What will be the solution of the problem can only be stated in the most general terms. Of course, substitutes for lead will be used to a greater extent. In the future lead will thus show each year a gain in output not commensurate with the gain in other metals. More careful and closer work in ore dressing and lead smelting will increase the natural monopolistic value of the great lead mines of the United States and make valuable deposits now of little worth. Energetic prospecting and exploration will develop new lead camps in America. And so natural causes will produce the proper remedy in the course of time. But at present it is a condition, not a theory, that presents itself to the lead smelter.

Ithaca Meeting of the American Electrochemical Society.

The ninth general meeting of the American Eelectrochemical Society will be held in Ithaca, N. Y., from May 1 to 3. The hotel headquarters will be at the Ithaca Hotel.

Ithaca may be reached either by the Lehigh Valley or the Lackawanna Railroads. Prof. L. M. Dennis is chairman, and Mr. R. C. Snowdon is secretary of the local committee, and will gladly furnish any information that members may desire.

The forenoons will be devoted to the reading and discussion of papers, the afternoons to visits and excursions.

On Tuesday afternoon the chemical laboratory, the dynamo laboratory and the engineering laboratories and shops will be inspected. In the evening of Tuesday, Prof. Wilder D. Bancroft will deliver his presidential address, and later there will be a smoker at the Town and Gown Club.

On Wednesday afternoon a visit will be paid to the filtration plant, the hydraulic laboratory, the power plant in the gorge and the Ithaca Gun Co. On Wednesday evening a subscription dinner will be served. On Thursday afternoon the works of the Remington Salt Co. will be visited, and perhaps those of the Morse Chain Works.

In the following we give a list of the papers which have been announced. Since the order in which the papers will be presented has not yet been arranged, the list is given in alphabetical order of the names of the authors:

Isaac Adams, "The Development of the Nickelplating Industry."

W. C. Arsem, "The Electric Vacuum Furnace."

W. D. Bancroft, "Lecture-Room Switchboard."

O. W. Brown, "The Reduction of Metal Sulphides."

C. F. Burgess and S. G. Engle, "Observations on the Corrosion of Iron Alloys."

C. F. Burgess and O. P. Watts, "A Microscopic Study of Electrodeposits."

F. K. Cameron, "The Movement of Suspended Clay and Other Similar Particles Under the Influence of the Current."

C. F. Carrier, Jr., "The Extraction of Sodium."

F. F. Colcord, "Impurities in Electrolytic Copper."

C. L. Collens, 2d, "Some Principles of Resistor Furnace Design."

B. E. Curry, "Electrolytic Corrosion of Copper-Tin Alloys."

B. E. Curry, "Electrodeposition of Bronze."

F. A. J. FitzGerald, "On the Density of Sodium Peroxide."

H. M. Goodwin and R. D. Mailey, "On the Physical Properties of Fused Magnesium Oxide."

H. M. Goodwin and H. A. Wentworth, "Electrometric Experiments with Reference to the Dissociation of Fused Salts."

G. A. Hulett, "Cadmium Standard Cell."

L. Kahlenberg and A. S. McDaniel, "Differences of Potential Between Manganese and Lead Peroxides and Various Aqueous and Non-Aqueous Solutions."

G. L. Kennicott, "Cathodic Disintegration of Carbon in Electrolyte of Fused Chloride."

M. LeBlanc, "Electrolytic Chromium."

L. Lyndon, "Electrolyte Density in Storage Cells."

John Nelson, "The Effect of Oxides on the Adhesiveness of Electrodeposited Metals."

J. W. Richards, "Electrolysis of Caustic Soda."

F. S. Shepherd, "Errors in Pyrometry."

R. C. Snowdon, "Electrolytic Precipitation of Lead from Acetate Solutions."

L. A. Sperry, "Electrochemical Processes as Station Load Equalizers."

M. Dole Thompson, Jr., "The Free Energy of Some Halogen and Oxygen Compounds Compared from the Results of Potential Measurements."

Massachusetts Tech, "Electrolytic Corrosion of Structural Steel."

A. Van Weelde, "Electrogalvanizing."

R. von Foregger and F. G. Brindley, "Report on Experiments with Fused Sodium Peroxide in the Regeneration of Air in Submarines."

W. H. Walker, "An Instructive Laboratory Experiment in Applied Electrochemistry."

O. P. Watts, "A New Silicide of Molybdenum."

G. R. White, "Alternating-Current Electrolysis with Cadmium Electrodes."

G. R. White, "Ferromanganese Anodes in Caustic Potash."

G. R. White, "Laboratory Resistance Furnaces."

Iron Reduction at the "Soo" By the Heroult Electric Furnace Process.

On page 84 of our last issue we noticed briefly the telegram sent by Dr. E. Haanel to the Canadian Minister of the Interior concerning the full success obtained at Sault Ste Marie in the electric furnace trials of Dr. Paul Héroult.

Further information has become available through a lecture of Dr. Haanel, delivered before the Canadian Club in Toronto on March 12. In the audience were a great many Canadian business men, and the subject of Dr. Haanel's address was an explanation of the peculiar situation of Canada with respect to the manufacture of iron and steel and an exposition of the possibilities which the successful outcome of the tests of the Héroult process opens for Canada.

The distribution of the raw materials required for iron reduction in Canada is peculiar. The necessary coking coals for blast furnace work are found at the extreme East and West, while Ontario, Quebec, Saskatchewan and Alberta have large ore deposits but no coal; on the other hand, there are large water powers here available. This peculiar situation is favorable to an electric smelting process, since the electrical energy for heating the charge could be generated from water power, so as to make unnecessary the expense of fuel for heating purposes.

However, there would still be required carbon for reduction, and the question now arose whether the coke used in the blast furnace for this purpose could not be replaced in the electric furnace by charcoal, which could be made from mill refuse and other available sources of wood useless for other purposes, or by peat coke made from peat, of which there are abundant deposits in Ontario and Quebec.

Finally, the chief ores available in Canada are magnetite, and most of them carry such a high sulphur content as to be valueless for blast furnace smelting. The question was, if these magnetites could be treated successfully in the electric furnace with practical elimination of sulphur.

These were the principal questions which remained to be answered by practical tests, and they formed the programme of the tests at Sault Ste Marie.

It will be remembered that a few preliminary tests on the reduction of iron from ore in the electric furnace were made in France in 1904, and fully analyzed and discussed in the report of the Canadian Commission before which these tests were carried out. (See our Vol. II., p. 470.) From these tests the metallurgist of the Commission, Mr. Harbord, concluded that pig iron can be produced on a commercial scale at a price to compete with the blast furnace, only when electric energy is very cheap and fuel very dear. On the basis of the price of the electric horse-power-year at $10, and of coke at $7 per ton, the cost of pig iron production in the electric furnace is approximately the same as in a modern blast furnace. Dr. Haanel, the head of the Commission, already pointed out at that time that these results were obtained in experiments made off hand, so to speak, in furnaces not at all designed for the reduction of iron ore, and that the results, therefore, indicated only the minimum that may be expected from the electric furnace.

It was for the purpose of giving the electric process a better chance of showing its possibilities, and for the further pur-

pose of solving the above-named special problems concerning high-sulphur ores and concerning the use of charcoal or peat coke, that the experimental plant was erected at Sault Ste Marie. The Lake Superior Power Co. offered a building in which to erect the plant, and the use of a 300-hp. generating set free of expense for four months. This offer was accepted.

The Canadian Government granted the amount of $15,000 for the tests.

Dr. Paul Héroult designed the plant, and the tests were carried out under his direction and responsibility. He was assisted by Mr. R. Turnbull and Mr. J. Sejournet. The Canadian Government was represented by Dr. Eugene Haanel, Dominion Superintendent of Mines. He was assisted by his son, Mr. B. Haanel, and by Mr. E. Nystrom.

The experiments with Canadian ores began in earnest the middle of February, the furnace being in operation night and day, with some intermissions, until March 5. During that time about 150 casts were made yielding 55 tons of pig iron. For the first experiments the ores employed were hematite, such as are used by the Algoma Steel Co. in their blast furnaces. For the remainder of the experiments different classes of Canadian magnetite, from the different sources of supply, all of high sulphur content, with the exception of Wilbur magnetite, which was low in sulphur, were employed. In Dr.

FIG. 1.—HEROULT PIG IRON FURNACE AT SAULT STE. MARIE.

Haanel's address the chief results of the tests are summed up as follows:

Magnetite can be smelted just as well in the electric furnace as hematite, there being no difficulty of using ores of high sulphur content for the production of pig iron containing only a few thousandths of 1 per cent of sulphur, even when the slag was not particularly basic. The silicon content can be varied as required for the class of pig to be produced.

Charcoal and peat coke can be substituted for coke as the reducing agent without being [illegible] difficulty whatever was experienced from this source.

In every instance the output was far greater, in several instances one-third greater, than the figures adopted by Mr. Harbord in the report of the Commission on electric smelting.

In continuation of work formerly done by Stassano (see Vol. II., pp. 153, 188, 207; see also the letter by Mr. S[illegible] on page 128 of this issue), Héroult succeeded in using roasted nickeliferous pyrrhotite for the production of a ferro-nickel pig of fine quality, and practically free from sulphur. The pyrrhotite contained about 1.6 per cent of sulphur, while the ferro-nickel pig produced contained 4½ per cent of nickel, and

FIG. 2.—PART OF THE PIG IRON PRODUCED IN ELECTRIC FURNACE.

was virtually free from sulphur. The estimated value of this product is $40.00 to $44.00 per ton.

Another improvement over former results, as stated by Harbord, relates to the consumption of the electrodes. This was small beyond expectation. When an electrode had been in use for three weeks, and during that time had been exposed to free air in an incandescent state for many hours, and had been used for melting down charges—which is always attended by a waste of electrode without a corresponding output of metal—even with this severe test the consumption per ton of pig iron produced was between 15 and 20 pounds of electrode. According to Dr. Héroult's estimate this item means an outlay of 30 cents per ton of pig iron.

From a commercial point of view it is important that [illegible] ore is made valuable by the electric process. A blast furnace will not usually handle high sulphur ores, and requires, therefore, an ore which cannot be bought in Canada at a low figure. Dr. Haanel stated that he understands the Algoma Steel Works paid $3.75 for the hematite ore which they use in their furnace. Pig iron equal in value and lower in sulphur content can be made by the electric process from high-sulphur ores, which can be bought for $1.25.

Pyrite cinders resulting from the roasting of pyrite in the manufacture of sulphuric acid, and which at present constitute a waste product, can be smelted into pig iron in the electric furnace. Titaniferous iron ores containing up to 5 per cent of titanium may also be treated successfully in the electric furnace. This latter fact is inferred from the successful treatment of ore containing up to 30 per cent titanic acid.

Dr. Haanel said that under the prevailing conditions in Canada it now only remains for the engineer to design a plant on a commercial scale of, say, 100 to 150 tons daily output, with all the necessary labor-saving appliances. Dr. Haanel concluded his address with the following appeal to the business men present: "The Government has furnished you with facts on which to base a sound judgment as to the feasibility of commercially engaging in the manufacture of pig iron by

the electric process; with that its duty to the nation is done, and it remains with you business men to apply, perfect and profit."

By the courtesy of Dr. Héroult we are enabled to reproduce two photographs which were taken at the Sault Ste Marie plant. Fig. 1 shows the 250-hp. electric furnace in operation. Fig. 2 shows a pile of pig iron produced, the gentleman standing on the right being Dr. Haanel.

We are obliged to Dr. Haanel for courteously placing at our disposal the photograph reproduced in Fig. 3, and showing the electric furnace with electric control apparatus and measuring instruments.

Dr. Haanel did not give in his lecture the exact figure for the consumption of electric power per ton of pig produced. Dr. Héroult states that an output of 12 tons of pig iron per

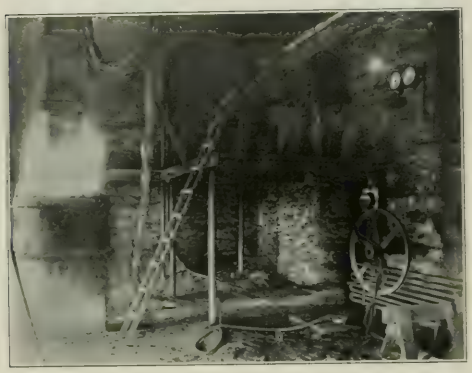


Fig. 3.—Electric furnace, together with control apparatus and measuring instruments.

day may be obtained with 1,000 electric horse-power. This means that 1 ton of 2,000 pounds requires some 83-hp. days (for comparison we may mention that in the report of the Commission two years ago Mr. Harbord stated it would not be safe to assume less than 0.350-hp.-year per ton of pig iron, which corresponds to 128-hp.-days per ton). Dr. Héroult emphasizes that this figure of 83-hp.-days per ton may be very much improved upon by the use of his method of changing all the carbon monoxide into carbon dioxide in the furnace itself. The object of this would be, of course, to utilize to the utmost the calories available in the fuel employed for reduction. Practical means for this purpose are described in a patent by Dr. Héroult, which has just been granted, and which will be found in this issue of our Analysis of Current Electrochemical Patents.

In this connection it should be stated here that in the main experiments at Sault Ste. Marie no air blast was used to oxidize the remaining carbon monoxide. Two short tests which were made in this direction indicated a great improvement in the operation, but these tests were too short to derive from them exact figures. While using the term air blast in connection with the Héroult furnace, we may emphasize that the air blast has, of course, here, a function entirely different from the blast in the blast furnace.

The full detailed report on the tests which is now being prepared by Dr. Haanel will be awaited with great interest. But from the general results recorded above it is already evident that the outcome has been satisfactory far beyond conservative expectations, and Dr. Héroult, as well as the Canadian Government and Dr. Haanel are to be heartily congratulated on the success of their enterprise.

Dr. Haanel's detailed report will be reported in these columns as soon as received.

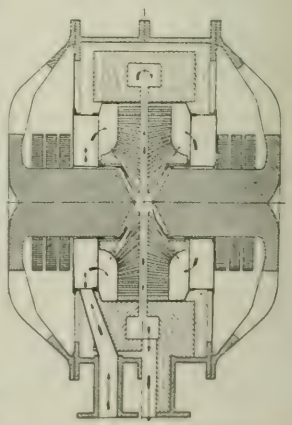
Recent Developments in the Production of Nitrates From Air.

Of the various processes for the fixation of atmospheric nitrogen in form of nitric acid or nitrates, the Bradley-Lovejoy process, which was in operation at the works of the Atmospheric Products Co., in Niagara Falls, was abandoned more than a year ago, as was noticed at the time in our columns. It is now stated that the Kowalski-Moscicki system (our Vol. I., p. 462, and Vol. II., p 152) which was tested on a large scale in Europe, has also been abandoned. The only survival of the processes which have passed through large-scale tests is that of Prof. Birkeland and Mr. Eyde in Kristiania, Norway, and this process appears to be in very good shape. Prof. O. N. Witt, in Berlin, and Prof. S. P. Thomson, in London, who visited the Norwegian plant as consulting experts, recently gave in lectures some new information.

The Birkeland-Eyde process was first brought before the public in a paper by Mr. J. S Edstrom before the St. Louis International Electrical Congress; this paper and further notes on the process may be found in our Vol. II., pp. 399 and 507, and Vol. III., p. 33.

The plant is located at Notodden on the shore of the Hitterdal Lake, whence there is water communication through a canal to Skien, and from there either to Kristiania or to Hamburg. The company also has a research laboratory at Vasmoen, near Arendal, which is stated to be magnificently equipped, and which has already undertaken the investigation of several other chemical processes of kindred nature.

The furnace is extremely simple. On each side of the "roaring disc of flame" there are fire-clay cheeks pierced with holes for the inlet of air, which is driven into the central region under a gentle pressure by a Roots blower, and passing radially arrives at a peripheral channel whence it is conducted away. The whole is enclosed in an external copper case, into which, on either side, there enter the poles of a large electro-magnet. The horizontal electrodes are of copper. They approach one another to within about one-third inch, and are hollow, to permit of their being cooled by a water circulation that keeps them from fusing. They are not consumed, and need only to be replaced at intervals of a few weeks, when they have become slightly eroded or roughened on the surface.



Electric furnace for combustion of atmospheric nitrogen.

The original form of the furnace was illustrated in Fig. 2, page 399, of our Vol II. An improved form of the furnace is shown in the adjoining illustration. The nitric oxide fumes formed in the arc absorb oxygen from the unconsumed part of the air, and turn to nitric peroxide, which, when treated with water, combines to form nitric acid. But if the company had chosen to make nitric acid as final product for sale, its transportation to places of consumption would not have been an easy matter. The production of sodium nitrate by absorption of the nitric acid in caustic soda would have required importing caustic soda to Norway. On the other hand a very pure limestone is avail-

able in large quantities in the neighborhood. For this reason it was decided to produce calcium nitrate.

This "Norwegian saltpeter" has proved a valuable fertilizer, and is claimed to have a certain advantage over Chili saltpeter, owing to the nature of its basic constituent. When large quantities of sodium nitrate are employed for agricultural purposes it is frequently observed that the sodium acts prejudicially upon the vegetation; whereas, especially on soils poor in lime, the calcium nitrate is valuable both for its nitrogen and for the lime it introduces into the ground. Again, the synthetical calcium nitrate is necessarily free from all chlorine compounds, particularly perchlorates.

Calcium nitrate forms clear crystals containing four molecules of water of crystalization. By further treatment a nearly anhydrous nitrate of lime is obtained containing about 13 per cent of nitrogen. Thirdly, by the further addition of caustic lime a basic nitrate of lime is produced which is quite anhydrous. The manufacture of this basic nitrate was suggested by Dr. Rudolph Messel, of London.

At present nothing but calcium nitrate is made at Notodden, and the whole output can be disposed of easily. Great difficulties were first experienced in working up into acid of proper strength the very weak acid and highly diluted acid vapors issuing from the apparatus, but these difficulties are stated to have been completely overcome.

There are three furnaces of 500 kilowatts each. Air is supplied by blowers at about 50 liters per minute per kilowatt. The hot gases first pass through boilers where steam is raised, the steam afterwards serving to evaporate the nitrate liquors which are finally obtained. In a new plant which is being erected the hot gases are to be led directly through the evaporating pans, which change will effect a considerable economy. The cooled gases then enter two large oxidation towers, constructed of acid-resisting brick, and pass slowly through the same, emerging at a temperature of some 50° C. to enter the absorption towers, about 50 feet high, built of granite slabs and filled with lumps of quartz, over which water trickles.

The first tower yields a 50 per cent nitric acid, the second about 25 per cent, the third 15 per cent, the fourth 5 per cent. The liquids from the fourth tower are raised by compressed air to the top of the third, those from the third to the second, those from the second to the first, thus gradually increasing in concentration up to 50 per cent, at which density the acid is drawn off.

The gases then pass through a fifth tower in which they are treated with milk of caustic lime, and finally through a wooden tower over beds of dry quick lime. The 50 per cent acid is treated with limestone to form nitrate of lime, which is concentrated and poured molten into iron canisters ready for shipment.

The number of workmen is surprisingly small, most of the processes being automatic. One single assistant engineer attends the three furnaces; and his principal duty is to enter in the log-book every quarter-hour the readings of the electrical measuring instruments.

In his St. Louis Congress paper Mr. Edstrom stated that one kilowatt-year yielded 990 kg. of HNO_3, the electrical energy being measured at the arc. According to Prof. O. N Witt the yield is between 500 and 600 kg. of anhydrous nitric acid per kilowatt-year, but not infrequently higher yields are obtained. It is not quite clear whether this reduction of the efficiency is an actual fact (due to a smaller efficiency on large scale than on an experimental scale), or whether it is only apparent, since the figure of Prof. Witt may give the kilowatt hours consumed at the bus-bars instead of at the arc.

The water-power at Notodden was first developed for a pulp factory, and the engineering conditions were so favorable that the capital expenditure was under $27.50 per horse-power, and the cost of production only about 10 kronen per horse power year which is about 0.04 cent per kilowatt-hour. The owners of the pulp factory sell part of their power to the saltpeter factory, at 23 kronen per horse-power-year, or at about $8.20 per kilowatt-year, or 0.094 cent per kilowatt-hour, yielding them a handsome profit.

Within a few months a new factory, more than double the size of the present one, will be erected at Notodden, and to supply its requirements another water-power in the neighborhood will be developed, and when delivered from the transmission line at the Notodden factory it is claimed that the cost of power will be less than $4.00 per horse-power-year. The projects of the company do not stop with the large factory at Notodden. Already powers have been acquired for the utilization of three other water-powers, one of 25,000 hp., a second one of 40,000 hp., and a third one still larger.

It is also reported that the Badische Anilin und Soda Fabrik, of Ludwigshafen, propose to lay down a water-power plant for the production of nitric acid by the Birkeland-Eyde process. However, instead of making calcium nitrate or sodium nitrate for artificial fertilizers, it is intended, in the first place, at least, to make potassium nitrate for explosive purposes.

In connection with these developments attention may be called to an excellent summary of the physical chemistry of the problem, which is given in an article by Prof. Guye, published elsewhere in this issue.

In Memoriam—Eugene R. White.

Eugene R. White, born in Buffalo July 19, 1872; graduated from Williams College in 1894; after leaving college he was engaged in newspaper work in Buffalo, and made some mark as a writer of prose and verse; in 1898 he became editor of the *Niagara Falls Gazette;* he died in Buffalo on March 16, 1906, after a short illness.

It is not necessary to dwell upon the loss of Eugene R. White as far as literature is concerned, for those who knew him and his purely literary work recognize fully not only the value of what he has done, but the promise there was of still better work in the future. But there is another aspect of the man which is likely to be overlooked, since from its very nature it could be known to but few.

There is a vulgar error, unfortunately very prevalent, that science and art are essentially antithetical. The blame for this lies partly with the man of science who will not take the pains to reveal his worth to the public, and partly with the literary man who generally is out of sympathy with scientific pursuits. Fortunately, there are exceptions to the rule, and a notable example was found in Mr White. Till he came to Niagara Falls in 1898 his work had chiefly been of a literary nature; but here he found himself thrown into the society of men engaged in work of a scientific character. From that time on his interest in scientific subjects, and more especially in those of an electrochemical and electrometallurgical nature, continually increased. His broad sympathy with all forms of intellectual activity enabled him to appreciate at their value scientific pursuits. He quickly recognized the importance of public recognition to those engaged in research work, whether the object of such work was commercial or purely scientific, and he devoted much of his unusual ability as a writer to secure such recognition for those in need of it. Work of this kind demands an unselfish character, since the only reward the worker receives is the knowledge that he has done something towards the general progress of humanity. Those outside of his most intimate friends know nothing of what he has done for others.

It was my privilege to be intimately acquainted with Mr. White from the time he first came to Niagara, and during that period I had abundant evidence of his ever present desire to assist the progress of science and industry. It is sad to think that little can ever be generally known of the help given by Mr. White in his own way to the progress of electrochemistry.

My last conversation with him was on the day before his fatal illness began, and we then discussed the question as to how he could be of assistance in the development of a recent advance in electrochemical work.

By Mr. White's death the electrochemical industries of Niagara Falls have lost a valued ally and those who knew him personally a friend ever ready with sympathy and encouragement. Francis A. J. FitzGerald.

New York Section American Electrochemical Society

By courtesy of the officials of Columbia University, a very enjoyable meeting of the New York Section of the American Electrochemical Society was held on March 22 in Havemeyer Hall.

Dr. Richard von Foregger presented a preliminary paper on the use of oxone for the purification and regeneration of air, and gave an account of a series of experiments made by him in conjunction with Mr. G. F. Brindley. These very interesting researches will form the subject of a paper at the Ithaca meeting, and will then be reported in our columns. An evident application of the method is, of course, for submarine boats, but the use of oxone was also suggested for purifying the atmosphere in the New York Subway.

Prof. Samuel A. Tucker explained the arrangement of the newly-equipped very fine electrochemical laboratories of Columbia University, which were then inspected. Refreshments were served afterwards. We hope to give a full description of these laboratories in a special article in a later issue.

CORRESPONDENCE.

Metallurgy of Zinc.

To the Editor of Electrochemical and Metallurgical Industry:

Sir:—The article by Dr. Franz Meyer in your last issue on the zinc industry in the year 1905 is extremely interesting and comprehensive. It, however, contains the incorrect statement that the American zinc metallurgists have learned how to treat ores containing as high as 20 per cent iron by the means of hydraulic-made retorts. The writer, over eighteen months ago, charged zinc ore averaging on an entire plant for some period over 22 per cent iron. The loss in zinc per ton of roasted ore was only some 10 pounds higher than the loss of zinc per ton of roasted ore in the furnaces operating on roasted concentrates from the Joplin field. The retorts used were the old-fashioned "tile-made" retort. This result was secured only by extreme care in roasting the ores, in blending them, in grinding the reduction material, in selecting the same and in blending ore and fuel carefully so as to produce a "non-corrosive" charge and one that would possess a higher reaction velocity.

The writer was not at all a pioneer in this line, but there are several other plants in the Kansas field who had made similar advances.

The writer does not wish at all to decry the merits of the Mehler hydraulic retort press. On the contrary, he is of the opinion that an installation of the Mehler press would pay for itself in from 90 to 120 days, due to increased recovery, from lessened zinc "seckage," from better condensation (due to the fact that the denser retort allows no filtration of the carbon dioxide, water gas and oxygen inside the furnace to the retort with a diminished reductivity of the indifferent gases), and also from the fact that the sacrificing material of the charge permeates to a less degree the cracks in the retort.

New York City. Woolsey McA. Johnson.

Manufacture of Ferro-Nickel.

To the Editor of Electrochemical and Metallurgical Industry:

Sirs—I notice in your March issue of the Industry, page 105, that in Mr. Steinhart's paper at the January meeting of the Institution of Mining and Metallurgy, reference is made to the experiments made here some years ago in the production of ferro-nickel from roasted pyrrhotite in the electric furnace in the following terms:

"The Lake Superior Power Co., in 1902, attempted, in conjunction with E. A. Sjöstedt, to produce ferro-nickel in the electrical furnace from dead-roasted Canadian ores, but this process never came into practical use, although a large plant was put down."

The facts in regard to this matter were clearly put forth in my paper at the Fifth General Meeting of the American Electrochemical Society in 1904, from which I, therefore, beg to quote the following paragraph:

"Simultaneously with the above work (the electric smelting of a partially roasted pyrrhotite, containing about 3 per cent S), experiments in dead-roasting the pyrrhotite were also carried out, and these being sufficiently encouraging to warrant the belief that this could be done at small cost, while at the same time the resulting SO_2 gas could be profitably utilized in the sulphite pulp industry, and as the electric smelting plant on the large scale intended *would necessitate the completion of a large power plant*, it was decided to carry out the original intention of roasting the ore, briquetting the roasted fines, converting the briquettes into pig iron, and finally converting this pig metal by means of the open-hearth or Bessemer process into a ferro-nickel steel."

Thus it is made plain (1) that no electric installation for the manufacture of ferro-nickel was ever put down here; (2) that at no time was it our aim or intention to resort to electric furnace smelting for producing a ferro-nickel from a *dead-roasted* pyrrhotite, but from an incompletely roasted ore—a material so high in S as to make it unfit for the common blast furnace process.

The successful roasting of pyrrhotites on the lines as above indicated, with the utilization of the SO_2 gas in the sulphite mill, was subsequently carried on here in a plant erected, capable of treating 40-50 tons of pyrrhotite per day, which plant was in continuous operation until, in 1903, the Consolidated Lake Superior Co. got into temporary financial trouble, when work was stopped; but as soon as the conditions of the new Lake Superior Corporation will permit, it is intended again to start these works, practically on the lines as at first suggested.

For the sake of furnishing a stronger SO_2 gas for the sulphite mill than what is possible when attempting to secure a dead-roast of the ore, our future practice would be not to aim at removing the S contents below 2 per cent, and to manufacture a ferro-nickel pig out of these "cinders" in the electric furnace.

The experiments lately carried out here by Dr. Heroult, under the supervision of the Canadian Government, have not only verified our previous results, but were made on a scale sufficiently large to encourage us to believe that, even in this small furnace, we will be able to profitably produce a ferro-nickel alloy on a semi-commercial scale, thus enabling us to follow up these researches under much more favorable auspices than previously.

I also wish to take exception to Mr. Steinhart's statement that "the companies 'La Société Electrometallurgique de Froges' and 'Le Ferro-Nickel' have produced ferro-nickel commercially," as I have been informed by parties lately connected with the first mentioned company that no such product has as yet been made by *them*, at least, on any commercial scale. Ernst A. Sjöstedt.

Chief Metallurgist, Lake Superior Corporation.
Sault Ste Marie, Ont.

Metallurgical Calculations.

By J. W. Richards, Ph. D.
Professor of Metallurgy in Lehigh University.

BALANCE SHEET OF THE BLAST FURNACE.

As the most important factor in the production of the most important metal, the blast furnace is the most important furnace or piece of metallurgical apparatus in the world. It is, therefore, proper that we should commence a series of articles on the application of metallurgical principles and calculations to the metallurgy of iron, by a discussion of the blast furnace; and since this discussion, to be complete, must include a wide range of topics, we will commence with the simplest, viz.: the balance sheet of materials entering and leaving the furnace. Later we can discuss the balance sheet of heat entering in, developed within and leaving the furnace, the reactions taking place in the furnace, the action of hot and of dried blast, the calculation of the proper constituents of the charge, the temperatures attained before the tuyeres, unused combustible energy of the gases, efficiency of the hot-blast stoves, and other interesting and practically valuable factors in the running of the furnace.

The blast furnace may be regarded from several points of view; we will mention two. First, it may be regarded as a huge gas producer, run by hot, forced blast, in which the incombustible portions of the contents are melted down (with a little unburnt carbon) to liquid metal and slag, and are run out beneath, while the gaseous products pass upwards through 50 to 100 feet of burden, and escape above. The escaping gases are primarily of the composition of producer gas, with some of its carbonous oxide changed to CO^2 by the oxygen abstracted from the burden, with some CO^2 added from the decomposition of the carbonates of the charge, and with the usual increment of moisture from the charge and volatile matter (if any) from the distillation of the fuel. From this point of view, the blast furnace is a huge gas producer, giving a rather inferior quality of combustible gas in very large quantities, and incidentally reducing to metal and slag the burden of iron ore and flux (limestone) which is put in with the fuel. The treatment of the furnace as a metallurgical problem may then proceed as the discussion of a gas producer, with the composition of the gas produced somewhat modified by the amount of oxygen given up to the gas by the reducible portions of the charge of the furnace.

The other viewpoint is to regard the furnace as primarily an apparatus for deoxidizing or reducing iron ore, for which purpose the ore is charged with sufficient carbonaceous fuel to do two things, viz.: to abstract all the oxygen from the reducible metallic oxides, and to furnish enough heat, or high enough temperature, to melt down to superheated liquids the pig iron and slag (combinations of irreducible metallic oxides) formed. In this view, the fuel must supply the reducing energy and the melting-down or smelting requirements; the first by acting upon the metallic oxides at a red to white heat, and abstracting their oxygen; the second, by being burned at the foot of the furnace by hot air blast, and there generating the heat and higher temperatures necessary for the smelting down of the already reduced materials.

Materials Charged and Discharged.

The materials put into a blast furnace may all be classed under four heads:

Fuel	Charged at the throat
Iron ore	Charged at the throat
Fluxes	Charged at the throat
Blast	Blown in at the tuyeres

The materials discharged from the furnace may be classed under four heads also:

Pig iron	Tapped from the crucible
Slag	Tapped from the crucible
Gases	
Dust	

We will discuss the [illegible] charged into the [illegible] of escape.

Fuel.

The fuel used is sometimes charcoal, but in the great majority of cases coke, with perhaps some raw bituminous coal or anthracite coal, or in a few cases all raw bituminous coal. The composition of these fuels consists of moisture, volatile matter, fixed carbon, sulphur and ash consisting of silica, iron, alumina, alkalies, etc.

The moisture is driven off near the top, and goes off with the gases as moisture. The volatile matter is expelled near to the top; almost all of it goes unchanged into the gases, but some of the hydrocarbons thus expelled may be decomposed and deposit fixed carbon on the iron oxides, etc., surrounding them. This carbon, however, will take up oxygen from the charge lower down in the furnace, and thus eventually pass into the gases as CO or CO^2. We can, therefore, assume without error that all the volatile constituents of the fuel pass into the gases, but cannot be certain in exactly what state of combination, except as regards the moisture. It will be quite exact if we know the ultimate composition of the volatile matters of the coal, as so much carbon, hydrogen, oxygen, nitrogen, sulphur, etc., to charge them thus entirely to the gases.

The fixed carbon all finds its way ultimately into the dust or the gases, either as CO, CO^2, CH^4 or HCN, or alkali cyanides, excepting the amount represented by the carbon in the pig iron. Subtracting the carbon in the pig iron from the total fixed carbon in the fuel, the difference can safely be put down as entering the gases or being in the dust carried away by the gases.

The sulphur in the fuel has a more varied history. When it is partly present in the form of iron pyrites, some may go into the gases as sulphur vapor, and eventually be burned to SO^2 when the gases are burned; another part may be oxidized in the furnace itself to SO^2, and as such appear in the gases; the rest, along with organic sulphur, passes either into the slag or the pig iron. Sulphur passing into the slag seems to do so as calcium sulphide, CaS, formed by some such reaction as

$$CaO + C + FeS = CaS + CO + Fe$$

or, if the sulphur was present in the fuel as gypsum,

$$CaSO^4 + 4C = CaS + 4CO$$

The amount of sulphur going into the iron depends really upon the opportunity for it to go into the slag. If the temperature of the furnace at the tuyeres is very high, and especially if the slag is low in silica, sulphur will keep out of the [illegible] and go into the slag, to the extent of ten or twenty times as much being in the slag as in the iron; but if the temperature is low and the slag rich in silica the reverse may be the case. A high temperature and a high percentage of lime in the slag are the blast furnace manager's means of keeping down the sulphur in the iron, although high magnesia or high alumina are also efficacious. In casting up the balance sheet it can be assumed that when using coke or charcoal all the sulphur in the fuel goes either into the slag or the iron, and knowing from the analysis of the pig iron made how much goes into it, the rest can be calculated as going into the slag as CaS. If raw coal is used, it is uncertain how much sulphur goes into the gases, and an exact analysis of either the slag or gases for sulphur, in addition to that of the pig iron, would be necessary to fix its distribution.

The ash of the fuel counts toward the [illegible] ingredients of the charge. Some of the silica of the ash is reduced to silicon and some of the CaO to Ca, but these [illegible] most of the [illegible] slag. It is in most cases uncertain whether the silicon in the pig iron comes at all from the [illegible] from the [illegible]

assume it all reduced to the metallic state, as is probably always the case.

Besides all these avenues of escape for the constituents of the fuel, it is sometimes necessary to take into account the possibility of some of it, in fine particles, being carried out of the furnace bodily with the outgoing gases. If the amount of this in the dust is determined, it must be subtracted *in toto* from the fuel charged, and then the remainder distributed as just discussed.

Illustration: A blast furnace is charged, per 1,000 kilos. of pig iron produced, with 925 kilos. of coke, containing by analysis: Fixed carbon, 86 per cent; volatile carbon, 2; hydrogen, 1; oxygen, 0.5; nitrogen, 0.5; sulphur, 1.0; iron, 2; silica, 5; lime, 1; moisture, 1. The pig iron contains 3.5 per cent of carbon and 0.1 per cent of sulphur. The dust carries 15 kilos. of dry coke per metric ton of pig iron. Required the distribution of the coke in the furnace per ton of pig iron made:

Charge		*Pig Iron*		*Slag*		*Gases*		*Dust*	
Coke	925.0 kilos.								
Dust	15.0 "							Coke	15
Fixed C	782.6 "	C	35.0			C	747.6		
Volatile C	18.2 "					C	18.2		
H	9.1 "					H	9.1		
O	4.5 "					O	4.5		
N	4.5 "					N	4.5		
S	9.1 "	S	1.0	S	8.1				
Fe	18.2 "	Fe	18.2						
SiO^2	45.5 "			SiO^2	45.5				
CaO	9.1 "			CaO	9.1				
H^2O	9.3 "					H^2O	9.3		

Ore.

Whatever the varieties of ore used they can be averaged together, so as to get the average composition of the ore charged. Then, knowing its weight per unit of pig iron made, the distribution into pig iron, slag, gases and dust can be made.

There may, first of all, be blown out as ore dust up to 25 per cent of the ore charged. This must be first deducted as dry ore and then the rest distributed to pig iron, slag and gases.

The moisture of the ore, also any carbonic acid, may be considered as going over bodily into the gases. The sulphur may partly go into the gases if present as iron pyrites (this amount would have to be checked by an analysis of the gases for sulphur or hydrogen sulphide), but mostly into the slag as CaS. Some of it may be put down as going into the pig iron, if the sulphur in the fuel does not account for all that appears in the analysis of the iron. The iron oxides present must be assumed reduced to metallic iron sufficient to furnish the iron in the pig iron from its analysis; the excess, if any, is put down as going into the slag as FeO; all the oxygen given off (the difference between the weight of iron oxide in the ore and the sum of iron going in the pig iron and ferrous oxide passing in the slag) goes to the gases. If there is not enough iron in the ore to account for all in the pig iron, then none is assumed to go into the slag.

The manganese oxides in the ore furnish the manganese in the pig iron, the excess going into the slag as manganous oxide MnO, the oxygen (by difference) goes to the gases. The proportion of manganese reduced to metal increases with the temperature at which the furnace is run and as the slag is less siliceous. The amount reduced is known, however, only by the analysis of the pig iron.

Zinc in the ore is partly found as ZnO in the slag, and partly as flakes of white zinc oxide in the gases, which latter partly deposit in the dust catcher and are partly carried by the current of gases into the stoves and under the boilers. The relative amounts going into slag and gases can be best controlled by analysis of the slag.

Copper, silver, gold, nickel, cobalt, phosphorus, antimony and arsenic are almost completely reduced into the pig iron; careful analysis of the latter will show exactly to what extent, but without this careful analysis they must be assumed to pass completely into the iron. Lead is mostly carried out as fume, a small amount passes into the pig iron, and, if present in quantity, a large amount may collect as metallic lead beneath the pig iron and, if it can, soak into the foundations of the furnace.

Alumina usually passes completely, as such, into the slag. When present in large amount, producing a slag rich in alumina, and with very hot blast, the pig iron may contain as much as 1 per cent of aluminium, the oxygen thereof passing into the gases. Magnesia may be assumed as passing completely into the slag; none is reduced. Lime goes into the slag, except a not-unimportant quantity which is reduced by carbon in the presence of sulphur compounds, and forms CaS, its oxygen passing into the gases; a very small amount may go as calcium into the pig iron. Alkaline metals partly go into the slag, while some may pass into the gases as alkaline cyanides. Titanium oxide, tungsten oxide, chromium oxide and the oxides of molybdenum, uranium, vanadium are sometimes reduced in small amounts, the more the hotter the furnace is run and the more basic the slag, while the bulk of them passes into the slag as the lowest oxide which each is capable of forming.

Silica mostly goes into the slag as SiO^2, but a portion is always reduced to silicon in the pig iron. The amount reduced is greater the hotter the furnace is run, the more slowly it is run, and the more siliceous the slag. In some cases as much as one-quarter of all the silica going into a furnace is reduced to silicon. It is probably reduced only by carbon dissolved in iron at the lower part of the furnace. The oxygen of the silica reduced goes into the gases.

Illustration: 1956.8 kilograms of ore is charged into a furnace per metric ton of pig iron made. The ore analyzes: Fe^2O^3, 71.43 per cent; SiO^2, 14.24; CaO, 2.05; MgO, 1.51; MnO^2, 4.15; SO^3, 1.40; H^2O, 5.00; Cu^2O, 0.22 per cent. The pig iron contains 93.03 per cent iron, 3.27 carbon, 1.20 manganese, 0.08 sulphur, 0.40 copper, 2.02 silicon. Assume the dry ore dust to weigh 4 per cent of the weight of ore charged, and that there is no sulphur found in the gases, but all the sulphur in the pig iron comes from the fuel. Cast up the distribution of the ore:

Charge		*Pig Iron*		*Slag*		*Gases*		*Dust*	
Ore	1956.8 kg.								
Dust	78.1 "							Ore	78.1
Fe^2O^3	1339.2 "	Fe	930.3	FeO	9.1	O	399.8		
SiO^2	267.0 "	Si	20.2	SiO^2	223.6	O	22.2		
MnO^2	77.8 "	Mn	12.0	MnO	48.0	O	17.8		
Cu^2O	4.1 "	Cu	3.6			O	0.5		
CaO	38.4 "			CaO	20.1				
				Ca	13.1	O	5.2		
MgO	28.3 "			MgO	28.3				
SO^3	26.3 "			S	10.5	O	15.8		
H^2O	97.6 "					H^2O	97.6		

In calculating the above we note that the dust is dry, and weighs 4 per cent of the ore, making 78.1 kilos. of dry dust, representing 82 kilos. of moist ore, leaving 1874.8 kilos. of moist ore to be distributed, plus the 3.0 kilos. of water from the dust, which also goes into the gases. This 1874.8 kilos. contains the weights given, calculating from its analysis. The 1339.2 kilos of Fe^2O contains 937.3 kilos. of iron; but there are only 930.3 kilos. in the ton of pig iron, therefore the other 7.0 kilos must go into the slag as $7.0 \times \frac{72}{56} = 9.1$ kilos. of ferrous oxide, while $1330.2 - 930.3 = 9.1 = 400.8$, the weight of oxygen going into the gases. The 267.0 kilos. of silica contains 124.6 kilos. of silicon, but there are only 20.2 kilos in the pig iron, therefore, 104.4 kilos must remain unreduced, passing into the slag as $104.4 \times \frac{60}{28} = 223.6$ kilos. of silica, while $267 - 120.2 = 223.6 = 22.2$ kilos. of oxygen goes into the gases. Another, and equally logical procedure, is to start with the 20.2 kilos. of silicon in the pig iron, which must have required $20.2 \times \frac{60}{28} = 42.4$ kilos. of silica to furnish it, yielding $42.4 - 20.2 = 22.2$ kilos. of oxygen to the gases, and leaving $267.0 - 42.4 = 223.6$ kilos of silica unreduced to go into the slag.

The 12.0 kilos. of manganese in the pig iron would be reduced from $12.0 \times \frac{87}{55} = 19.0$ kilos. of MnO^2, furnishing, therefore, 7.0 kilos. of oxygen to the gases, and leaving 77.8 — 19.0 = 58.8 kilos. of MnO^2 to go into the slag as MnO. The molecular weights of MnO and MnO^2 being respectively 71 and 87, there is $58.8 \times \frac{71}{87} = 48.0$ kilos. of MnO going into the slag, while 10.8 kilos. more of oxygen will be supplied to the gases, making altogether 10.8 + 7 = 17.8 kilos. of oxygen given up by the MnO^2. The 4.1 kilos. of Cu^2O contain 3.6 kilos. of copper, all of which enters the pig iron and contributing 0.5 kilos. of oxygen to the slag.

The 38.4 kilos. of CaO must supply enough Ca to form CaS with the S of the SO^3. The latter quantity is $26.3 \times \frac{32}{80} = 10.5$ kilos., to supply which there is needed $10.5 \times \frac{40}{32} = 13.1$ kilos. of calcium. The latter will be supplied by $13.1 \times \frac{56}{40} = 18.3$ kilos. of CaO, furnishing 5.2 kilos. of oxygen to the gases, and leaving 38.4 — 18.3 = 20.1 kilos. of CaO unreduced to go into the slag. The MgO in the ore goes directly into the slag. The oxygen of the SO^3 goes into the gases. The 5 per cent of water of the whole quantity of ore charged goes into the gases as vapor.

Flux.

The flux is used for the purpose of making a fusible slag with the slag-forming ingredients contributed by the ore and fuel. If we consider the distribution of ore and fuel given in the preceding illustrations we see that the chief material to be fluxed is silica, with smaller quantities of FeO, MnO, CaO, MgO and CaS. The cheapest and most available material to flux silica is limestone, the slag formed being a silicate of lime, magnesia and alumina, with CaS and smaller quantities of other basic oxides. We will not discuss at present the considerations governing the amount of flux used, since this is a calculation requiring separate treatment, as the proper working of the furnace depends fundamentally upon it. We may remark here that enough flux must be used to make an easily fusible fluid slag, rich enough in lime, magnesia or alumina to carry away satisfactorily the bulk of the sulphur, and so produce good pig iron.

The flux usually contains CaO, MgO, Al^2O^3, SiO^2, FeO, CO^2 and H^2O. Its H^2O and CO^2 are driven off in the upper third of the furnace, and may be put down as going as such into the gases. The FeO may be reduced if the slag is very clean, but under ordinary conditions may be put down as all going into the slag, unless in quite large amount, because the iron in ore and fuel usually supplies the total weight of iron in the pig iron. The silica and alumina may be carried over bodily into the slag. The magnesia can be put at once into the slag, but the lime cannot in many cases be treated that way, because quite frequently some is needed to supply calcium for the sulphur of the fuel. In the fuel previously illustrated, for instance, there is not enough CaO present to furnish Ca for the S, whence it follows that some CaO from the flux will be needed to make up the deficit. We may, in such a case, either consider all the CaO of the fuel to form CaS with part of the sulphur, and then take enough CaO from the flux to unite with the remainder. Or, it is equally permissible to take all the CaO necessary to furnish Ca to all the sulphur of the fuel, and to let the CaO of the fuel figure as passing entirely into the slag. The latter requires a little less calculation.

Illustration: A blast furnace receives 503 kilos. of limestone flux per metric ton of pig iron made, which contains CaO 29.08 per cent., MgO, 2.05; SiO^2, 3.07; Al^2O^3, 2.67; FeO, 0.42; CO^2, 42.66; H^2O, 0.50 per cent. Assume 8.1 kilos. of sulphur in the fuel, for which the flux must provide calcium. Required the distribution of the flux, assuming it to make no dust:

	Charge.	Pig Iron.	Slag.	Gases.
Flux	503.0 kg.			
CaO	149.3 "		[illegible] 26.3	[illegible] 1.5
MgO	10.3.1 "		MgO 10.3.1	
SiO^2	15.4 "		SiO^2 15.4	
Al^2O^3	13.4 "		Al^2O^3 13.4	
FeO	2.1 "		FeO 2.1	
CO^2	214.6 "			CO^2 214.6
H^2O	2.5 "			H^2O 2.5

The only calculation needed above is that 8.1 kilos. of sulphur require $8.1 \times \frac{40}{32} = 10.1$ kilos. of calcium, which would be furnished by $10.1 \times \frac{56}{40} = 14.2$ kilos. of lime, leaving 4.1 kilos. of oxygen to go into the gases and 135.1 of lime to go into the slag.

Blast.

The remaining item needed to complete the balance sheet is the amount of blast. This may be roughly estimated by obtaining the piston displacement of the blowing engines, and assuming a coefficient of delivery into the furnace. This is very rough, because the efficiency is not known, and may vary anywhere between 0.5 and 0.95. Another rough approximation may be obtained by observing the pressure of the blast, its temperature, the back pressure in the furnace, and knowing the area of the tuyeres, and assuming a coefficient of contraction of the hot air jet as it emerges from the tuyeres. Here, again, are several uncertain factors, and the coefficient may vary between 0.9 and 0.98. Calculations on this basis are very rough.

The only satisfactory way to determine the blast is to carefully analyze the gases, determining carefully all the carbon, oxygen and nitrogen which they contain. Since the carbon comes only from the charges, the amount of gas produced per unit of pig iron made becomes known, and thence the oxygen and nitrogen contained in them. These, minus the oxygen and nitrogen coming from the solid charges, leave the oxygen and nitrogen which must have come from the blast. The oxygen in the blast, minus $\frac{3}{10}$ the nitrogen, gives the oxygen entering as water vapor; but this last calculation is not so satisfactory as to observe the atmospheric conditions, and calculate the air and moisture on the basis of the contained nitrogen.

The blast contains oxygen, nitrogen and moisture. All its constituents pass into the gases, being put down as so much oxygen, nitrogen and hydrogen. Just how much of that hydrogen gets into the gases as free hydrogen and how much as water vapor is not known. Argon and other rare gases in the blast are counted and treated as nitrogen. The carbonic acid of the air is present relatively in such a small amount that it can be neglected, as far as all ordinary calculations are concerned.

Problem 51.

A blast furnace at Hertang, Sweden, runs on ore briquettes made by pressing and calcining fine concentrates. The analyses of briquettes, charcoal and limestone flux are as follows (see Journal Iron and Steel Institute, I., 1904):

	Briquette.	Limestone.	Charcoal.
Fe^2O^3	85.93	0.18	0.82
FeO	3.96		C 80.37
SiO^2	5.30	3.14	0.19
MnO	0.68		N 0.8
Al^2O^3	0.76	0.32	O 3.54
CaO	2.23	53.74	0.89
MgO	0.97	0.17	0.10
P^2O^5	0.006	0.006	0.0065
S	0.010	0.001	0.0150
Cu	0.007	CO^2 42.42	H^2O 14.04
			K=O 0.50

The pig iron contains phosphorus, 0.012 per cent; sulphur, 0.007; manganese, 0.025; silicon, 0.60; carbon, 2.70; iron, 96.656. There is used in charging the furnace:

Briquettes	1,190 pounds
Limestone	90 "
Charcoal	530 "

And the fuel consumption is 682 pounds of charcoal per 1,000 pounds of pig iron made.

The gases at the throat (dried) analyze: N^2, 57.3 per cent; CO, 23.1; CO^2, 14.8; H^2, 4.3; CH^4, 0.5 (Rinman). Assume blast dry. Dust in gases neglected. Required: (1) A balance sheet of materials entering and leaving the furnace, per 1,000 pounds of pig iron made. (2) The percentages of iron, manganese, silicon, sulphur and phosphorus going into the furnace, which go into the pig iron.

Solution: (1) Balance Sheet

Per 1000 of Pig Iron Made.

	Charges		*Pig Iron*		*Slag*			*Gases*
Ore, 1580.2 lbs.	Fe^2O^3	1314.9	Fe 920.4		....		O	394 5
	FeO	60.6	Fe 46.2		FeO 1.2		O	13.2
	SiO^2	84.2	Si 6.0		SiO^2 69.6		O	8.6
	MnO	9.6	Mn 0.25		MnO 9.3		O	0.1
	Al^2O^3	11.6	....		Al^2O^3 11.6		....	
	CaO	34.1	....		CaO 34.0		O	0 03
	MgO	14.8	....		MgO 14.8		....	
	P^2O^5	0.092	P 0.04		....		O	0.05
	S	0.153	S 0.07		CaS 0.19		....	
	Cu	0.11	Cu 0.11		....		....	
Limest. 115.8 lbs.	Fe^2O^3	0.2	....		FeO 0.2		O	0 02
	SiO^2	3.6	....		SiO^2 3.6		....	
	Al^2O^3	0.4	....		Al^2O^3 0.4		....	
	CaO	62.2	....		CaO 62.2		O	0.00
	MgO	0.2	....		MgO 0.2		....	
	P^2O^5	0.007	P 0.003		....		O	0.00
	S	0.001	....		CaS 0.00		....	
	CO^2	49.1	....		....		CO^2	49.1
Charcoal, 682 lbs.	C	547.7	C 27.0		....		C	520.7
	N	0.5	....		....		N^2	0.5
	O	24.1	....		....		O	24.1
	Fe^2O^3	2 2	....		FeO 2.0		O	0.2
	SiO^2	1.3	....		SiO^2 1.3		....	
	CaO	5 1	....		CaO 5.9		O	0 06
	MgO	0.7	....		MgO 0.7		....	
	P^2O^5	0 046	P 0.020		....		O	0 03
	S	0 116	....		CaS 0.25		....	
	K^2O	88 4	....		K^2O 3.4		....	
	H^2O	95 8	....		....		H^2O	95 8
Blast	O	557.7	....		....		O	557 7
	N^2	1859 1	....		....		N^2	1859 1
	Totals	4.744 0	1,000.0		220 8			3.543 7

(2) The total iron in the charge is 969.2 kilos., while that in the pig iron is 966.6; the efficiency of the reduction of iron is therefore 99.7 per cent.

The total manganese in the charge is $9.6 \times \frac{55}{71} = 7.4$ kilos., of which only 0.25 gets into the pig iron, or 3.4 per cent.

The total silica charged is 89.1 kilos., representing 41.6 kilos. of silicon, of which 6.0 kilos. enters the pig iron, or 14.4 per cent.

The sulphur charged is 0.270 kilos., of which the pig iron contains 0.07, or 25.9 per cent.

The phosphorus charged is 0.063 kilos., while the analysis of the pig iron shows in it 0.12 kilos. It is thus evident that all the phosphorus goes into the pig iron; for while the analysis shows more phosphorus in the pig iron than was put into the furnace, yet the divergence is evidently due to segregation or concentration of phosphorus in the sample taken, and the practical conclusion is that all the phosphorus in the charge finds its way into the pig iron.

NOTES ON THE BALANCE SHEET.

The Fe^2O^3 of the ore is assumed all reduced, because the 920.4 kilos. of iron in it is less than the 966.6 kilos of iron known to be in the 1,000 kilos. of pig iron from its analysis. The FeO, however, cannot be assumed all reduced, because it would furnish 47.1 kilos. of iron, and there is only 966.6 − 920.4 = 46.2 kilos. of iron yet to be supplied. We, therefore, put down 46.2 kilos. of iron as going to the pig iron, thus furnishing all the iron in the pig iron, and leaving 0.9 kilos. of iron to go over into the slag as 1.2 kilos. of FeO. Having thus allowed for all the iron in the pig iron, the Fe^2O^3 in the limestone and fuel must be assumed as passing entirely into the slag as FeO.

The 6 kilos. of silicon in the pig iron is put down as coming entirely from the SiO^2 of the ore, of which 14.6 kilos. is thus used up, leaving 15.6 kilos. to go into the slag. The SiO^2 of flux and fuel must then be regarded as passing entirely into the slag.

The 0.25 of manganese in the pig iron comes from the MnO of the ore, requiring 0.35 of MnO, and leaving 9.3 of MnO to go into the slag.

The Al^2O^3 and MgO of ore, flux and fuel go bodily into the slag.

The sulphur in the ore, 0.153 kilos., is more than enough to supply the 0.07 kilos in the pig iron. We, therefore, put down 0.07 kilos. as going into the pig iron, supplying all the latter contains, and calculate the remaining 0.083 kilos. to CaS going into the slag. The CaO necessary to furnish this calcium is 56 for every 32 of sulphur (CaO = 56, S = 32), or 0.14 kilos., which, therefore, must be deducted from the 34.1 kilos. of CaO present in the ore. The oxygen of this 0.14 kilos. of CaO finds its way into the gases.

The 0.092 kilos. of P^2O^5 present in the ore contain only 0.04 kilos. of phosphorus, and since the pig iron contains, from its analysis, 0.12 kilos., we may assume all of this going into the pig iron. The same remarks are true of the P^2O^5 in flux and fuel; altogether, they come somewhat short of supplying all the phosphorus in the pig iron, and are, therefore, considered as completely reduced. The copper goes entirely into the pig iron, although not given in the analysis.

The Fe^2O^3 of the limestone must be transferred entirely as FeO to the slag, since all the iron needed for the pig iron has been already provided. The same is true of the Fe^2O^3 of the fuel; and an analogous statement applies to the SiO^2 and sulphur of both flux and fuel. The sulphur of the fuel does not produce an amount of CaS which counts in significant figures, and the CaO required is likewise insignificant, as is also the oxygen thus furnished the gases. In such cases, instead of ignoring the item altogether, or putting down wholly insignificant quantities, the amounts are expressed as 0.00, denoting no significant amount.

The fixed carbon of the fuel, only, furnishes the carbon in the pig iron, the rest going into the gases. The blast is calculated as follows:

Carbon in CO^2 of flux $= 49.1 \times \frac{12}{44}$	$= 13.39$ kilos.
Carbon in gases from fuel	$= 520.70$ "
Carbon in gases altogether	$= 534.09$ "
Carbon in 1 cu. meter of gas $(0.231 + 0.148 + 0.005) \times 0.54$	$= 0.20736$ "
Volume of gas per 1,000 of pig iron $= 534.09 \div 0.20736$	$= 2575.6$ m^3
Nitrogen in this gas $= 2575.6 \times 0.573$	$= 1475.9$ "
Weight of nitrogen $= 1475.9 \times 1.26$	$= 1859.6$ kilos
Nitrogen from fuel	$= 0.5$ "
Nitrogen from blast	$= 1859.1$ "
Oxygen from blast $= 1859.1 \times 0.3$	$= 557.7$ "

Chemistry of Electric Incandescent Lamps.

Remarkable results are reported from Europe concerning the new Kuzel lamp (see, for instance, *Electrical World*, 1900, Vol. XLVII., pp. 354 and 487). It seems to bring into prominence a chapter of physical chemistry which was so far only of theoretical interest, namely, colloids. Kuzel brings metals and metalloids of high melting point into the colloidal state in water, where they become plastic without any binding material. They can then be easily moulded like clay, and when forced through a die and dried can be employed as filaments. Eutectic alloys are also used.

The Use of Electrolytic Hypochlorite as a Sewage Sterilizing Agent in the United Kingdom.

By John B. C. Kershaw, F. I. C.

Introduction.

The electrolysis of solutions of the alkali metal chlorides always produces chlorine at the anode, and the hydrate of the base (sodium, potassium or magnesium) at the cathode. In the electrolytic alkali processes, the aim of the inventor has been to effect removal of these products before they can react one with the other. In the electrolytic bleaching and disinfecting processes the chlorine and the caustic hydrate are allowed to react, in order that the chemical changes represented by the following equations may occur:

1. $3NaOH + 6Cl = 3NaOCl + 3HCl$, sodium hypochlorite and hydrochloric acid.

2. $3NaOH + 3HCl = 3NaCl + 3H_2O$, sodium chloride and water.

If the temperature be allowed to rise above a certain point a further chemical action will occur and chlorate will be produced. It is, therefore, necessary in all electrolytic bleaching and disinfecting processes to pay great attention to the temperature, as chlorates are useless for bleaching purposes.

The production of hypochlorite solutions for bleaching and disinfecting purposes by means of electrolysis has undergone little development in the United Kingdom, and only two types of cell are known to the writer to be at present in operation, namely: the Volgelsang and the Atkins cells.

Early in the nineties a considerable amount of money was expended, however, upon experimental trials with the Hermite cell for sewage purification and sterilization, and much attention was given to the results of these trials. The failure to establish this method of sewage treatment upon a commercially sound basis led to the bankruptcy of the firm financing these trials, and to a collapse of interest in all processes and cells for producing, by aid of electrolysis, a solution containing free chlorine for bleaching or disinfecting purposes. It is only lately that attention has again been given to this subject of sewage disinfecting and sterilization in the United Kingdom. The proposal of the medical officer of health for Poplar, to use electrolytic hypochlorite for disinfecting purposes within the Borough, has aroused considerable interest.

The low price at which hypochlorite of lime, or "bleach," is now selling in the United Kingdom (£4.10.0 per ton) is likely, however, to hamper the extension of these processes, for it is more than doubtful whether any of them can compete in price with the older bleaching agent. For certain purposes, hypochlorite of soda is much to be preferred to hypochlorite of lime, the active agent in bleaching powder; and in these cases the higher cost of the former, even when produced by the electrolysis of brine, is unlikely to retard its utilization as soon as a cheap, reilable and durable electrolytic cell for producing it is placed upon the market. The Hermite cell and process in its original form did not fulfil these conditions, and the same judgment must be passed upon many of the patented forms and processes which have succeeded it. Even with cheap power and a durable and reliable cell, electrolytic hypochlorite is unlikely to take the place of bleaching powder for ordinary bleaching purposes, until the price of the latter has advanced to over £6 per ton, and, therefore, the development of this industry is somewhat doubtful in countries where bleach is manufactured in large quantities, by either the old Le Blanc or newer electrolytic processes.

The following are details of the hypochlorite cells which have been patented and worked in the United Kingdom:

Charles Watt's Hypochlorite Cell.

The remarkable omnibus patent of Charles Watt (No. 13,755, of 1851) contains a reference to the method by which electrolysis may be applied to the formation of hypochlorite and chlorates in the following words:

"A warm solution, and nearly saturated, of the chloride of potassium or other chloride to be operated upon, is prepared and placed in a vessel furnished with two electrodes, and in Fig. 1 (Fig. 4 in the patent specification) I have shown a vessel furnished with a pair of electrodes, constructed in such a manner as to be convenient for carrying this part of my invention into effect.

"In this figure *B* is the interior of the vessel, which is furnished with a steam-jacket *A*, with connection to a boiler (*a*) and waste-steam tap *dg* thermometer; *EE* electrodes placed

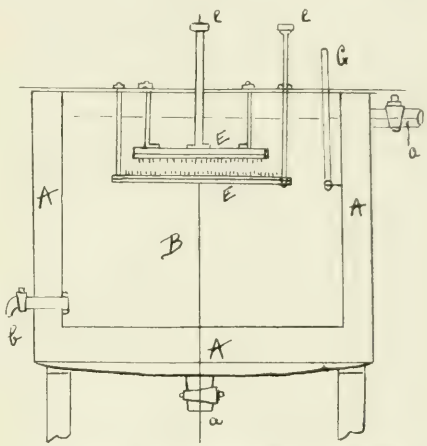


Fig. 1.—Vertical Section of Watts Chlorate Cell.
(Reproduced from the patent.)

over each other, the lower one being intended for the elimination of the chlorine, and the upper one for the elimination of the alkali; *ee*, attachments of the electrodes. I have found it advantageous to add to the solution of the chloride a portion of free alkali or alkaline earth equal to about one-tenth of the quantity of the saline matter to be decomposed. The connection is now to be completed between the apparatus which supplies the electric current or currents and the vessel, and if the object be the formation of chlorate, I maintain the action until from one-half to two-thirds of the original saline matter shall be converted into a chlorate. The agency of the electricity first decomposes the chloride, the chlorine being eliminated at one of the electrodes, and the alkaline or earthy metallic base at the other electrode. The metal thus set free from its combination with chlorine combines with oxygen, and so becomes converted into an oxide, the oxygen being obtained from the water of the solution, a portion of which is therefore decomposed and hydrogen gas evolved; and this gas may be allowed to pass away or, if desired, collected in another vessel. The liberated chlorine will, when it is set free, combine with a portion of alkali or alkaline earth in solution, and a hypochlorite will be formed. The hypochlorite thus formed will, by the continued action of heat, be resolved partly into a chlorate of the alkali or earth and partly into a chloride of the metallic base; and the chloride will again be subjected to decomposition, and a hypochlorite formed in like manner as herein before described, and this action or succession of decompositions and changes will go on until I stop the process, which I prefer to do when from one half to two thirds of the saline matter has been converted or formed into the required chlorate. The quantity of chlorate thus obtained, or the greater portion of it being then separated by crystallization from the solution, the undecomposed saline matter remaining

in solution may, with an additional portion of the same saline matter, be subjected to the same process, for the convertion of a portion of it into a chlorate of the alkali or earth, as already described.

"If I desire to produce a hypochlorite of the alkali or earth, I merely keep the vessel warm (say from about 100° to 120° F. thermometer) without applying the higher temperature, which it is well known is requisite to produce a chlorate from a hypochlorite; and I continue the process until as much of the saline matter has been converted into a hypochlorite as may be required for the purpose to which the solution is to be applied. This mode of forming a hypochlorite of the alkalies and alkaline earth may be used for preparing a bath for the purpose of bleaching various kinds of goods, and the bath may be strengthened from time to time by the action of the electric current or currents."

The fourth section of Watt's 1851 patent relates to the use of electrolysis for separating and refining metals; but in the above extract Watt shows that he had a clear grasp of several of the important details, upon which the formation of hypochlorite and chlorate in the electrolytic cell depends.

Of course, at the date of Watt's patent no industrial application of this method of producing hypochlorite was possible, and it remained for Hermite to recall attention to the subject in the years 1884-1894.

The Hermite Hypochlorite Cell.

The Hermite cell contains a horizontal spindle, bearing a large number of discs made of zinc plates, capable of rapid rotation by means of a pulley wheel. These zinc discs are the cathodes of the cell, while an equal number of pieces of platinum gauze, fixed in lead or glass frames, hang between each of the consecutive pairs of zinc discs and act as anodes. The rotation of the spindle bearing the discs and the rapid circulation of the electrolyte cause a complete mixture of the solution, and little chlorine escapes as gas from the cell, the greater portion combining with the hydrate to form hypochlorite, in accordance with the chemical equation given above.

In Hermite's original patents a mixture of magnesium and sodium chloride solution was recommended as the electrolyte for use in this cell. The electrolytic decomposition led to the production of insoluble magnesium hydrate at the cathodes, and the zinc discs speedily became coated with a deposit of

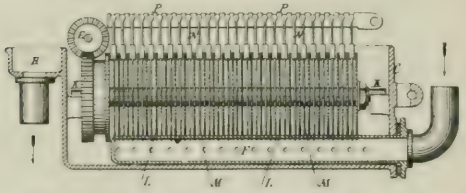


FIG. 2.—LONGITUDINAL SECTION OF HERMITE HYPOCHLORITE BELT.

this compound. Fixed scrapers did not satisfactorily remove it from their surfaces, and the E.M.F. necessary to work the process was permanently increased, owing to the resistance offered by this film of hydrate on the zinc plates. The writer believes that on this account the procedure was altered, and that sodium chloride was used alone in preparing the electrolyte for the cells.

The Hermite cell, as described above, was used at Worthing, Lytham, Ipswich and at Netley Hospital, Southampton, for sewage sterilization in the years 1890-1894. At Ipswich and the other places sea water was used as electrolyte.

One of the forms of electrolyzer used (see Figs. 2 and 3) possessed a pair of horizontal spindles *KK*, each carrying a large number of the zinc discs *LLL*. The platinum gauze *MMM* was carried by a framework of glass rods. This form of electrolyzer produced 250 grms. chlorine, as hypochlorite, per hour. The electrodes were in parallel, and an EMF of about 6 volts was requisite to work the cell to the best advantage.

A modified form of the electrolyzer had been designed for the use of currents having a higher EMF than 6 volts. In this a number of the zinc and platinum pans were coupled in series. A full description of the apparatus will be found in British patent No. 6,497, of 1894.

The energy required to produce 1 kg. chlorine in the form of hypochlorite with this apparatus was given as .48-hp. day, or 11.4-hp. hours. At one-tenth of a penny per EHP hour, this represented a cost of 1.14d. = 2.28 cents per kg. of active chlorine. The solution obtained by the electrolysis of sea-water contained both sodium and magnesium hypochlorites. The latter quickly decomposed into magnesium hydrate and hypochlorous acid when mixed with the sewage effluent or when allowed to stand.

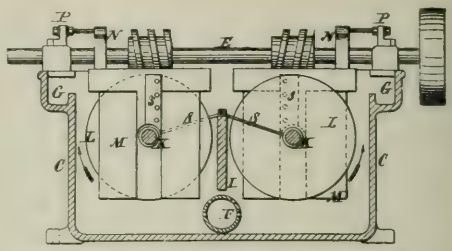


FIG. 3.—END SECTION OF HERMITE HYPOCHLORITE BELT.

This solution, according to M. Hermite, was a powerful deodorizer and an efficient sterilizing agent. The reports of the numerous experiments that had been carried out respecting its action on sewage were, however, very conflicting. With a clear effluent there appeared to be no doubt of its deodorizing and sterilizing properties when used in sufficient quantity; but upon solid faecal matter its effect was much less satisfactory. The most important independent criticism of the process, as applied to sewage, was made by Sir H. Roscoe and Mr. Lunt. They found that the weaker solutions of hypochlorite were exceedingly unstable, while the stronger solutions containing 1.0 gram active chlorine per liter, though much more stable, required a considerable loss of energy in the electrolyzer in order to obtain them. They, nevertheless, stated that the solution containing .25 gram active chlorine per liter was an excellent deodorizer.

The experiments made with the process at Ipswich in 1895 led Mr. Napier, the borough analyst, and Dr. Elliston, the medical officer, to report favorably upon the process as applied to the sewage of their town, the desire here being merely to arrest decomposition of the organic matters until the sewage had been carried out to sea.

The Ipswich plant, which cost $12,000, was started in May, 1895. A 30-hp. engine drove dynamos yielding 600 amps. at 28 volts; four electrolyzers, coupled in series, yielded 2.4 kgs. chlorine per hour. The electrolyzers were equal to providing 1 gram chlorine per head of the population per 24 hours, and the cost of running the installation was stated to be $1,680 per annum, or 3 cents per head per annum.

As regards the failure of the Hermite sewage disinfecting process, the following criticism, published by the present writer in 1898, is still of interest:

"As regards the use of the Hermite fluid for deodorizing sewage and for disinfecting purposes generally, the future is less certain. In many cases, where the sewage is turned directly into rivers or into the sea, there is at present no obligation to deodorize it, and the adoption of the Hermite treat-

ment would simply be resisted as an unnecessary expenditure. In cases where treatment of the sewage is absolutely essential the Hermite process fails, in that it is only successful when applied to the clear effluent, and the nuisance arising from the decomposition of the solid foecal matters is in no way abated. As an alternative procedure to the use of chloride of lime and carbolic acid, for the mere purpose of lessening the effluvia from the main sewers, it may find adoption; but it certainly cannot be regarded as the solution of what is still one of the

FIG. 4.—VOGELSANG HYPOCHLORITE CELL AT NOTTINGHAM.

greatest problems of the age—the production of innocuous and useful materials from the sewage of our great cities."

The Vogelsang Hypochlorite Cell.

The Vogelsang cell resembles in its main features the Hermite cell; but instead of working the intermediate electrodes of the cell in parallel, these are worked as secondary electrodes in series, and only the two end electrodes are connected to the terminals of the current conductors. These secondary electrodes are coated with platinum, in order to resist the action of the free chlorine and hypochlorite. By varying the number of intermediate electrodes a cell can be arranged to take a current of either 65 or 110 volts, and therefore can be adapted to the ordinary lighting supply of electricity.

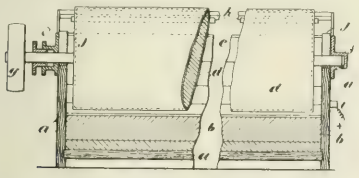


FIG. 5.—LONGITUDINAL SECTION OF ATKINS CELL.

An electrolytic cell of the Vogelsang type, taking 85 amps. and 115 volts, is said to produce bleaching liquor sufficient to bleach 4,000 pounds cotton yarn in a 10-hour working day. This weight requires 20 kgs. active chlorine, and the yield of the Vogelsang type of cell, *if the above figures can be relied upon*, is therefore 1 kg. active chlorine per 6.6-EHP hours, or an energy efficiency of 38.5 per cent.

The Vogelsang patents for electrical bleaching have been exploited in the United Kingdom by the Electrical Bleaching Co., of Nottingham, and a small experimental plant for demonstration purposes has been in operation in Nottingham since 1903. Fig. 4 is a view of this small plant, which is utilized chiefly for bleaching fine work and lace of Nottingham manufacture. In 1904, however, the exploiting company underwent financial reconstruction, and a new company, with headquarters in Manchester, and a capital of £75,000 was formed, to purchase the assets and to carry on the industrial development of this cell and process.

The Atkins Hypochlorite Cell.

This cell is covered by patent No. 5,597, of 1901, and is being exploited in the United Kingdom by a company registered as "Oxychlorides, Limited," with a capital of £150,000 and offices in London.

The following is a description of the Atkins cell and process for producing hypochlorites. (See Figs. 5 and 6.)

A wooden trough *a* of semi-circular cross-section forms the cell, and is lined first with lead and then with carbon blocks shaped to fit the wooden vessel; these blocks form the anode of the cell. A wooden cylinder *c*, also lead covered, revolves on bearings placed at the ends of this trough, and forms the cathode of the cell. A copper collecting ring *g* and brushes at one end of the bearings supply the electrical connection for this revolving cathode. The trough is filled with a 10 per cent solution of sodium chloride, and the electrolysis of this solution proceeds in the ½-inch space which separates anode and cathode. Figs. 5 and 6 show diagrammatic, longitudinal and cross-sections of the Atkins cell. The cell with which the preliminary experiments have been conducted is 10 feet in length, and the cathode cylinder is 2 feet in diameter.

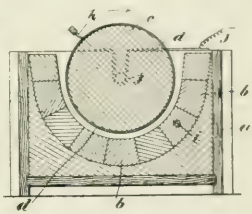


FIG. 6.—CROSS-SECTION OF ATKINS CELL.

A current of 1,500 amps. at 3.5 to 4 volts is used to operate this cell, and a current density of 50 amps. per square foot is reported to be obtained. This represents a cathodic area of 30 square feet for the submerged half of the cylinder.

The revolution of the cathode is designed to lessen polarization and reduction by the hydrogen liberated at the negative pole, and it is said to be effective in attaining this result. No figures are given for the electrical efficiency of the appartus, and in the absence of these it is impossible to compare its yield of available chlorine with that of other cells.

The Atkins cell has been worked upon a small scale for demonstration purposes at Forest Hill, near London, and towards the end of 1904 considerable attention was aroused by trials of the solution obtained from it for sewage sterilization purposes at Guildford, in Surrey. Three tanks were constructed at this place, each 50 feet long by 30 inches wide, with a total capacity of 4,000 gallons, and into these tanks the Guildford sewage, with the requisite amount of the hypochlorite solution, was pumped daily. Chemical and bacteriological tests were made, both with the untreated sewage and with the effluent from these tanks, on behalf of "Oxychlorites, Limited," by Dr. Rideal, of London, and the results obtained are held to be favorable to this method of sewage sterilization. An addition of 3¼ gallons of hypochlorite solution per 1000 gallons of sewage is stated to be sufficient to nearly exterminate all the more dangerous bacteria of the sewage, while with the effluent from septic tanks a greatly reduced proportion of the hypochlorite solution suffices to kill, or render innocuous, the pathogenic organisms. No details as to the cost of this method of treatment have, however, been published, and in the absence of these figures no judgment can be passed upon the future of the process. In the light of the previous failures with the Hermite process at Ipswich and other places, with the Wolf process at Maidenhead, and of similar processes in France and America, the future prospects of this application of electrolytic hypochlorite to sewage sterilization cannot be considered very hopeful.

The Woolf Hypochlorite Cell.

This cell, patented by an American engineer named Woolf,

was operated at Havana, in Cuba, at the close of the Spanish-American war, and was introduced into England by "The Electrozone Co.," in 1895, and an experimental plant for treatment of sewage with the solution produced was erected at Maidenhead in 1896.

As worked at Havana, the Woolf electrolyzer consisted of a large circular wooden vat 3 feet in depth, containing 417 anodes and 418 cathodes, placed ½ inch apart. The anodes were of platinum-iridium alloy, the cathodes were of zinc. Sea-water was used as electrolyte. At Havana 216 kw. was said to have yielded 935 kgs. active chlorine per 24 hours, equivalent to 7.5 EHP-hours per kg. of active chlorine. This equals an energy efficiency of 31.5 per cent.

Although the trials at Maidenhead attracted much attention and proved satisfactory from a bacteriological point of view, they were not continued, owing to the cost of the process, and the company financing them ultimately went into liquidation.

Fixation of Atmospheric Nitrogen.*

By Prof. Philippe A. Guye.

Nitrogen, bound in form of ammonium salts and of nitrates, represents one of the vital elements of modern civilization. Besides numerous applications in the chemical arts (for instance, the use of ammonia in the production of sodium carbonate), nitrogen compounds are of fundamental importance in the industries of explosives and of fertilizers.

In view of the fact that the natural supply of Chili saltpeter is by no means inexhaustible, the importance of the problem of utilizing the atmospheric nitrogen for the production of useful nitrogen compounds is evident.

Atmospheric nitrogen is practically inexhaustible. The quantity of nitrogen contained in 1,000,000 tons of Chili saltpeter—which represents the annual consumption of Europe—is equal to the amount of nitrogen contained in the atmosphere covering 2 hectars, or 5 acres, of the earth's surface.

Two principal methods—both of which are closely connected with the progress of electrochemistry during the last decade—appear at the present time specially promising for the commercial fixation of atmospheric nitrogen. One of these methods produces calcium cyanamide (German trade name Kalkstickstoff, French chaux azotée = nitrogenated lime), by a reaction between calcium carbide and atmospheric nitrogen. The other method is based on the combination of the nitrogen and oxygen of the air by means of the electric arc.

I. Calcium Cyanamide.

Dr. Frank observed that barium carbide BaC_2, when heated to a high temperature, binds the nitrogen almost quantitatively, according to the equation

$$BaC_2 + N_2 = Ba(CN)_2$$

thus yielding barium cyanide.

When Dr. Frank tried to apply this reaction to calcium carbide, he was surprised to find that the quantity of cyanide formed was much less than the theoretical amount. Further study revealed the fact that the reaction is different from that which occurs in the case if barium carbide, and that in the case of calcium carbide it is represented by the equation

$$CaC_2 + 2N = CaCN_2 + C$$

The carbide gives off half of its carbon and changes not into cyanide, but into calcium cyanamide.

When the cyanamide is treated with water, under proper conditions, ammonia is set free according to the reaction

$$CaCN_2 + 3H_2O = CaCO_3 + 2NH_3$$

When calcium cyanamide is distributed in the ground, the same reaction goes on more or less slowly, and it is, therefore, easily understood that cyanamide has given quite satisfactory results when used directly as a fertilizer.

On a commercial scale calcium cyanamide may be prepared by passing nitrogen gas over powdered calcium carbide, heated to a temperature of about 800° C. The reaction then proceeds without great expenditure of fuel, since it is strongly exothermic. This is the principle of the method used at present.

However, it has also been proposed to produce calcium cyanamide by treating lime, carbon and nitrogen at the very high temperature of the electric furnace. Whatever may be the method of operation the atmospheric nitrogen must be first separated from the oxygen, with which it is associated in the atmosphere.

Theoretically, the mixture of calcium cyanamide and carbon, produced by the reaction given above, should contain about 30 per cent nitrogen. As a matter of fact, the content of nitrogen in the raw calcium cyanamide product is less than this amount, either on account of the impurities in the calcium carbide or of the changes which it undergoes in course of operation. According to Frank the content of nitrogen varies between 14 and 22 per cent. The product of recent manufacture contains about 20 per cent.

Without entering into details which the industry, of course, wishes to keep secret, there can be no doubt that the production of calcium cyanamide is closely connected in principle with the manufacture of calcium carbide in the electric furnace. The necessary quantities of energy are about the same If we want to estimate the cost of production of calcium cyanamide, containing about 20 per cent of nitrogen, for instance, and if we base our estimate on the cost of production of calcium carbide at 140 francs ($28.00) per ton—which figure is actually realized in very favorably situated electrochemical plants[1]—the kilogram of fixed nitrogen will cost about 0.70 franc (14 cents), which makes it on a par with the nitrogen fixed in ammonium salts if all costs of manufacture are taken into account.

On the other hand, experimental trials in agriculture have given quite satisfactory results with calcium cyanamide as fertilizer. Competent authorities do not fully agree as to the relative value of calcium cyanamide in comparison with ammonium salts and nitrate. It seems, however, that in various respects calcium cyanamide occupies a position intermediate between these two products.

Up to the present time calcium cyanamide was furnished by an experimental station in Berlin. The first more important plant, utilizing 3,000 hp., will be opened this year in Italy Other similar products are also being studied.

II. Nitric Acid.

Calcium cyanamide solves only one of the economical problems of the nitrogen industry. It is a chemical fertilizer promising to be able to replace, at least partly, the Chili saltpeter and, perhaps wholly, the ammonium salts. If the present prospects should realize it will form a partial solution of the problem.

But Chili saltpeter also plays an important part as starting material for the manufacture of nitric acid, the largest part of which is consumed for the production of explosives and of powder for war, mining and general engineering purposes. In Europe the amount of nitrate consumed for these purposes represents about 20 per cent of the total import from Chili Moreover, statistics proves that the production of nitric acid is continually increasing.

Two ways of utilizing the atmospheric nitrogen are open for this purpose

(1) The first is to fix the atmospheric nitrogen in some way as ammonia—which may be done, for instance, by the reaction of calcium cyanamide with water—and it becomes then necessary to find an economic process for oxidizing ammonia to

* A paper read before the Section of Natural Science. Slightly abridged.

[1] The figure is given as manufacturing cost of the metric ton, and not as selling price, and refers presumably to some places in Europe where water-power may be had at a very low rate, say $5 or $6 per h.p. year. In this country the results would be quite different. Ed.

nitric acid. In this respect it is interesting to note that the oxidation of ammonia is strongly exothermic. The reaction

$$H_3N + 4O = HNO_3 + H_2O$$

evolves theoretically 97 calories. It would, therefore, seem that under proper conditions this reaction should proceed spontaneously.

This explains why researches have been made by various investigators with a view of realizing this transformation. But up to the present time it seems that in spite of numerous trials, tending to discover effective catalyzers, a practical solution, capable of industrial application, has not been found.

(2) The second way for producing nitric acid from air has, as starting point, an observation of Cavendish (1784), who found that nitrogen and oxygen combine slowly under the action of electric discharges to form oxides of nitrogen which then change, by more or less complex chemical processes, into nitric acid, nitrous acid, or nitrates and nitrites. The fundamental reaction is, therefore, according to the happy expression of Prof. Crookes, a real "combustion" of nitrogen in oxygen. But we may add, in order to be precise, that this combustion is "slow and lazy," since it goes on only as long as the electrical energy continues to act and since it also stops as soon as the content of nitrogen oxides in the gas has attained a certain limiting value.

For about ten years attempts have been made to devise a process of manufacture of nitric acid on this basis. I do not intend to describe in detail the various methods which have been suggested, but will summarize the general conclusions which may be drawn from these researches. I should state, however, that the first attempt of this kind, made in Switzerland, was due to Aloys Naville in 1893, who proposed to me shortly afterward to study this problem in collaboration with him and Prof. C. Eug. Guye. The results of this work were taken up in 1896 by the Société d'Etudes Electrochimiques, in Geneva, and tests have been carried out continually since that time on a moderately large scale. Other trials in the same direction have been made by the Atmospheric Products Co., at Niagara Falls (Bradley-Lovejoy method[2]), the Groupe d-Initiative in Freiburg, Switzerland (method of Kowalski[3]), and the Actieselskabet det Norske Kvaelstof Compagni in Norway (method of Eyde and Birkeland[4]). Independently of these developments, which have a more or less industrial character, very interesting laboratory researches have been made by Crookes (1897), Lord Rayleigh (1897), McDougal and Howles (1900), Muthmann and Hofer (1903), Nernst (1904), von Lepel (1903) and others.

At the beginning of these researches contradictory results were often announced. It appeared as though the effects produced would be different, according to whether the arc is produced by alternating or direct current, or whether the arc is elongated or shortened. Some have recommended currents of small intensity, others currents of large intensity. The presence of water vapor in the air has sometimes been claimed to prove favorable, sometimes unfavorable. Even the form of the electrodes appeared to play a more or less important rôle. To sum up the combustion of nitrogen at the tempreature of the electric arc appeared to be subject to very peculiar, capricious and mysterical laws, and seemed to depend on new strange factors.

All the factors mentioned above are now known to be secondary and accessory. The detailed study of the problem has proven that the phenomenon is strictly governed by the fundamental laws of chemical dynamics.

This method of looking at the problem has not only the advantage of simplicity, it also furnishes valuable indications on the possibilities which may be realized, and so far there exists good agreement between theory and experimental results. The following discussion is based upon this point of view:

III. Chemical Phenomena Due to the Arc in Air.

The chemical phenomena which occur when an electric arc passes through atmospheric air, may be suitably discussed under three different headings: First, the initial reaction; second, the influence of temperature; third, the reverse reaction.

Initial Reaction.—At the high temperature of the electric arc in air, the molecules of nitrogen and oxygen dissociate into their atoms, which is followed by the combination of these atoms to form nitric oxide.[5] The reactions are, therefore,

$$N_2 = 2N \text{ and } O_2 = 2O$$

then

$$2N + 2O = 2NO$$

Like most chemical reactions, this reaction is incomplete. This means that for a given temperature and for given initial conditions of pressure, composition, etc., the reaction stops when the content of nitric oxide reaches a certain value.

The underlying reason of the stoppage of the reaction is its reversibility. The stoppage will take place when the above reaction—the combination of the N and O atoms to nitric oxide—and the reverse reaction—the dissociation of the nitric oxide into N and O atoms—hold each other in equilibrium; *i. e.*, when in the same time the same number of NO molecules are formed by one reaction as are dissociated by the opposite reaction. The reversibility is expressed by the equation

$$\frac{1}{2} N_2 + \frac{1}{2} O_2 \rightleftarrows NO$$

It is difficult to measure the content of the nitric oxide gas in the state of equilibrium, since in the presence of an excess of oxygen (which is always the case in these experiments) the nitric oxide changes rapidly into peroxide at temperatures below 500° or 600° C.

$$NO + O = NO_2$$

The results of analysis are, therefore, generally given in form of NO_2. But this does not modify the preceding conclusions.

Effect of Temperature.—The reaction proceeds the further the higher the temperature, other things being equal. The following figures were found by Nernst and controlled by calculation on the basis of the law of mass action:

Absolute Temperature.	*Per Cents by Volume of NO, Observed.*	*Per Cents by Volume of NO, Calculated.*
1811	0.37%	0.37%
2033	0.64	0.67
2195	0.97	0.98
3200	5.0	4.4

The time in which the limit of NO gas is formed; *i. e.*, the time in which the condition of equilibrium is reached at a certain temperature, is the shorter the higher temperature. We give some figures of the same author indicating the time during which the reaction is half completed; *i. e.*, the time in which half the above limiting values are reached. It is 100.0" at 1,540° and 3.5" at 1,737°.

It follows that a double advantage is derived for the combustion of nitrogen from the use of as high a temperature as possible. First, the content of nitric oxide is increased, and, secondly, the transformation occurs more rapidly.

It is true that these advantages are partly compensated by the fact that considerably more heat energy will be required at an elevated temperature, since the electric arc has to furnish the calories, raising to the desired temperature not only the

[2] Our vol. I., pages 20, 100, 380, 386, vol. II., page 250; vol. III., page 285.

[3] Our vol. I., page 462; vol. II., page 152.

[4] Our vol. II., pages 399, 507; vol. III., page 33.

[5] Some authors have assumed that nitrogen peroxide is directly formed. But this assumption is untenable in view of the experiments of Richardson, who found that the gas NO_2 dissociates completely into NO and $\frac{1}{2}O_2$ at a temperature between 550° and 700° C., which is much below the temperature of the electric arc.

nitrogen and oxygen which combine, but also the excess of the two gases which do not combine. If one makes a careful calculation, however, the larger expenditure of energy required at the higher temperature represents an expense which is more than counterbalanced by the better efficiency obtained thereby. The use of a high temperature is therefore an advantage.

Prof. F. Haber[6] gives the following figures: 1 kw.-year (of 365 days of 24 hours each) can theoretically yield the following maximum amounts of nitric acid by combustion of atmospheric nitrogen:

1850 kg HNO_3 at 4200° C.
819 kg HNO_3 at 3200° C.

A decrease of the temperature by 1,000° C. results, therefore, in this case in a reduction of the efficiency by more than 50 per cent.

Reverse Reaction.—The formation of nitric oxide at the high temperature is followed by the dissociation of the oxide into nitrogen and oxygen during the period of cooling.

This is a necessary consequence of the laws of chemical dynamics applied to reversible reactions. If, for instance, the fundamental reaction

$$\frac{1}{2} N_2 + \frac{1}{2} O_2 \rightleftarrows NO$$

is carried out at a temperature of 3,200°, whereby the limit of formation of nitric oxide in volume per cents is 5 per cent (Nernst's figure), and if the gas mixture is then cooled very slowly to 2,200° and equilibrium is again established at this temperature, the volume percents of the NO will be reduced to 1 per cent. During the slow reduction of the temperature by 1,000°, one loses, therefore, 80 per cent of the NO which was produced before at 3,200°. This is the phenomenon of the reverse reaction.

We saw before that with raising temperature equilibrium establishes itself with greater speed at higher than at lower temperatures. In the same way, when the temperature is diminished, the reverse reaction proceeds more rapidly at higher temperatures than at lower temperatures. This is also a consequence of the general phenomenon of reversibility.

It follows that the most dangerous temperatures—those in which the reverse reaction during cooling will undo what has been performed before—are high temperatures, near the temperature of the primary reaction. From this results the necessity of cooling the gas mixture as suddenly as possible from the temperature of the arc, in order to bring them in the shortest possible time to such temperatures at which the velocity of the reverse reaction is practically zero. In the case of combustion of nitrogen this condition is realized more easily, for the reason that below 600° NO combnies gradually with the excess of oxygen to form nitrous vapors which escape the reverse reaction.

In practice the tendency has been to obtain this result first by rapidly sweeping the gases out of the region of action of the arc, while more recently use has been made of electrical or mechanical devices by which the arcs were successively lighted and interrupted several thousand times per second, or the arc has been forced to play in different regions of the space. In all these cases the gas mixture which has been brought instantaneously by the arc to a very high temperature is cooled also instantaneously in the mass of surrounding cold air, and the effects of the reverse reaction are, if not suppressed, at least considerably diminished

To sum up, the fundamental considerations which govern the combustion of atmospheric nitrogen are the following: First, it is necessary to work at a high temperature in order to increase the efficiency and the velocity of the reaction; second, to cool the gas instantaneously in order to avoid the reverse reaction.

It is at once evident that it is rather difficult to realize simultaneously these two conditions in practice. According to whether the experimenters have satisfied more or less one or the other condition, the results have apparently been contradictory. This explains the very peculiar former results to which we referred above.

The gases obtained in this way contain some 1 or 2 per cent by volume of nitric oxide when they pass out of the arc chamber, and must then be treated in order to transform the nitric oxide gas into nitric acid or into nitrates and nitrites. These operations are more of chemical than of electrochemical interest, and it will be sufficient to indciate their principle, although the practical operation involves certain difficulties. It is, indeed, necessary to handle a considerable dead weight of inert or indifferent gases. By cooling the nitric oxide gas, NO is transformed into N_2O_3 and N_2O_4 as soon as the temperature falls below 500° or 600°. Proper reactions with water, or with alkaline solutions (caustic soda, lime water, etc.), give either dilute nitric acid or nitrates or a mixture of nitrates and nitrites.

We may finally call attention to a characteristic feature which is common to all the methods which have been studied.

Whatever may be the method which is adopted, whether one operates with direct current or with alternating current, or with rapid electric oscillations in air in motion, or whether the electric discharges are displaced in space, the arc, in order to be lighted, requires a higher voltage than that which is sufficient to maintain the arc in the state of stability after it has once been established. It is, therefore, necessary to install between the source of electrical energy and the arc chamber either a considerable resistance with direct current or one or several induction coils with alternating currents or oscillatory discharges.

From a practical point of view, this means that in the arc only a fraction of the nominal capacity of the dynamo can be utilized. With alternating currents, for example, there will always be a considerable phase difference, measured by a power factor which has often a rather unfavorable value. Estimates of installations must take this point into account, since it amounts to an increase in the cost of dynamos, and therefore in the cost of the electrical energy.

The extended and expensive researches of recent years have, therefore, very greatly simplified the understanding of this problem which formerly was so mysterious. It may be interesting to briefly state what has been accomplished in practice. The best results which have been published indicate a production of 800 to 900 kilograms of HNO_3 per kw-year, measured at the arc. In order to take into account the accessory losses of electrical energy and other difficulties, it is wise to admit that in industrial practice the above figure will be reduced to about one-half and to assume, therefore, that one-half a ton of nitric acid can be produced per kw-year on an industrial scale by electrochemical processes fixing the atmospheric nitrogen.

If we take the cost of the electric kw-year as $12, which price can be realized in large installations, the energy required for producing 100 kg. of nitric acid would be about $2.40.

Now the market price of 100 kg. of nitric acid is about $9 in the state of concentrated nitric acid, and $7 in the state of virtual acid (in the nitrate at $5.20 per 100 kg.). The margin left between cost of production and selling price seems, therefore, sufficient to try, with pretty good chances for success, the electrochemical manufacture of nitric acid on a large scale, especially if the first cost of installation is not too high, which is evidently the fundamental point on which great stress should be laid.

An experiment of this kind is now actually undertaken in Norway in a plant using 2,000 or 3,000 hp.[7]

[6] F. Haber, Thermodynamik technischer Gasreaktionen (München, 1905), page 250.

[7] See our January issue, page 23.

IV. Use of Liquid Air.

We may finally say a few words concerning the rôle which liquid air may play in the development of the nascent industry of the fixation of atmospheric nitrogen.

The calcium cyanamide process needs as starting material pure nitrogen free from oxygen. Up to the present time this has been produced by passing atmospheric air over easily oxidizable substances (ferrous and cuprous salts, etc.) which absorb the oxygen of the air. Recently, in the plant in course of construction in Italy,[7] distillation of liquid air is being used for the same purpose. As is well known[8] pure nitrogen may thereby be obtained under more or less economical conditions. But in the present case the problem arises what to do with the oxygen which remains, especially if considerably large quantities are handled.

Now we have the extremely interesting fact that the combustion of atmospheric nitrogen takes place with a very appreciably better efficiency if it is carried out with an excess of oxygen. It follows that if the two industries—manufacture of calcium cyanamide and production of nitric acid by arc discharges through air—are installed side by side, they will be able to utilize both products, the nitrogen and the oxygen obtained from liquid air.

Instead of competing with each other, the two processes should, therefore, really be allies. In working together they will be in a position to bind the atmospheric nitrogen under much more economical conditions than if each process would be carried out alone. It would be an interesting development if in the way just sketched liquid air should play in future an important part in uniting the two most promising electrochemical methods for the fixation of atmospheric nitrogen on a commercial scale.

University of Geneva, Switzerland

Containers for Alkaline and Lead Accumulators.

By M. U. Schoop and C. Liagre.

(*Concluded from page 104.*)

Containers of Sheet Iron or Steel.

The large-scale industrial manufacture of sheet-steel vessels can be carried out by different methods, namely:

1. Folding.
2. Electric welding or welding with the oxyhydrogen flame.
3. Spinning.
4. Pressing with extremely high pressures (Huber method).
5. Electrodeposition.

Folding.

According to our experience it is very difficult to maintain single or double-folded steel-sheet walls absolutely dense. This is especially the case with the bottom, while vertical seams may be easily made dense by using strong pressures. Some advantage may be derived from using proper intermediate layers, for instance, parchment, thin rubber strips, etc.

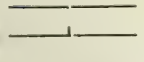


FIG. 2.—WELDING METHODS.

For laboratory purposes, it is also possible to apply a metal which is soluble with difficulty in caustic solution, from the outside onto the folded seam by means of the oxyhydrogen flame. Apparently the best metal for this purpose is copper in form of wire of 1.5 mm. thickness. With some experience it is thus possible to produce fairly good and reliable containers, which are preferably plated afterwards with nickel on the inside and outside. It is still better to use at once nickel-plated steel sheets, although they are rather expensive.

Electric Welding and Welding with the Oxyhydrogen Flame.

The practical details of these methods shall not be discussed at length at this place. Welding with the oxyhydrogen flame has been described in detail in the books of V. Engelhardt, on the electrolysis of water, and of M. U. Schoop, on the industrial electrolysis of water (Enke-Stuttgart). The principle of the method is that by means of a suitable mixing-burner, hydrogen and oxygen are mixed in the proportions of about four to one and burned, which yields a temperature at which iron and copper melt like tin, and welding seams are obtained which are very similar to the so-called seams of lead sheets.

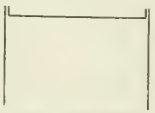


FIG. 3.—WELDING THE BOTTOM.

An excess of hydrogen is absolutely necessary for working most metals; the flame must have reducing properties because the atmospheric oxygen participates in the oxidation of the hydrogen emitted by the burner.

It is important that both gases have an opportunity of mixing thoroughly before burning, since, otherwise, the temperature of the flame is not homogeneous. It is easy to avoid explosions if the speed of the exit of the gas mixture is made greater than the speed with which the burning of the oxygen-hydrogen gas mixture proceeds.

While even the welding of lead sheets by this method requires some experience, the difficulties are even greater with the welding of iron and steel sheets. Two different methods of making the seams are shown in Fig. 2.

With very thin steel sheets distortion may easily result, which is indeed a weak point of the process in general. It is possible to prevent this to some extent by welding the seams, after having preheated the vessel to a rather high uniform temperature.

The vertical seam may be placed at will, either at an edge of the vessel or in the center of a side wall, and turns out best (*i. e.*, with very small deformation of the vessel) if some points at distances from each other are first welded, which the workingmen call basting. While the bottom offers considerable difficulties in the folding process, as mentioned above, it is a comparatively easy matter with the welding process (Fig. 3).

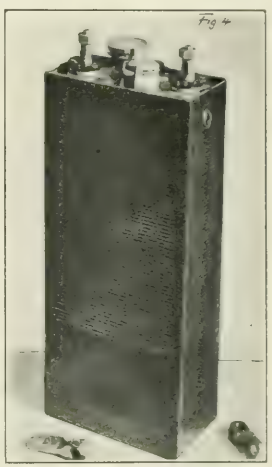


FIG. 4.—EDISON STORAGE CELL.

Every 1 meter of seam between sheets of open-hearth steel requires

Thickness of Sheet	Time.	O Consumption.	H Consumption.
½ mm	4½ minutes	7½ liter, about 1½ liter per minute.	30 liter, about 6 liter per minute.
1 mm	6 minutes	2½ liter, about 2 liter per minute.	50 liter, about 8 liter per minute.
3 mm	14 minutes	50 liter, about 4 liter per minute.	225 liter, about 16 liter per minute.

In this table it is supposed that the work is done by a skillful, trained workingman.

It is self-evident that with welded vessels there is no danger of places being not dense, since a test of the vessel will show at once any flaws.

Edison used for some time a cadmium-tin solder for his iron-nickel accumulators, but it is stated that the results obtained with it were poor. This cadmium solder has now been definitely replaced by electric welding. In Edison cells which we saw some months ago the bottom and top were welded in this

[8] See, for instance, our January issue, page 30.

way, and the boxes were provided outside with a black and well-adhering coating of paint (Fig. 4).

To be successful in welding with the electric current it is important to use the proper voltage. It is also important that the iron rod which forms one pole and from which the current passes over to the place to be heated is made negative. Another condition for perfect results with electric welding is great purity of the material.

FIG. 5.—SEAMLESS STEEL-SHEET VESSEL.

Spinning.

The method which at present seems most promising, and probably most suitable for manufacture of sheet steel vessels on a large scale, is the method of spinning, as it is used, for instance, for cartridges, scarpnells, etc. By this method the whole vessel including bottom, is made in one seamless piece. But the manufacture of spun vessels, 300 mm. high, 140 mm. long, and 90 mm. broad, with a thickness of walls not more than 0.5 or 0.7 mm., meets with extraordinary technical difficulties, so that it must be considered at present as an art, and will probably remain an art for some time to come. It is also evident that with this process each new type involves considerable expense, so that it can be profitable only if large quantities of one and the same type are to be made. While with the welding process clean, sharp corners are easily obtainable, equally sharp corners cannot be expected from spun vessels, since in this case with sharp corners no material could withstand the enormous stress.

Figs. 5, 6 and 7 show photographs of spun sheet steel vessels made of one piece. These vessels are first spun in cylindrical form, the bottom forming a cone or cap, and the change to a rectangular cross-section produces, therefore, at the bottom an excess of material which may be seen in form of four laps in the illustrations. Fig. 5 shows a vessel with plane walls, Figs. 6 and 7 vessels with walls which are corrugated, in order to increase the mechanical resistivity.

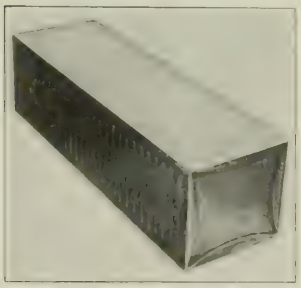


FIG. 6.—SEAMLESS SHEET-STEEL VESSEL.

A disadvantage of these vessels is that the plating of the interior with nickel offers considerable difficulties, and that in the laps in the bottom the pickling acid may stick so that it will be extremely difficult to remove it entirely.

Hydraulic Pressing.

Huber conceived the idea to produce changes of form of bodies in the interior of the press-cylinder itself. The steel sheet to be pressed is placed together with the die into the cylinder itself, which is filled with water, and by the very strong water pressure, which is uniform on all sides, the steel sheet is pressed into the die. The power element which produces the pressure is, therefore, the water, and it acts uniformly from all sides.

Since, for most metals, like iron and soft steel, the elasticity and fluidity limit is below or not much above 40 kg. per square millimeter, it is possible, by applying a water pressure of 50 kg. per square millimeter, or 5000 atmospheres, in the interior of the press-cylinder, to bring most metals while cold to the fluid state and press them thereby into dies.

The water is the active pressing material, and its pressure does not act in a single direction but acts upon the material to be treated from all sides in uniform strength.

With a hollow body with water inside and outside the pressure is the same inside and outside, and there will be a change of form only at those places where the steel sheet is closely fitted on the die, the steel being forced into the depressions of the die. In the interior of the press-cylinder it is therefore possible to produce manifold changes of form without any pounding or punching, if the following two conditions are fulfilled: First, that the water pressure is high enough to bring the steel sheet to the fluid form and force it into the depressions of the die; and, second, that the die has greater resistivity than the steel sheet, the form of which is to be changed, and that the steel sheet is closely fitted to the die.

The operation begins with filling the press-cylinder with water. The steel sheet is then fitted to the die, and the contact rims are made dense, and the whole is then placed into the water. A piston is now forced into the press-cylinder, whereby a pressure of 4,000 to 8,000 atmospheres is produced in the inside of the cylinder. The change of form begins with bending the steel sheet against the die, and completed after the fluidity limit of the metal has been transgressed.

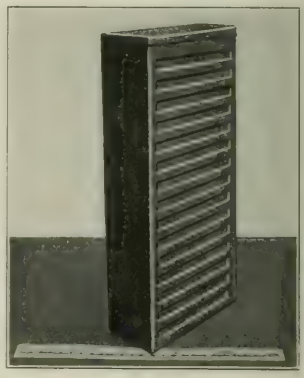


FIG. 7.—SEAMLESS SHEET-STEEL VESSEL.

It is, of course, important that the rims where the steel sheet makes contact with the die are made dense and absolutely water proof, so as to prevent any water of high pressure from getting between steel sheet and the die. This would, of course, be fatal for the success of the method employed.

A rectangular vessel with bottom is made as follows: A circular steel sheet of 8 to 10 mm. thickness is first preliminarily spun so as to get a form somewhat like a hat, as shown by the dark, black line in cross-section in Fig. 8. This is then placed on the box which forms the die, and the lines of contact are made dense with a proper material, as indicated at the top of Fig. 8. The whole is then placed into the press cylinder, and by means of water pressure from all sides the steel sheet (the dark, black line in Fig. 8) is pressed into the rectangular box form by means of the water pressure, which is indicated by arrows in the illustration.

To make steel boxes, 250 or more millimeters deep, the pressing operation is to be carried out in several steps, one after the other, with annealing between two successive steps. Even with the process carried out in several steps, only one die is necessary, if in the first steps of the operation a suitable filling material is used which is afterwards removed.

At the exposition at Düsseldorf, in 1902, a Huber press was shown in the pavilion of Fried. Krupp. The press-cylinder had an inside diameter of 600 mm. and a primary pressure of 530 atmospheres was produced, while in the upper reservoir of the apparatus this pressure was increased fourteen times, and thus brought up to about 7,000 atmospheres. The press was con-

nected with an electrically-driven pump giving 40 liters per minute.

Steel boxes may also be made by the Huber process by pressing in this way first the four side walls and by adding the bottom afterwards.

Electrolytic Deposition.

It is now possible to produce rather thick layers of steel by electrolytic deposition, and it is not impossible that later on this method may be made use of in the manufacture of steel boxes. Iron deposited by electrolysis is extremely hard, probably on account of occluded hydrogen, which produces on one hand great brittleness and on the other hand great hardness. Another extremely valuable property of electrolytic iron deposits is that they do not rust, and that they have a much greater resistivity against chemical influences than ordinary iron.

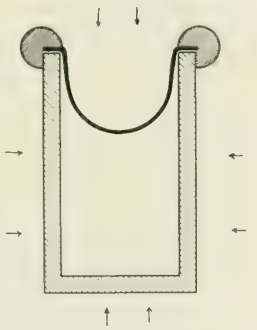


FIG. 8.—MAKING A STEEL BOX HYDRAULIC PRESSURE.

Celluloid, which is so much used with lead accumulators and which combines with transparency an almost ideal capability of being machined, cannot be used, unfortunately, for alkaline accumulators, since it is quickly attacked in solutions of 0.2 specific gravity. Only with 5 to 8 per cent solutions the attack becomes negligibly small.

The foregoing notes show that the manufacture of perfect iron or steel boxes still offers considerable difficulties, which are much greater than the production of hard-rubber boxes or sheet-lead boxes.

Finally, the proposition may be mentioned to enamel steel boxes. In this case it is no longer necessary to have the steel containers absolutely dense before they are enameled, since the enamel guarantees that the box becomes liquid proof, as is the case with many kitchen utensils, like coffee pots, etc. Riveting would, therefore, be sufficient. The enamel would also offer an excellent heat insulation, which is obtained in the Edison accumulator by a well-adhering black varnish coat; at the same time the vessel would be well insulated in electrical respect.

Jubilee of the Coal-Tar Industry.

The year 1906 marks the fiftieth anniversary of the epoch-making discovery by William Henry Perkin of the dyestuff "Mauve," by which the foundation was laid of the coal-tar industry and great stimulus given to the study of organic chemistry and chemistry in general.

At a public meeting at the Mansion House, in London, presided over by the Lord Mayor, it was decided to celebrate this anniversary in a manner befitting the importance of this great discovery. The Germans, headed by Dr. Caro, have already joined in the movement, and it is hoped that the celebration will become an international one.

In order to secure American cooperation a meeting is to be held at the Chemists' Club on April 7, 1906, at 8.15 P. M., with the object of considering what steps should be taken to give effect to the suggested jubilee celebration. At this meeting a general committee and an executive committee will be elected, and the form of memorializing the event in America, and of doing honor to W. H. Perkin, who is, fortunately, still in full activity, will be discussed. The circular of invitation is signed by Dr. C. F. Chandler and Dr. H. Schweitzer.

The Eldred Flame.

By K. P. McElroy.

What has become known to fame as the "Eldred flame" is really not a flame at all in the ordinary acceptance of that term; and the word flame is only adopted for lack of a better. A flame, as usually understood, is, so to speak, a cored structure, a system of layers of different composition. There is, as in the ordinary gas jet, firstly, a core of unburnt gas; secondly, a jacketing layer on which and in which active combustion is occurring, and, finally, an irregularly stratified layer of uprushing heated air and products of combustion. The same structure is found in all common flames, whether from a candle or a reverberatory hearth. They are all cored and layered. The "Eldred flame," on the other hand, is structureless when completely developed. In lieu of combustion taking place only on or in a layer, it is occurring throughout the whole mass at the same time. It is, to borrow another term, an exploding mass with the explosion simultaneously taking place at all points; but the explosion instead of taking place in a fraction of a second, as with such explosive gas mixtures as are used in gas engines, is retarded to require as much time as may be desired, even minutes. In practice, when used in a furnace, the explosion is so timed as to require the full period of the passage of the gases through the hearth chamber, being completed just before the stack is reached.

In other words, heat is evolved in every part of the hearth chamber in lieu of being generated at some one point, generally outside, and conveyed into and through the chamber, as in the ordinary practice.

The artifices adopted to accomplish this desirable end are of quite astounding simplicity practically, though the theory of the operation is by no means so free of complexity. An ordinary grate chamber containing a shallow bed of ignited fuel is closed against access of pure air, and through the fuel is passed an accelerated draft current of air containing a regulated proportion of products of combustion, the rate of its passage being greater than the ability of the fuel to act upon it. The issuing body of gas consists of nitrogen, unchanged oxygen, unchanged carbon dioxide and carbon monoxide from the fuel, with, perhaps, a little steam or hydrogen from some coals. It is this mixture that forms the "Eldred flame." The fuel on the grate, because of the composition of the draft current, burns with relatively cool combustion, producing carbon monoxide, and though the consumption of coal per hour may, if desired, be even larger than in ordinary firing, yet the heat never becomes great. Roughly speaking, of every 100 units of heat yielded by the total combustion of carbon, thirty are due to the formation of monoxide and seventy to the conversion of monoxide into dioxide. Under the circumstances stated, dioxide is not formed and, on the contrary, much of the dioxide in the draft current is reduced with absorption of heat. The gross result is that the fire burns cool.

If a mixture of combustible gas and air be made in equivalent proportions, it is, of course, instantly exploded by being brought into contact with an object sufficiently hot to induce ignition. As the proportion of air is increased and that of combustion diminished, explosion becomes more and more difficult, until finally the mixture will not explode at all, though if a body of it be sufficiently heated from an outside source it will burn throughout its mass, though obviously, since air and combustible in the imagined case are admixed, molecule against molecule, no cored ordinary flame is possible. Such a very dilute mixture, which would not burn or explode at all on contact with a flame, would nevertheless burn if brought into the heated region of a reverberatory. It would then be in a way an "Eldred flame," but worthless for practical purposes, since the great dilution would mean useless consumption of heat in warming the excess of air, and there would hardly be enough margin of heat to keep the reverberatory region at the required temperature.

In the case just stated, while the combustion or explosion—the words may be used synonymously for present purposes—is retarded, it is done almost purely physically by the dilution. In the Eldred flame in its commercial embodiments, however, positive retarding means are employed, enabling the use of much larger relative proportions of combustible, and, therefore, the production of useful heating phenomena. Retardation is more a chemical than a physical function.

For the present purpose, the Eldred flame may be regarded as consisting of a mixture of oxygen, carbon monoxide and carbon dioxide, the presence of the diluting nitrogen being for the moment disregarded, with the monoxide and oxygen existing in about the theoretical combining proportions. Theoretically, every mixture of CO, O and CO_2 at a temperature above that necessary to institute chemical reaction must result in a product, after the reaction—if any—containing all three in certain relative proportions, with a certain "partial pressure" of each. This partial pressure of each will depend upon the temperature and certain other considerations. But, as a matter of theory, all three must be present in some relative proportions in every gas mixture containing carbon and oxygen and above a red heat. That is, for instance, if carbon dioxide itself be heated a certain amount of it will tend to split up, or dissociate, to form carbon monoxide and oxygen; this amount varying with the temperature, but, in the stated case, constant at each temperature.

The relations of the three bodies, however, become quite complicated when carbon monoxide and oxygen are not present in equivalent amounts, as they are in the example just given. Disregarding the influence of the partial pressure of the carbon dioxide in such a mixture, the amount of either the monoxide or the free oxygen is inversely proportional to the excess of the other. That is, the greater the partial pressure of the oxygen, the less is that of the carbon monoxide capable of co-existing and inversely. With a sufficient excess of oxygen, the amount of carbon monoxide, as in ordinary "complete combustion," may become vanishingly small; with a sufficient amount of carbon monoxide in excess the amount of free oxygen may approach the vanishing point. It is obvious that at the point where the two gases balance each other chemically, more of each can exist together than at any point where either is in excess. Further, and this is an important point, the velocity of the union of the two gases slows down more than proportionately as the reaction approaches the point where there remain only the partial pressures of the two gases which are capable of co-existing.

Taking the carbon dioxide now into consideration, its partial pressure in the gas mixture influences enormously the reaction between the other two bodies; the greater its amount the more of the two can co-exist without combination, and, also, the more sluggish is their reaction or combination. It has a specific retarding influence on the reaction. Put in other words, the presence of carbon dioxide tends to retard the formation of more carbon dioxide, and the greater its partial pressure in the gas mixture, the more this retardation is evident. It does not prevent the combination of carbon monoxide and oxygen beyond a certain partial pressure of each (this partial pressure of each depending on the ratio of the partial pressures of the three components and on the temperature), but it does positively slow up, or retard, the velocity with which the oxidation of the monoxide takes place.

Turning from theory to practice, in the Eldred flame there is present such a three component mixture of carbon monoxide, carbon dioxide and free oxygen, the whole being further diluted by the nitrogen of the air and of the products of combustion. The relative rates or partial pressures of the three are dependent on the composition of the draft current, the depth of the fuel bed and the speed of the draft; obviously all matters susceptible of easy regulation. And with this regulation will vary concomitantly the composition of the gas mass produced. With a deep bed of fuel and a slow draft, not exceeding the rapidity of reaction of the carbon, the apparatus becomes a gas producer, yielding a gas containing much monoxide and only as much of the inevitable free oxygen and carbon dioxide as the partial pressures of those two gases amount to in a mixture of that composition and temperature. Such a gas is, so to speak, in a resting state; it has no inherent capacity for reaction. If, on the other hand, the bed of fuel be made shallow and the draft current rapid, products appropriate for the Eldred flame result; products which are in a dynamic state in lieu of the static state of the producer gas.

This mixture of gases as usually produced in practice has enough oxygen distributed throughout it to combine with the carbon monoxide similarly contained, but combination is slackened both by the dilution and by the specific retarding action of the carbon dioxide. Further, the fact that there is no marked excess either of monoxide or of free oxygen likewise slackens the reaction. The result is a mixture which, while capable of burning or exploding throughout its mass, like the mixture in a gas engine cylinder, nevertheless, does so very slowly; and which, in most embodiments of the process, is of such a nature that it requires to be run into a hot-walled chamber to burn with any speed. Therein, however, under the influence of the radiant heat it burns, giving out its heat uniformly at all points within its mass. In other words, in lieu of a real flame there is a sort of ignited atmosphere filling the chamber. In its perfection the "Eldred flame" is not visible to the eye, there being no solid matter to radiate light, and the walls of a furnace fed with it glow with heat without apparent cause.

In such a furnace the development of heat is practically uniform at all points. Herein lies the great advantage of the flame for practical purposes. In the usual grate firing with production of ordinary flame, this flame is but short and lean; often, as with anthracite, extending but a few inches from the fuel bed. The evolution of heat by the fuel is intense, but it is evolved near the grate, and for heating objects further away the sensible heat of the products of combustion must be relied on—the heat of the flame gases. But sensible heat is given up to the first cooler object encountered, and the result is that a hearth chamber so heated is uneven in temperature, the parts near the entrance being overheated and those more remote not hot enough. The heating is uneven. In the case of a kiln of bricks, the bricks near the grate are over-burnt, those near the stack are under-burnt. This spottiness of heating may be, and often is, remedied in great part by running the flame reducing; using soft coal and supplying part of the air needed in the hearth chamber itself. But in the case of a reverberatory this means a long, thin flame hugging the arch and separated from the materials on the hearth by a stratum of colder air. Further, the size of the furnace which can be so heated is limited by the length of the flame which can be produced. And in any event the amount of heat is only that yielded by a film or series of films of burning gases.

With the Eldred flame, on the other hand, the possible size limits of the furnace are much extended; the heat is evolved where it is needed instead of somewhere else, and in lieu of a thin flame as remote from the material it is to heat as the limits of the hearth chamber will permit, there is an ignited atmosphere in actual contact, or as near as may be desired, with the material. Further, and this is a most important point, the character of this atmosphere, whether reducing or oxidizing, is under instant control of the furnace man. It merely requires the touch of a valve to adjust conditions as may be desired, giving a coal-burning furnace all the convenience of a gas furnace and more. And, since with the cool combustion on the grate there is no slagging or clinkering of ash, nor destruction of grate-bars and fire-box linings, it is possible to use a forced draft and burn much more fuel per square foot of grate area than is usual, thereby enabling the use of colossal furnaces and resultant economy of labor. The lack of struc-

ture of the "Eldred flame" allows it to fill any hearth chamber whatever its shape or size.

But in the embodient which has been described, it does not give any intense or localized heat; its leading characteristic is absolute uniformity of heat throughout great spaces. In certain uses it is desirable to get such a localized heat at points remote from the grate, and this has led to an interesting evolution founded on the same basic principles.

As already stated, the absence of an excess of free oxygen retards combustion inordinately. This is a familiar matter of observation, for furnace men, anxious for quick and intense combustion, always consider it necessary to admit a large excess of air over and above that chemically required; 20 per cent is a common estimate. Now, in the Eldred flame, as described, there is no such excess of oxygen, commonly there is just about enough to combine with the carbon monoxide. Therefore, at any spot in the furnace chamber where it is desired there should be an intense development of heat, a penetrating jet of air is sent through the Eldred flame. It does not carry 20 per cent of air over and above that in the flame or anything like it, thereby wasting heat, but it does supply at the spot touched a local excess of air, and in its presence combustion becomes vivid and heating intense. This air is not supplied to burn the combustible, there is enough there already for this purpose; it is added to quicken the burning by furnishing an excess partial pressure of oxygen. It is, so to say, supplied for its physical rather than its chemical effect. With a needling jet of this type an intense heat can be supplied at any spot whatever; the furnace-man not being limited by the characteristics of his fuel or his furnace; neither is he supplying an inordinate volume of cold air to the whole furnace. It is obvious what a perfect control this, in connection with the properties of the Eldred flame proper, gives over furnacing operations.

Like most inventions this grew out of a need. Mr. Eldred found the lime burners perplexed by the fact that coal burnt no more lime than wood, although it gave many more heat units, and his flame was the result. The explanation of the lack of utility of coal, after he discovered it, was simple enough —the coal flame was too short and lean, and its intense heat was of no particular avail, being merely expended in overheating and burning the lime next the grate. Calcining lime is a peculiar reaction in that it requires no great intensity of heat, but it does require a great deal of it, and, further, a large volume of hot gases to sweep off the carbon dioxide as fast as formed. In a way, it might be said that the intensity of the heat required and the volume of ambient gases are reciprocal factors. The Eldred flame with its large volume of ignited gases and its "in situ" evolution of heat, gave the necessary conditions for a cheap and profitable lime-burning method.

After gaining a foothold among the lime burners, most of whom now use it, the idea steadily grew, as is the habit of ideas, and modification after modification followed, until it is now in use in a variety of arts; but in all its avatars it has preserved the simple central idea of a slow-exploding gas mixture, burning and evolving heat throughout its mass.

Of course, the idea of returning smoke and gas to a fire to complete their combustion is a very old one; it dates back probably to the time when the first board of health and the first factory chimney were antagonistically arrayed. And the records of the patent office are full of propositions, mostly rather foolish, for making a fire burn its own smoke. None of them amounts to much, for the simple reason that without a furnace chamber capable of withstanding indefinite pressures, or as big as the atmosphere, it is impossible to keep on indefinitely returning furnace gases without an outlet—which is something that many of the inventors evidently forgot—and if any reasonable proportion, say 80 to 95 per cent, of the total products of combustion be allowed to escape there is no perceptible diminution of smoke.

But in the Eldred invention, while smoke is incidentally obviated by regulating and perfecting combustion, there is no thought of returning and burning it. The simple and beautiful underlying idea is that of mixing air with certain regulated amounts of carbon dioxide for the purpose of obtaining certain relative proportions or partial pressures of oxygen and carbon dioxide, and ultimately obtaining therewith what may be called an "auto-combustible" mixture having certain mutual relations of the partial pressures of carbon dioxide, carbon monoxide and free oxygen. Products of combustion are merely chosen as a source of carbon dioxide because cheap and convenient

Factory Scale Experiments With Fused Electrolytes.—I.

By Edgar A. Ashcroft.

Introductory.

In this series of articles the writer intends dealing with a few special lines of practical electrolytic work with which experimental trials have been conducted on a scale comparable to factory conditions, as distinguished from laboratory experiments. The subject will be divided into several short articles dealing with: the electrolysis of fused zinc and lead chlorides, the direct electrolysis of sulphides, the magnetic stirring of fusion baths, electrolysis of sodium chloride and the production of alkali metals. Other subjects will be incidentally touched upon.

Outside the practice of a few special arts, developed for specific purposes, and associated in the public mind exclusively with individual metal recoveries or allied processes (such as the aluminium industry or the sodium and fused caustic processes, the secrets of which though, for the most part, closely guarded as to small details of practice, are, nevertheless, well known in all essential principles), but little is generally known of, and little interest seems to center in, the potential and actual openings for this line of applied electrochemistry. I place "potential" first, because the true explanation of the lack of interest and general information on the subject lies precisely in the fact that up to date the openings *are* more potential than actual (or commercial).

This is largely due to the chance circumstance that, where a particular salt (such as zinc or lead chloride) is peculiarly suitable for direct fused electrolysis, this salt does not happen to be one either widely diffused in nature or easy of manufacture from natural ores at remunerative prices and in a sufficiently pure form. It is especially characteristic of electrolytic processes (as distinguished from electric furnace methods) that very pure raw materials are essential to even moderately economical work. For the labor, breakage and general disorganization involved in any frequent dismounting and cleaning out of such apparatus as we are here considering is so great as to be prohibitory if it should become necessary as an item of regular work.

It is, therefore, usually preferable to purify the raw material rather than to produce an inferior product and subject this to a refining process (a method characteristic of most non-electric metallurgical operations). A typical case in point is the aluminium process, which, however, is sufficiently advantageously placed in other respects as to competitive methods, etc., to stand this handicap (as well as the still more severe handicap represented by the actual consumption of so expensive a form of fuel as carbon anodes are, even at their cheapest).

In spite of this actual limitation of the immediate commercial importance of the subject of fused electrolysis, a few clear ideas as to its practical bearings will be of some value to the ever-increasing army of those engaged in pioneering the ground on which may rest the commercial interests of another decade or two. Moreover, there are a few notable exceptions to the above rule which are worthy the immediate attention

of practical men, and round some of these it is practically certain that greater interest will center in the near future.

A good case serving to illustrate the less favorably placed competitors for recognition in this field, is the elcetrolysis of fused zinc and lead chlorides. When the writer first tackled this problem a good many years ago he was under the impression (an impression which still rests in many minds) that he had a very difficult problem before him. Apart from the erroneous theories mooted from time to time on the basis of wrongly conducted or badly conceived laboratory experiments (some professors a few years back were particularly guilty in this respect), there was an almost universal conviction that the practical details (refractory materials, etc.) were almost hopelessly difficult, and the necessary conditions of work consequently extremely onerous. This may be partially true where heat has to be conveyed to the electrolyte through walls of stoneware or similar material from outside sources, or where it is impossible, owing to the strong affinity of the fused metal for iron or steel (as in the case of fused zinc), to use metal containers for the exteriorly heated bath.

The actual fact proved far otherwise, however, if the bath is heated from within by means of the current; for the electrolysis of pure anhydrous fused zinc chloride with a fluid cathode is one of the easiest, cheapest and smoothest working processes which it is possible to conceive, and the same may be said, of the electrolysis of pure fused lead chloride, and of all fused electrolysis work in general where internal heating is applied, and where specific difficulties, due either to impurities or incompatible physical conditions (such as impossible fusing or vaporizing points, etc., and baleful interference of such impurities as iron salts, which cause great losses of efficiency by reason of the alternate ferrous and ferric reactions at cathode and anode respectively) do not arise.

The reason then that these operations have not yet been extensively applied lies in the fact that the salts suitable for

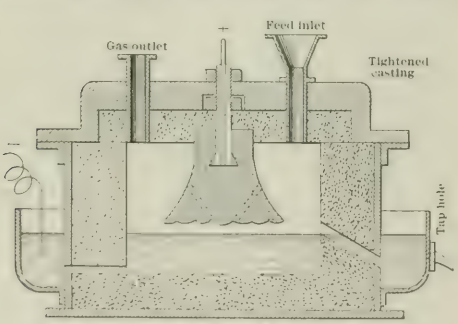


FIG. 1. VAT FOR FUSED ELECTROLYTE.

electrolysis have not yet been produced in the right commercial relation to the value of the products; not in any inherent difficulty or great cost in the electrolytic process itself. Perfectly pure anhydrous zinc chloride, for instance, is very difficult to obtain at all, and even the slightly hydrated product of commerce, and also pure lead chloride, have nearly equal market values to the metal contained; and the chlorine recovered in the treatment will seldom pay for the cost of operation.

If, however, conditions should arise where these limitations do not apply, or if it should be desired to electrolyze these salts for special reasons in spite of them, the procedure is then as follows, and I give it most willingly in detail, as it forms the basis of much of the work on derived or allied processes, which I shall describe later in this series of articles:

Fig. 1 shows the internally heated zinc vat in its latest improved form. The same apparatus in all respects is suitable for the lead process. The distinctive features of this improved vat are: the solid anode, which is cheaper and more efficient than the older forms; the means of tapping, using a continuous flow through a slanting hole into a well, which acts much like a weir, and thus maintains the distance between anode and cathode constant, and the exterior connection to the molten cathode, which enables a purer product to be obtained and is also a cheaper and more efficient construction than the steel blocks formerly employed to make connection with the molten zinc.

The apparatus needs little description. The outer casing may be of thin boiler iron, the brick lining of "Stourbridge" or similar material, preferably made into suitable "specials" or slabs and blocks, and for the cement a mixture of zinc oxide with silicate of soda solution is employed. This cement sets quickly and makes an excellent job, impervious alike to moisture and to the fused electrolyte and clean and strong. Electrical connection is made with the cathode by means of the outer pocket, where the metal is allowed to set; the sloping passage to the second pocket constitutes a kind of weir over which the metal flows into a well before it is periodically withdrawn from the tap hole. The thickness of the lining of brick work is only of importance in its influence on the loss of heat by radiation and convection. I have used a vat quite successfully having only a 4½-inch slab lining, and though a thin lining means somewhat high volts (to compensate for the loss of heat); some advantage is also obtained by the increase of current density and consequent reduction in the size of the apparatus for a given output.

The electrical connections to the carbon block are of some importance, as these are apt to offer undue resistance and thus both absorb voltage, and give out destructive heat. The best connection for a single carbon block is secured by having a 2-inch hole some 6 inches deep in the carbon, into which is placed a hollow copper rod, 1½ inches in diameter and pierced with a ½-inch hole. This connection is then made good by means of molten lead and strengthened by any convenient mechanical-holding device.

In practice on a large scale with this process it would probably (except in special localities where power costs are high) be most economical to use a fairly high current density (1,000 amps. for 1 square foot of anode surface and 2 square feet of cathode, the anode being measured only on the under side); to employ a lining of about 9-inch thickness, and to maintain the temperature of the workroom at the maximum practical to work in (say 100° F.) by means of steam or other heaters.

The exterior casing of the apparatus never reaches a very high temperature under the circumstances of this work, and the room temperature is, therefore, relatively very important as a factor in the conservation of heat. Very little attention is required at the vats, and the occasional tapping of the zinc in such a temperature as 100° need not be a serious hardship on the workmen. A better plan still might be to enclose the vats in a kind of flue, leaving only the tops and one side (at the tapping holes) accessible. The prevention of radiation by this means is much more efficient than any amount of increase in the thickness of the lining. Any thickening of the lining inside the containing tanks is also apt to increase the cost of the apparatus considerably. But the commercial relations of these things will be again influenced by the relative cost of heat for the room and the cost of electricity.

The vat constructed as above described is worked closed and under a very slight suction (not more than ½ inch of water gauge), and the chlorine gas may by this means be drawn off with only a slight admixture of air (it is extremely difficult, however, to get richer gas than 75 volumes per cent, owing to inevitable small in takes of air, etc., through inspection holes or defective joints of brickwork and tank. When quite pure gas is required, as is the case in certain operations, a special device which I will describe later is resorted to.

The temperature of working is 450° C., and may vary quite considerably without affecting the output. The temperature

of the electrolyte must, however, be maintained above the fusing point of metallic zinc (410° C.), otherwise the formation of trees of chilled zinc between the anode and the cathode occurs and soon stops the deposits. The distance between anodes and cathodes must be very small (from ½ inch to ¼ inch), as the resistance of these salts is appreciable in the presence of such heavy volumes of current. To obtain the best results the writer always uses an admixture of molecule of sodium chloride, which remains unchanged during the electrolysis of the zinc or lead chlorides. The results then obtained are substantially quantitive, and may be calculated with perfect safety for estimating purposes as "90 per cent current efficiency."

The voltage at which such a vat can be worked may vary considerably. The theoretical voltage of zinc chloride decomposition is 2.2, and of lead chloride 1.8, but in practice, (unless electricity is very dear) it will hardly pay to work at less than 4 to 5 volts. The resistance of these salts decrease as the temperature rises, and *vice versa*, so that a perfect automatic regulation is obtained within wide limits when the vats are worked in series; an arrangement always much more practicable and convenient than any other.

The cost of outfit for this process (exclusive of plant for the preparation of the chlorides) is very small indeed, owing to the simplicity of the apparatus and the high current density employed. The carbons, when the salts are quite pure, are not consumed at all, and the construction of the vat, being solid and having everything enclosed, there need be very little accidental breakage.

The labor of charging the raw material and discharging the zinc is also trifling. The cost of the operation is thus almost confined to the cost of electrical energy, for this item bulks so large in proportion to all other costs that these seem insignificant in comparison.

The energy required per ton of zinc per annum (with vats designed so as to require 5 volts total per vat and a yield 90 per cent current efficiency) will be

$$\frac{1}{1.4} = 0.715 \text{ hp}$$

for 1 hp.-year will produce

$$\frac{746 \times 1'2 \times 90 \times 8760}{100 \times 5 \times 1{,}000{,}000} = 1.4 \text{ metric tons.}$$

The probable cost of the cheapest average for water-power in large quantities in the near future (with capital charges added) will amount to, say, $10.00 per horse-power per year. The account will then be as follows:

Electricity	7.15
Upkeep and labor, say.....	2.5
Total cost of operating.....	$9.65 per metric ton.

The cost per ton of lead recovered by a similar process from lead chloride would be about one-third this figure, or, say, $3.00 per long ton.

These are figures which would compare favorably with most metallurgical operations were the conditions of raw material, supplies, etc., suitable. Moreover, products of the extremest purity can be readily obtained by these methods.

To bring such a process into profitable commercial operation on an extensive scale it will be necessary, therefore, either to have some extraneous source of pure raw material (such as might incidentally arise as an adjunct to some other industrial operations), or to create a cycle in which the salt suitable for electrolysis is produced cheaply and of sufficient purity.

Much of the writer's work a few years ago (in conjunction with the staff of the Metallurgical Trust, Ltd., resulting in what is now generally known under the name of the Phœnix process for treating mixed sulphide ores) was directed to this end. So far, however, market conditions have made it more profitable to supply the existing markets with zinc chloride manufactured by these methods by the aid of the cheaper gas obtained from electrolytic alkali processes, than to produce electrolytic zinc and regenerate the chlorine.

Though, as stated above, the direct fused electrolysis of certain salts is a simple matter, other salts or raw materials present greater difficulties. It frequently happens that it is necessary to agitate the electrolyte somewhat violently in order to prevent the settling out of solid matter dust or impurities of various kinds on the surface of the liquid cathode. In other operations it happens that it becomes necessary to gently mix the metal of the cathode or even to cause its systematic transference into and out of adjacent receptacles. An example of the first set of conditions is found in the process

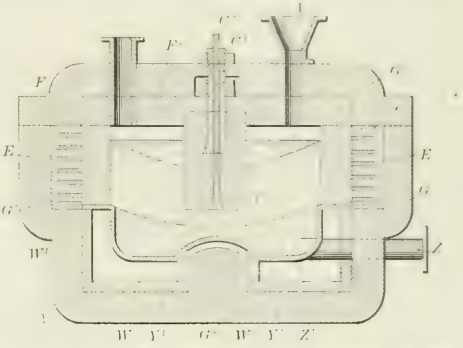


FIG. 2.—VAT FOR FUSED ELECTROLYTE. MAGNETICALLY AGITATED.

(original patent of J. Swinburne) for the systematic treatment of mixed sulphide ores by direct electrolysis of the ore suspended in fused zinc chloride. An example of the second order is found in the writer's recently patented two-cell sodium process.

To comprehensively meet these two, and other classes of requirements the writer has devised the apparatus, the latest improved form of which is illustrated in Fig. 2. This magnetically agitated cell will be here briefly described, its special applications being dealt with later on in more detail. For the purpose of general description I cannot do better than quote from the provisional patent specification, as follows:

"This invention relates to improvements in electrolytic cells or furnaces adapted to contain fused material, the object being to provide means whereby the contents of the cells may be agitated or transferred from one cell to another. The invention is based on the same electromagnetic phenomena as the process described in the previous British patent granted to me, No. 12,083, of 1905.

"The invention is applicable in the production of sodium lead alloys from fused materials, in the treatment of sulphide or other ores by electrolysis of the raw materials suspended in a fused medium, in the production of various alloys of metals from fused materials, and in other industrial processes.

"According to this invention an electrolytic cell containing fused material and having a substantially radial current is provided with an outer casing of magnetic conducting material, and a current-carrying coil is arranged within the casing to produce substantially axial lines of magnetic force in the cell.

"Preferably the casing of magnetic conducting material has a central pole projecting up from the base and is covered by a lid or yolk, also of magnetic conducting material, so that the lines of magnetic force outside the coil are conducted through the casing while the lines of magnetic force inside the

coil extend between the central pole and the lid or yolk in a substantially axial direction.

"Conveniently a metal pan, forming the base of the cell, constitutes one electrode and the coil connected to the pan and carrying the electrolytic current serves to generate the magnetic field.

"In order that the cell may be heated without interfering with the coil, the pan forming the bottom of the cell is so arranged within the casing as to leave a space beneath the pan for furnace gases.

"The adjustment of the magnetic exciting effect of the coil may be accomplished by varying the number of turns of the coil in circuit, and for this purpose the several turns of the coil may be connected to a commutator or the like.

"The accompanying drawing (Fig. 2) is a transverse vertical section of an electrolytic cell illustrating by way of example one method of carrying this invention into practice.

"An outer casing G of magnet steel completely encloses the cell, and is provided with a lid or cover G_1, also of magnet steel. Within a recess G_2 in the casing is arranged a coil E of heavy copper strip, which serves as a conductor for the electrolytic current, and generates a magnetic field in the cell. A central pole G_3 projects up from the center of the base of the casing, and the magnetic field is thereby concentrated into a stream of lines of magnetic force radiating from the pole to the bottom of the lid.

"A cast steel or other metal pan W having a flange W_1 at the edge and dished in the center forms the bottom of the cell, and is supported in the casing by means of the flange W_1 in such a position that flue space Y is afforded beneath the pan for a gas fire, furnace or the like. The lower end of the coil E rests on the flange W_1, and makes electrical connection therewith so that the pan forms one electrode or conductor for the cell. The pan may have an indentation above the pole G_3 to leave an air space between the pan and the pole.

"Inside the coil E the cell is lined with a ring of refractory material F, and the cover is similarly lined at F_1. The space in the recess G_2 between the casing, the lining and the coil may be filled with suitable insulating cement to ensure the rigidity of the whole. The flue space Y is provided with a refractory tile lining Y_1, and a tube of refractory tiles Y_2 protects the pole G_3.

"A metal inlet and outlet pipe Z passes through the casing into the cell. This pipe is preferably of oval section, and may be divided into two conduits (inlet and outlet) by a partition. The partition may project into the cell to form a baffle Z_1 for circulating purposes.

"The central electrode of the cell consists of a pole of carbon or the like C supported from the cover by a composite holding bolt C_1, having a central conductor C_2. The cell is represented as used for the electrolysis of a fused salt over a fused electrode. The passage of the main current several times round the interior of the magnetic steel casing sets up a powerful magnetic field in the cell, the lines of force being substantially axial. The lines of current flow between the electrodes are substantially radial, and the contents of the cell are thus caused to swirl violently, and this swirl may be deflected in any desired direction by suitably placed baffles.

"The advantages of this cell are: (1) That by placing the coil in the position shown, the bottom of the pan may be heated without interfering with the coil. (2) the screening effect of the steel pan is not detrimental to the magnetic effect in this form of cell. (3) the heat generated by the current carrying coil in the case is usefully employed to lessen the radiation of heat from the fused charge through the refractory lining and the cell wall, so that the cell works economically as to voltage and current. (4) the construction of the apparatus is simple, inexpensive and strong."

It will be seen that this is a modification of the writer's former method of placing a powerful separately excited magnet beneath the cell and providing a non-magnetic bottom to the apparatus, in order to allow the lines of force to pass freely.

I shall describe some experimental applications of the above principles in my next article.

Notes on Electrochemistry and Metallurgy in Great Britain

(From Our Special Correspondent.)

The Faraday Society.

In my last letter I only described the commencement of the Faraday Society's meeting on Jan. 30. Immediately on the finishing of Mr. Minet's paper followed Dr. Harker's furnace demonstation. Dr. J. A. Harker prefaced the demonstration of his furnace by explaining its construction. The furnace which he had to show was virtually an overgrown Nernst lamp, with a glower of 70 to 75 per cent of zirconia and 10 to 15 per cent of mixed yttrite earths. The furnace shown had been produced to avoid evolution of reducing gases near the platinum, as these always affected the thermoelectric properties. The tubes produced had a cold resistance of 1,000 megohms, and the resistance remained infinite until a temperature of 450° C. had been reached. At 1,600° C. the tube exhibited took 1 amp. at 1,600° C., and was arranged with a resistance in series with it across a 200-volt main. The best tubes were made by powdering up old tubes, mixing with a suitable binding material, and squirting the material from die at about the consistency of putty. The squirted tube was carried away on a traveling belt. In use it was found that the material softened at 2,200° C., but general life was fairly lengthy if kept under the melting point of platinum. As contact was a matter of vital importance, he had heated the tube externally with a spiral of nickel wire, which extended the life of the tube itself.

A sketch of the furnace was given in Vol. III. of this journal, p. 273. Having completely described the furnace, Dr. Harker then appealed to his hearers for enlightenment on one point. Had an theory been advanced by which one could calculate in advance the conductivity of any material? Upon the matter, Dr. Borns and Dr. Lowry gave references, and furnace questions poured in upon Dr. Harker. "Did the diameter of the tube affect the temperature attained?" Had he really succeeded in measuring the melting point of platinum, had he not been measuring the melting point of a platinum rhodium alloy?" "Had there been any trouble with the external nickel wire?" "Had magnesia tubes been tried?" To all of which questions Mr. Harker answered in a manner which the Proceedings will report at length. The temperature attained was not affected by diameters at present used. The melting point of platinum had been measured by placing fine platinum wire between a platinum-iridium and platinum-rhodium thermocouple. The platinum wire melted, forming a globule of pure platinum. The external nickel wire was fairly durable if encased in an outer tube. Magnesia and thoria might be used for tubes, but zirconia required a lower voltage. Gum had been used as a binding material.

The next item on the agenda was a note by Mr. E. B. R. Prideaux, "On the Production of Ozone by Electrolysis of the Alkali Fluorides," which was read but not discussed, owing to the lateness of the hour. (An abstract of this paper was given on page 168 of our last issue.—Ed.)

The Institution of Mining and Metallurgy.

Ferroalloys.

The papers presented for reading at the meeting held on Feb. 15, were purely of a mining nature, so that even their titles do not call for enumeration. The same envelope that contained these brought me the official copy of the discussion on Dr. Steinhart's paper, of which I made mention last

month. In that discussion many leading metallurgists participated, some with axes to grind; *i. e.*, alloys to boom, others, such as Mr. Hadfield, to discuss the paper in a general spirit.

Mr. Hadfield at the outset expressed his regret that there was at present a tendency to considerably advance the price of alloys and other metals used in the manufacture of special steels. His opinion was that unreasonable advances would seriously check the demand, and also turn attention to the substitution of orindary carbon steel. This might be a blessing in disguise, as no doubt in this steel, which was cheaper to produce, there were greater possibilities by careful heat treatment than was yet thought possible.

Concerning ferrochromium containing low percentages of carbon, in view of its higher cost, he did not see much advantage in its use. Nearly all steel required carbon, and therefore a certain percentage of it present in the ferroalloy was not injurious, especially as it enabled a cheaper alloy to be used. With regard to the use of tungsten for armor plates, he thought nothing had so far been done seriously. Chromium was a much cheaper element, and so long as they could get projectiles and armor plates by its use he thought makers would prefer to adhere to this metal. Of course, tungsten for high-speed tool steel was very valuable. The engineering world was much indebted to those men who had introduced tungsten by means of the ferroalloys described in the paper. Therefore, he thanked those who, like Dr. Steinhart, had done their best to bring these alloys to the notice of the steel maker.

Mr. J. Kent Smith took the discussion into a fresh region by discussing the tests to which steel should be submitted, and then dealt with vanadium steel, a subject specially investigated by him and by Captain Sankey, and described in a paper read before the Institution of Mechanical Engineers. He concluded by observing that in the development of the special steels, in whose preparations alloys, to his mind, *must* play a part, vanadium, assisted by chromium or nickel, or perhaps in some cases both, would have a "considerable amount to say."

Dr. Schack-Sommer discussed the price of nickel, and said that the most important feature in his mind in the extraction of nickel was a process which could deal cheaply with a nickel ore containing less than 6 per cent of the metal.

Mr. G. T. Holloway pointed out that, although chrome-iron ore was not readily saleable if containing below 50 per cent of chromium oxide, or only saleable at a much lower price per unit per cent than was the richer ore, the more friable varieties were more sought after than the denser mineral, and enquiries for 1,000-ton lots, apparently required for making furnace linings, etc., had been received for ore of from 40 to 42 per cent chromium oxides. In the case of molybdenite, the ore was often bought by inspection instead of by assay, and hand-picked material, such as was shown on the table, would often fetch even higher prices than the author had quoted. Ten years ago it was practically unsaleable, but the French were now using molybdenum by the ton for hardening steel. Dr. Steinhart had no doubt rightly placed tungsten third in the list of importance among the rarer metals, but he (the speaker) thought it was rapidly taking a more important position, as tungsten tool steel, etc., had properties as to hardness and retention of "temper" at high temperatures which none of the other steels possessed.

Mr. Benedict Kitto asked if Dr. Steinhart could inform them why it was that scheelite was so much objected to by the ferrotungsten manufacturers. He thought the author mentioned that it was because it was mixed with sulphur. The samples of scheelite that had passed through his own hands could have been easily treated.

Mr. A. F. Wiener discussed the sources of vanadiferous ores, and dealt with the poorness of the vanadium ores existing in California, pointing out that American users of vanadium obtained their supplies in Europe. This speaker suggested that the whole question of the employment of steel alloys was as yet barely beyond its infancy. We wanted to [illegible] the past generation. Armaments were required which were not dreamt of thirty years ago. It was quite clear that the properties of the steel of the past were no longer sufficient for all the needs of the present day, and totally insufficient for the probable future requirements. Improved materials must be found, and these would be made by the use of alloys. Chrome and nickel alloys were being used more and more, although, after all, much room yet remained for research and experiment. Vanadium has only come into notice within the last three years, but the results obtained by its use have been extraordinary. In the high-speed tool steels of the present day as much as 25 per cent of tungsten was being used. He thought that the study of these and other alloys was one of the most interesting subjects which recommended itself to the present generation.

Dr. Steinhart replied in detail to the various points raised, pointing out that low-carbon alloys were expensive articles, but they were necessary for certain steels, when it was a question of several per cent of the respective metal which had to be added to the steel, and at the same time it was necessary to keep the carbon low. He did not hold with these excessive prices, because, after all, if the manufacturers put too high prices on the metals and alloys the steelmaker would look round for others. The numbers of variations were innumerable. Mr. Hadfield's manganese steel, fortunately, did not require the use of very high-priced metals or alloys for its production, and, apart from this, it had superseded other more expensive steels (for instance, chrome steels) for many purposes. One particularly interesting use was the application of Hadfield's steel for mining machinery, the excellence of which he could testify to from practical experience.

In reply to the question of Mr. Schack-Sommer, he was not one who advocated the use of wet processes. He had seen a good many wet processes, but give him a straightforward smelting process, for it was always the best. That applied to the sulphuric acid process and the use of ammonia—all these processes had been tried, and he was afraid they had not come to much good. There was a good deal of garnierite shipped from New Caledonia under 5 per cent, and he remembered such shipments to the United States. He was much interested in what Mr. Holloway said with regard to chromite. The physical condition, such as hardness and the ease with which it is pulverized, was an important point. They all knew chromite ore was most difficult to crush, and this, for instance, was one of the expensive steps in the manufacture of chromates. What Mr. Holloway said as to the price of molybdenite he agreed with. There was no fixed price for such minerals, as often one person did not know what his neighbor paid for it.

He thought scheelite was not used so largely for this reason: in melting the ore with soda ash, as a certain percentage of lime had a tendency to get into the crystals, and as the hydrochloric acid often contained sulphates, these finally gave rise to bad metal.

A cordial vote of thanks to Dr. Steinhart brought the meeting to a close.

Market Quotations.

One or two slight irregularities have to be reported in regard to the market for chemicals. At the end of February ammonia sulphate is 3s. 9d. cheaper, at £12 7s. 6d. per ton; copper sulphate is 10s. per ton cheaper, at £24 10s., but zinc sulphate is unchanged, at £6 15s. The soda products are maintained at last month's quotations. For the coal tar products there is a good demand, and fractional increases in price have occurred. In the metal market, Cleveland warrants have closed at 49s. 2d., and hematite at 63s. 8d. The latest cash quotation for copper is £79. Tin has closed at £149 per ton. English lead is considerably cheaper, at £16 7s. 6d. per ton. Quicksilver fetches from £7 5s. to £7 7s. 6d. per 75-pound flask.

London, March 10.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

Industrial Electrochemistry.

Chlorates and Perchlorates by Electrolysis.—An interesting paper, by M. Couleru on this subject, is published in the *Chemiker Zeitung*, 1906, Vol. XXX., No. 21. Sodium perchlorate is produced electrolytically from sodium chloride as starting material. But sodium perchlorate has no direct importance for the industries, but only an indirect one as an intermediate product in the manufacture of potassium perchlorate and ammonium perchlorate by the reactions:

$$NaClO_4 + KCl = KClO_4 + NaCl$$
$$NaClO_4 + NH_4Cl = NH_4ClO_4 + NaCl$$

The perchlorates of potassium and of ammonium are used in pyrotechnics and in the explosive industries. A disadvantage of these perchlorates is their unsuitability for use in aqueous solutions, since they are no real oxidizing agents. Potassium perchlorate is only slightly soluble in cold water (15 grams per liter), which greatly facilitates its precipitation and separation from sodium chloride. Ammonium perchlorate is easily soluble in cold water (about 250 grams per liter); but since it is much more easily soluble at a higher temperature, it can be separated from sodium chloride without much difficulty. The perchlorates may be easily produced at a purity of about 99 per cent. The treatment, evaporation and crystallization of ammonium perchlorate solutions are best carried out in stoneware or enameled cast-iron vessels, steam being preferable to direct fire. The perchlorates themselves are not dangerous, but become so when strongly heated or when mixed with oxidizable substances. The manufacture of sodium perchlorate is carried out in two steps, the first being the production of the chlorate, the second that of the perchlorate.

All older processes of sodium chlorate manufacture involved considerable expenses in separating the chlorate from the chloride. This was done by evaporating the solutions containing about 300 grams $NaClO_3$ per liter. The expense was a big item, especially for manufacturers in the mountains, where fuel is expensive. All modern methods intend to avoid this expensive evaporation and complicated separation from the sodium chloride, and for this reason the manufacture is similar to the production of potassium chlorate. The electrolyte is not removed before the solution contains about 750 grams $NaClO_3$ per liter, care must be taken to always have sufficient sodium chloride present. Of electrolytic methods two are specially important for the practice, one using additions of calcium salts, the other using additions of chromate or bichromate and removal of alkali. Both methods give ampere-hour efficiencies of 80 to 90 per cent. In this way concentrated solutions of sodium chlorate are obtained which may be used directly for making perchlorate. But it is better to dissolve the raw chlorate which precipitates from these solutions and use the new solutions, since larger quantities of chloride diminish the output of perchlorate. The ampere-hour efficiency in the production of sodium chlorate is about 85 per cent, the watt-hour efficiency about 60 per cent. Platinum may be used for the anodes, iron for the cathodes. Cells up to 1,000 or 1,500 amps as maximum are practicable. The hydrogen developed during electrolysis may be used for heating and lighting purposes, but the proper precautions should be taken.

The electrolytic production of sodium perchlorate from sodium chlorate is very simple and efficient if the following conditions are fulfilled: platinum anode, any cathode (iron being best, because cheapest), concentrated solutions of sodium chlorate as electrolyte; presence of as little sodium chloride as possible in the electrolyte, also neutralization or removal of any alkali which may form, low temperature of electrolyte, preferably between 0° and 10° C., but certainly not above 25° C. If these conditions are fulfilled the ampere-hour efficiency is only slightly influenced by current density, additions of chromate, etc. It is possible to get in the beginning an ampere-hour efficiency of 95 per cent or more, but the efficiency decreases later on, a fair mean during the whole operation being 85 per cent. The only difficulty is to keep the temperature low; this may be done by pipes placed in the electrolyte and supplied with a cooling medium or by means of internally cooled electrodes.

Kryptol.—In *Elektrotechnische Zeitschrift*, March 1, J. Bronn gives a review of his work on the use of granulated carbon as resistor for electric furnaces. This he calls kryptol. (The material which is now in commerce under the name of kryptol is not simply granlated carbon but a mixture of carbon, graphite, sand, clay and other silicates, carborundum, etc.) The author's experiments were made with a uniform material of granulated carbon; he does not think that as long as the resistor operates properly, arcs are formed between the particles in contact, since all the characteristic features of the arc, like its minimum voltage, are here missing. He designates the phenomenon rather as a kind of "transition resistance heating." When such a resistor is used for the first time from a constant potential circuit, the ohms first decrease and later increase again until they reach again about the initial value. This period he calls the "starting period." All changes of the ohms after that are simply due to variations of temperature, and the latter depends essentially upon the question whether the loss of heat by conduction and radiation is greater than the supply of electrical energy or the reverse. The author has studied in detail what happens during the starting period, and has found that the decrease and increase of resistance in that time is primarily due to a development of gases within the resistor, which results first in an increase of pressure within the mass until the gases have time to escape. This phenomenon is troublesome in practice and should be avoided; this can be done by regulation of voltage; *i. e.*, by beginning with a low voltage and increasing it slowly, or secondly, by designing the apparatus in such a way that it has a large free surface, enabling the gases to escape. On the other hand, the addition of any materials which would develop gases or vapors increases the difficulties. In apparatus which are to be used for higher temperatures it becomes necessary to replace the resistor from time to time. The author recommends to use a material of as simple and uniform composition as possible in order to avoid uncontrollable changes.

Acetylene.—London *Engineering*, of March 2, gives an illustrated description of the Atkins dry process for generating acetylene. The essential feature is that the gas is produced by mixing calcium carbide with a dry substance containing oxygen and hydrogen—ordinary soda crystals being preferred—and not with water. With washing soda the reaction is as follows: $9CaC_2 + 2Na_2CO_3 \cdot 10H_2O = 9C_2H_2 + 4NaOH + 2CaCO_3 + 7Ca(OH)_2 + H_2O$. "The heat accompanying the reaction is very slight, not exceeding 95° C." and therefore much below that required for the polymerization of acetylene—namely, 400° C. To this absence of excessive heat is largely due the fact that the resultant gas is perfectly free from such impurities as benzene and other complex gaseous and liquid hydrocarbons. The apparatus is illustrated.

Alternating-Current Electrolysis.—A summary of investigations on this subject, together with a list of papers relating to it, is given by H. Danneel in the *Elektrotechnische Zeitschrift* of March 1.

Metallurgy.

Iron and Steel.

Iron Castings for Soda Melting.—V. Portisch, in *Stahl und Eisen* for January 15, gives some special directions for casting these kettles, using a built-up brick mold, which stands as much as thirty-five castings. The Kramatorska Works, in South Russia, sells its pots with a guarantee that they will stand forty meltings; some of them have stood as many as ninety-seven. The cast iron used contains 1.5 to 2 per cent of silicon, 1.3 to 1.38 per cent of manganese, 0.15 to 0.18 per cent of phosphorus, and 0.02 to 0.03 per cent of sulphur when melted in a reverberatory furnace, but 0.07 to 0.10 per cent of sulphur when melted in a cupola. The pots made from iron melted on a hearth averaged over twice the life of those made from cupola-melted iron. This points the way towards the use of a reverberatory or an open-hearth furnace in making these castings; makers of malleable castings, already equipped with such furnaces, could very profitably take up this line of work.

Cleaning of Blast-Furnace Gas.—Director Meyjes, of Zweibrucken, discusses very fully, in *Stahl und Eisen* for Jan. 1, the present standing of this question, the paper being a report of a lecture delivered before the Southwest Germany and Luxemburg Ironmasters' Society, at Saarbrüch, on Nov. 12. The ordinary furnace gas carries 8 to 15 grams of dust per cubic meter; gas engine builders require that the gas used in their engines shall not contain over 0.02 gram per cubic meter; over that they will give no guarantee of the durability of their machines. Cowper stoves and boilers, however, work much better with the cleaned gas. Director Bian, of Dommeldingen, calculates that the higher blast temperature obtained from stoves using cleaned gas, at his works, produced a saving of coke amounting to $9,000 a year, on a 100-ton furnace; but it does not pay to take the trouble to clean the gas for the stoves and boilers quite as clean as is required by gas engines. Illustrations and detailed description of the working of six different types of cleaning plant are then given. The cost of plant, to clean 60,000 cubic meters of gas per day, varies between $40,000 and $125,000; the running costs per year $5,000 to $7,500. The detailed article will be very interesting reading to blast-furnace managers and gas-engine constructors.

Briquetting of Iron Ores.—Prof. H. Wedding delivered before the Verein deutscher Eisenhüttenleute, at their Düsseldorf meeting, Dec. 3, an interesting lecture on this subject, which is reported at length in *Stahl und Eisen* for Jan. 1 and 15. The lecture was largely historical, but very complete, and while we cannot reproduce the descriptions, yet the classification of processes is so instructive as to what has been proposed, that we reproduce it entire:

I.—Briquetting Without a Binder.

1. Moistening with water:

Cornelia	mine ores, containing clay.
Used for	pyrites cinders at Langloan.

2. Pressure, without heating to sintering or melting

Cöln-Müsener Co.	Surface reduced.
Jacobi	Partial reduction
Ronay	Heating by gas.
Daelen	Blocks pressed into a binder.
Butler	Cast iron run around them

3. High pressure, followed by sintering:

Ronay	Very high pressure used

4. Sintering by high temperature:

Blezinger	In revolving furnaces
Edison	In chain furnaces.
Gröndal Co.	In rotating furnaces
Ruthenburg	In electrical furnaces.

5. Melting:

Wedding (1865)	In reverberatory furnaces.
Thau	In reverberatory furnaces, with or without fluxes.
Stein	Melting with reducing agents.

II.—Briquetting With a Binder.

A. *With Inorganic Binding Material*

(1) With iron ores:

(*a*) With clayey ores:

Concordia Works	Lump ore and clay mud.
Henzel	Pyrites cinders and clay mud.

(*b*) With brown hematites:

Kleist	Upper Silesian brown ores

(*c*) With purple ores

Georgs-Marien Works	Pyrites cinders and flue dust.
Langloan	Pyrites cinders and magnetic ore.

(*d*) With flue dust:

Georgs-Marien Works	Throat dust and pyrites residues

(2) With clay:

Henzel	Pyrites residues and clay.

(3) With lime:

(*a*) With limestone:

Lang & Frey (1860)	Ore, limestone, coke.
Berthier (1830)	Slag, coal-dust, limestone.
Königer	Limestone, borax, sulphuric acid.

(*b*) With burnt lime:

Edison	Burnt lime and clay.
Wedding	Burnt lime and carbon dioxide.
Duisburg Copper Co.	Lime and ashes.
Kleber	Lime, blast-furnace slag, hydrochloric acid and steam.
Kleber	Lime, silicate, steam.
Kleber	Lime silicate, chlorides, steam

(*c*) With slaked lime:

Straschitz	Milk of lime

(*d*) With gypsum or cement:

Cramer	Gypsum, lime, Portland cement
Löwenthal	Magnesium chloride and sand.
Lehmann	Magnesium sulphate and soda.
Reinke	Lime and cement.

(*e*) With lime silicate:

Schumacher	Wollastonite, from quartz and lime.

(*f*) With slag and water-glass:

Thomlinson	Cold blast furnace slag
Stein	Putting into fluid slag.
Kleber	Lime, slag, hydrochloric acid, steam.
Oberschulte	Blast furnace slag and steam
Schuchtermann	Basic Bessemer slag
Wust	Water glass
Mewes	Water glass and asbestos
Rouse	Water glass and steam

B.—*With Organic Binding Material.*

(1) With coal or lignite:

Mindary & Soudry	Coking with soft coal
Kerpely	Coking with soft coal
Wedding (1866)	With fat brown coal
Dobbelstein	With fat coal-dust
Landin	With soft coal

(2) With tar, pitch, asphalt, petroleum residue

Hoffelmann	Coke and pitch
Weissmann	Soft coal and pitch
Wedding (1890)	Asphalt or petroleum residue, with flue dust

(3) With resin:
Edison ..... Resin soap.

(4) With starch:
Marton ..... Starch, from corn, made fluid by pressure.

(5) With residues:
Fegan ..... Naphthalene and paraffine.
Trainer ..... Lignosulphuric salts

The lecturer gave a critical discussion of the present state of this industry, after which Dr. Weiskopf, of Hanover, and Prof. Mathesius, of Berlin, added interesting remarks. The conclusion seemed to be that no one process was the best for all kinds of ores, but that each process had its strong points for some ores, in some localities, and for some particular conditions and furnaces.

Open-Hearth Practice.—The practice of using larger and larger ore charges in making open-hearth steel is receiving more and more attention—and richly deserves it. The idea of oxidizing out carbon, etc., by the slow action of the gases, while iron oxide will do it as quickly as is needed and save iron from the ore into the bargain, is rapidly being seen in its true light. C. Dichmann, in *Stahl und Eisen* for Dec. 1 and 15, gives very instructive figures respecting this subject, the eighteen large pages deserving the closest attention of all steel makers. The writer first states that "the temperature necessary for carrying out the open-hearth process cannot be attained without the presence of an oxidizing flame in the furnace," and proves this by showing that during melting down of the scrap, the charge absorbed from the gases 2.33 kilos. of oxygen per minute, and while melted absorbed 1.6 kilos. per minute. This rate of oxidation being too slow, iron ore is used to hasten the removal of impurities, but at efficiencies which are variously stated as between 0.67 and 0.90. In a careful experiment, 3,276 kilos. of ore was put into a furnace with 1,000 kilos. of limestone and 20,300 kilos. of fluid pig iron. The results confirmed the general reaction of the Monell process, and the figures showed that 85 per cent of the iron in the ore used was reduced into the bath. The slag was brought down to 10 per cent of ferrous oxide. These tests prove that fixed furnaces can be used for the pig and ore process, that scrap is not a *sine qua non*, and that Mr. Monell was a pioneer in one of the most promising improvements in open-hearth practice

Electric Power for Iron and Steel Works and Rolling Mills.—In a paper recently presented by W. J. Belsey before the Dublin Section of the (Brit.) Institution of Electrical Engineers (which is printed in the London *Electrician*, March 2), some interesting notes are given on the use of electric power in rolling mills. Since the power varies very much, the fluctuations of power would react severely on the system if no special provision was made for storing the electrical energy at the periods of light load and giving it out at moments of maximum load. This method of storing and giving out energy must, of course, be automatic. A storage battery would do, but it is cheaper and more efficacious to install a buffer machine for storing the energy in the heavy fly-wheel. A diagram of the connections is given. An interesting feature recently adopted in one of the largest steel works in Scotland has been the use of the exhaust steam from the mill engine and steam hammers, etc., for driving a steam turbine. This is arranged by exhausting all the steam from the rolling mill engines, steam hammers, etc., into a calorific reservoir, which consists, in some cases of a covered steel tank, which is loosely filled with pig iron or some other substance which will readily absorb heat. From this the steam is passed through a turbine and exhausts into a very high vacuum. This idea has proved very successful in the utilizing of hitherto waste steam, for the steam which escapes under conditions of light load from rolling mill engines and steam hammers is practically live.

Iron and Its Alloys at Liquid Air Temperatures.—Perhaps the hardest task which can fall to an abstractor is that of dealing with such a paper as that presented by Mr. Hadfield at a meeting last year of the Iron and Steel Institute. The author of a paper who, without being in any degree redundant, writes a treatise of this length, might well be pardoned if he had slightly expanded it and issued it in more pretentious manner to the world in the shape of a cloth-bound octavo book. Considering that 1,600 separate observations were made concerning some 140 distinct alloys, it will be well understood that the space at our disposal compels us to present only in a condensed form the general conclusions in which Mr. Hadfield reviews the subject.

Extraordinary effects have been produced on all the alloys examined, many of which it would have been impossible to have foretold from any known laws. Test pieces were prepared and allowed to remain several minutes in liquid air, until all semblance of "boiling" ceased. This insured each bar being reduced to the temperature of liquid air, or — 182° C. With few exceptions the effect of this treatment is to increase in a remarkable degree the resistance of iron and its alloys to tensile stress, ordinarily known as the breaking load or tenacity, and to reduce its ductility, as measured by elongation, from the highest point; for example, in mild steel 30 to 40 per cent to practically nil. The changes take place to the same extent in the softest Swedish wrought iron, and also English wrought iron, and in carbon steel samples from 0.10 per cent or 0.20 per cent to the high percentages such as 1.25 per cent or 1.50 per cent. Thus the absence or presence of carbon in ordinary carbon steel in which other special elements are not present seems to have but little influence. That there is no error in this statement is proved, independently of the tensile tests, by the fact that several bars of the Swedish charcoal iron and mild steel specimens were submitted to the low-temperature test, and tested by hand-hammer immediately after immersion. In all cases they exhibited great brittleness, breaking off instantly upon being struck with the hammer; there was an entire absence of ductility. The Brinell hardness-ball test confirmed these results, it being found that the specimen of Swedish charcoal iron (designated as "S. C. I." throughout the paper), which at normal temperature had a normal hardness number of 90, had this increased to 266, or about equal to the hardness of 0.80 per cent carbon steel at normal temperature.

The addition of nickel has a marked toughening effect upon iron at low temperatures. On reference to the detailed tests it will be seen that the purest iron, as represented by the "S. C. I.," containing 99.82 per cent iron and of specially high quality and purity becomes brittle to an extraordinary degree under the influence of the low temperature — 182° C., whereas nickel itself tested at the same low temperature has improved rather than deteriorated, not only in tenacity, which iron also does, but in ductility, in which latter quality iron entirely breaks down. If nickel, therefore, is present in an iron alloy containing but little carbon, or comparatively low in that element, it acts as a preventive of brittleness, or is a very considerable modifier of that objectionable quality.

At ordinary temperatures the toughness or ductility of nickel is no greater than that of iron. For example, in comparative tensile tests, made by the writer, of nickel and pure iron, the ductility of iron was greater. The reduction of area in the material generally shows its condition as regards ductility; in the specimens in question the reduction of area in the tensile test bars was nearly 20 per cent greater for iron in both the author's "S. C. I." and Arnold's pure iron than in the nickel specimen tested.

Iron, to a more or less degree, at any rate in manufac-

turing operations, always seems to be endeavoring to wander out of the "paths" of ductility and toughness; it is constantly endeavoring to become brittle. It will often assume its apparently brittle nature on the slightest provocation, and the metallurgist, by his arts, is always trying to correct this tendency. As with humanity, there seems to be a law of tendencies, and iron by heredity is constitutionally weak. It would appear, therefore, that iron, a cheap and convenient metal itself, must be permeated by some element that will mask or modify its properties. Until comparatively recently carbon was the only element known to modify the properties of iron; but, as will be seen in this research, this element, where great toughness is required, only helps to make matters worse.

Fortunately for the metallurgist the addition of nickel permits of the maintenance of toughness and ductility at low temperature. The explanation of this is not easy, but Mr. Hadfield suggests that, "possibly some interpenetration of the atomic mass causes a change which cannot as yet be deducted by any known chemical investigation. Iron, too, is a very crystalline metal, whereas nickel appears to be much more amorphous; it is possible, therefore, that nickel tends to prevent iron crystallizing in this manner, or prevents it cooling in such large or dangerous type of crystals. This action of nickel is simply marvellous in certain of the alloy specimens; for example, in test No. 114, which is an alloy of iron, carbon 1.18 per cent, nickel 24.30 per cent, and manganese 6.05 per cent. Here the ductility is extraordinary at not only ordinary but low temperatures, probably the highest known for any iron alloy, and certainly for an alloy having such tenacity as 84 tons per square inch. There is still present in this alloy 68 per cent of iron, yet the tendency of the latter metal to wander into the paths of brittleness is not only entirely checked at the liquid air temperature—and this brittleness, as shown so clearly in this research, occurs to an extraordinary extent in pure iron cooled to — 182° C.; but the elongation or ductility, already so great, is considerably increased, namely, from 60 per cent to 67½ per cent. There is also an increase of tenacity in both cases, namely, a rise of from 10 to 38 per cent. Thus the nickel present—as these results cannot apparently be ascribed to any other cause—enables the bar under this high tension and at — 182° C. to remain far more ductile than the very best of ductile iron of one-third the tenacity. Although the action of nickel has been specially referred to, it must not be overlooked that in this alloy there is also present 6 per cent of manganese, which in its ordinary combination with iron, that is, with no nickel present, would confer intense brittleness upon the iron and render it more brittle than if not present. This treble combination of nickel manganese with iron appears to reverse all the known laws of iron alloys."

The various changes which result in pure iron being rendered so extraordinarily brittle, and in alloys of iron with nickel and manganese being free from these tendencies, are, in the author's opinion, "not chemical, and it is rather to the physicist that we must eventually look for a full and correct explanation of the many curious results obtained in this research."

Metallography.—Prof. H. Le Chatelier gives some excellent advice on the technics of metallography in the *Revue de Metallurgie* for July. The *coarse grinding* is best done by hand on sheets of emery paper; if it is moistened with turpentine the work proceeds faster, but coarser lines are left on the specimen; if moistened with soap or paraffin the grinding goes slower, but deep scratches are avoided. (American practice has found carborundum paper still more effective.) In *polishing* it is of the utmost importance that the powders used be carefully graded, for 1 per cent of coarser grains in a fine powder will spoil the effect of the latter completely. The No. 2 emery powder of commerce is sieved between two sieves of 150 and 200 meshes per inch, and only that portion is used for coarse polishing which falls between these sizes. Emery for finer polishing is obtained by flating the emery which passes the No. 200 mesh sieve. The finest polishing is done with alumina, made by precipitating ammonia alum, and washing carefully with water acidulated slightly with ammonia, and then with very dilute nitric acid. The precipitate is then suspended in a large quantity of water, and that part used which deposits between 3 and 12 hours. The support for polishing is a piece of fine flannel stretched tight over a glass plate. The flannel is impregnated with a solution of soap, powdered emery put on its surface, spread uniformly, and then allowed to dry. The polishing may then be done on the dry surface, or it may first be moistened with water.

Methods of Attack.—The best acid in general use is a 5 per cent solution of picric acid in absolute alcohol. The 4 per cent solution of nitric acid in amyl alcohol is almost equal to it. A mixture of 1 part of 4 per cent solution of nitric acid in ordinary alcohol, with 3 parts of a saturated solution of nitrophenol in ordinary alcohol is useful, in that it colors troostite and similar compounds, leaving the rest of the metal uncolored. By using boiling solutions of alkalies containing oxidizing agents, such as picric acid, cementite is easily colored without attacking any other constituents of steel. A 25 per cent solution of caustic soda, containing 2 per cent of picric acid, heated to 100° on a water bath, works beautifully.

Microscopes.—Prof. Le Chatelier has abandoned the use of the mercury arc lamp for illumination, because of its fragility and the length of time necessary for photographing, and prefers strongly the use of a large Nernst lamp with two filaments, each taking 1 amp. of current. The photographic camera is disposed horizontally. The details are very practical, and adapted for giving simple courses of instruction in this subject.

Furnaces.

Dimensions and Efficiency.—Dr. W. Borchers makes a splendid effort, in *Metallurgie* of Sept. 8, to compile data concerning the comparative dimensions and efficiencies of the most various kinds of metallurgical furnaces. Thirty-six roasting furnaces are compared, but with not much discrimination, so that the conclusions are only of the most general character; next, eleven lead shaft-furnaces, seventeen copper shaft-furnaces and ten copper reverberatories. The data are so numerous that the article does not admit of abstracting satisfactorily, but metallurgists in the copper and lead smelting lines will probably find the data and comparisons interesting, if not valuable.

Analysis of Current Electrochemical Patents

Electric Furnace. M. Ruthenburg, 815,221, March 13. Application filed Aug. 14, 1905.

The object is to reduce the consumption of carbon electrodes. A layer of inert material, for instance fused slag of bauxite, floats on the fused bath (steel, etc.), and a comminuted mass of coke is placed on the bauxite. The coke serves as resistor, the carbon electrodes dipping into it. The cheap coke is consumed in preference to the more expensive electrodes. The layer of inert slag transmits the heat to the metal bath below and prevents carbon from getting into the metal bath.

Refractory Material for Electric Insulation.—D. M. Stewart, 810,270, March 27. Application filed Feb. 28, 1903.

One hundred parts by weight of finely divided siliceous earth and twenty parts of waterglass are thoroughly mixed to a plastic or semiplastic mass of about the consistence of workable putty. The mixture is then subjected to heavy pressure in a hydraulic press, and dried in a kiln. It is finally charged into an extremely hard compound by heating it to from 1,500° to 2,000° C.

Electric Zinc Furnace.—W. McA. Johnson, 814,050, March 6. App. filed May 24, 1904.

This patent is interesting since it contains the first printed information on the extended experimental work done by the inventor at the works of the Lanyon Zinc Co. In contradistinction to the only electric furnace which is used on a commercial scale for zinc production (the arc furnace of De Laval) a resistance furnace is employed by Johnson. Two sections are shown in Fig. 1. The furnace walls (1) are constructed of fire-clay. The electrical connections are made through the side walls by means of carbon or graphite blocks (5, 6). These blocks are arranged in oppositely disposed pairs, one of these pairs being located near each end of the retort. The blocks (5-5) are connected to one pole (9), the blocks (6-6) to the other pole (10) of the electrical circuit. Over the hearth a layer (11) of refractory material (silica or high-grade fire-clay or bauxite) is distributed. Over this layer and across the furnace between the carbon blocks (55) and (66) loose masses of conductive coke (12) are arranged,

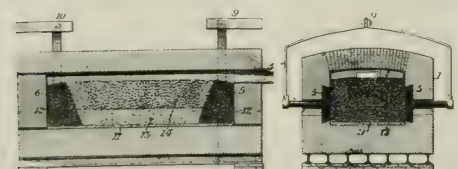


FIG. 1.—ELECTRIC ZINC FURNACE.

which constitute the active electrodes. Above the refractory layer (11) and extending upward against the side walls of the furnace, a body of high-grade ore (13) mixed with high-resistance coke is arranged, for instance roasted zinc ore containing 80 to 90 per cent zinc oxide with small percentages of iron, lime and lead, mixed with 8 to 10-mesh coke, together with sufficient fine coke to serve for the reduction. Above and within this high-resistance portion of the charge the main charge (14) is placed, consisting of a low-grade ore containing 50 to 70 per cent of zinc oxide, 15 to 30 per cent of iron, and smaller proportions of lime, lead, copper, etc., this low-grade ore being mixed with coke to form a charge of relatively low resistance. Both high and low-resistance portions of the charge are in electrical contact with the electrodes of coke (12, 12). The charges are so distributed and of such composition that the heat throughout the furnace is uniform. The layer of high-grade ore, containing little iron, protects the walls against injury from the low-grade ore; 4 is an outlet for the volatile products of the reaction.

Zinc From Sulphide.—F. T. Snyder, 814,810, March 13. Application filed June 23, 1905.

The inventor is connected with the American Zinc Extraction Co. The object is to produce zinc from blende without roasting. "The sulphide ore is mixed with carbon and with slag forming material, and the mixture smelted upon a bath of molten slag in an electric furnace from which air is excluded, whereby the metallic sulphide is broken down, liberating the metal in a free state and forming carbon bisulphide by the reaction of the sulphur and carbon." An electric furnace provided with carbon electrodes and tightly closed to the air during operation was started by heaping scrap lead between the electrodes and melting it down by the current. Slag forming material consisting of the zinc ore (in this case blende, containing 20 per cent Zn, 20 Fe, 5 Pb, 35 S, 20 earth: silica and alumina) mixed with iron and lime as fluxes was then fed into the furnace and melted to form a fused bath of slag, most of the lead being then tapped off. With 1,500 to 1,800 amps. at 7 to 15 volts direct current, the furnace was heated to about 1,200° C. The ore was reduced, metallic zinc being liberated in the form of vapor near one electrode, while carbon bisulphide was formed near the other electrode. "It is difficult to say to what extent the reduction of the sulphide is due to the electrolytic effect of the current as distinguished from the reducing action of the carbon at the high temperature employed, but electrolysis undoubtedly takes place, and I therefore prefer to employ direct current instead of alternating current." The inventor claims that it is entirely possible by his process to recover 94 per cent or more of the amount of zinc in the charge. To get the zinc in liquid form instead of in the form of zinc dust, the zinc vapor should be as far as possible free from diluting gases. With direct current the production of carbon bisulphide is confined to the neighborhood of one electrode, and the zinc vapor liberated near the other electrode can be kept substantially undiluted. "In the reduction of the zinc from sulphide ores to produce carbon bisulphide the proportion of the zinc vapor relative to that of the other gas is very much higher than when oxide ores are reduced and carbon monoxide produced. This is due to the fact that the production of a given volume of carbon bisulphide involves the reduction of approximately twice the quantity of ore which is reduced in producing the same volume of carbon monoxide."

Electric Furnace Process for Pig Iron Reduction.—P. L. T. Héroult, 815,016 and 815,293, March 13. Application filed June 14, 1905.

No. 815,016 relates to the process, No. 815,293 to the apparatus. The patents represent Dr. Héroult's chief ideas of increasing the efficiency of the electric furnace for pig iron production. The object is to insure that the gas escaping at the top of the furnace shall be all carbon dioxide, so that all the heat units developed by the carbon shall be developed within the furnace itself. The carbon is introduced at the base of the charge, and the latter fed into the top of the furnace without admixture of carbon. A mixture of carbon monoxide and carbon dioxide will be formed at the base of the furnace, and as this mixture of gases passes upward through the charge there will be no further consumption of carbon, but a further reduction of iron ore and a conversion of part of the carbon monoxide into dioxide. At a certain height, however, the mixture of gases is so dilute with carbon dioxide as to have substantially no further reducing effect. At this point air is blown into the furnace, and the carbon monoxide is thereby oxidized to dioxide. The higher the point where the air is blown in, the greater will be the utilization of the reducing power of carbon monoxide, though such reducing power is a diminishing quantity. The lower the point of injection of air the greater is the heat developed and the less the reduction of ore. The reduction of 1 kg. Fe from Fe_2O_3 requires 1,640 calories; the oxidation of 200 grams C to CO furnishes this energy; in addition the process requires sufficient heat to melt the iron and slag, namely, for 1 kg. Fe 250 calories, and for 0.5 kg. of slag 250 calories, a total of 500 calories; 200 grams of carbon and 1 electric hp-hour (= 630 calories) will, therefore effect reduction and smelting and provide an excess of 130 calories for losses by radiation and the heat escaping with the gases. The furnace is shown in Fig. 2. The pulverized carbon is introduced through the hollow electrode *B*, and forms at the base a sort of mushroom *O* surrounding the lower end of the electrode and serving, in fact, as the working electrode. The molten metal *P* collects at the bottom, and is drawn off from time to time through the tap-hole *A*. The molten slag *Q* floats on the top of the molten metal *P*, and shades gradually into the solid charge above. The oxygen

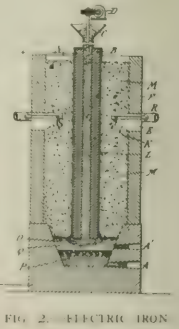


FIG. 2.—ELECTRIC IRON FURNACE.

of the ore combines with the carbon, while the iron is melted and trickles in drops through the lower part of the charge and the slag to the pool beneath. The carbon monoxide and dioxide formed with the oxygen of the ore then pass upward through the ore, taking oxygen therefrom in gradually diminishing quantities. The electrode *B* is protected by an iron shell against the carbon dioxide. The remaining monoxide is oxidized by air blown through the nozzles *E*.

Treating Copper-Nickel Matte.—W. McA. Johnson, 814,049, March 6. Application filed May 27, 1903.

The important industrial problem which this patent intends to solve was summarized in an editorial in our January issue, page 1, while two patents of Hybinette, relating to the same problem, were analyzed on pages 33 and 34 of the same issue. Like Mr. Hybinette, Mr. Johnson was formerly connected with the Orford Copper Co.

The copper-nickel matte to be treated may contain 39 per cent nickel, 39 per cent copper, substantially 1 per cent of iron and cobalt, and 20 per cent of sulphur, together with small proportions of platinum and palladium. The crushed matte is treated with hot sulphuric acid (concentration 5 per cent, temperature 80° to 100° C.), yielding hydrogen sulphide, a mixed solution of nickel, cobalt and iron sulphates, and a residue "R." The hydrogen sulphide is burnt to sulphur dioxide. The sulphate solution is freed from cobalt and iron by treatment with chlorine and caustic soda or sodium carbonate, or by the action of hypocholrites. The resultant nickel sulphate solution is treated with caustic soda, whereby the nickel is separated as hydrate. By means of neutralization with sulphuric acid nickel hydroxide is then dissolved and the nickel sulphate solution is electrolyzed, nickel being deposited on the cathode, and an equivalent amount of nickel hydrate being added to keep the solution neutral.

The residue "R" from the acid treatment of the matte consists chiefly of copper sulphide, some undissolved nickel sulphide and small quantities of platinum and palladium. This residue is roasted to oxide and then treated with sulphuric acid, yielding a solution containing, say, 9 per cent of copper and 1 per cent of nickel, in form of sulphates. This solution is electrolyzed with inert lead anodes, while the gases from the roasting operation are passed through the electrolyte. The gases serve the double purpose of reducing the necessary e. m. f. by their depolarizing effect and of yielding sulphuric acid needed later for leaching the residue. The electrolysis is continued until the copper content is reduced to about 0.8 per cent, and the acid electrolyte is then returned to the vats for leaching the roasted residue. Then follows again electrolysis, etc. When the nickel content in this copper electrolyte has risen to such an extent (say 8 per cent nickel) as to interfere with the deposition of copper, it is electrolyzed in a second series of vats with revolving cathodes until the copper is reduced to 0.4 per cent. The electrolyte is then returned to the vats for leaching the original matte.

Reduction of Galena.—C. P. Townsend, 815,881, March 20. Application filed April 12, 1902.

The object is to reduce galena in a fused bath, consisting of a haloid salt of an alkali or alkaline-earth metal. The lead is obtained in the fused state by cathodic reduction. By reason of the great difference in specific gravities the ore floats upon the surface of the lead beneath the electrolyte, and remains, therefore, in the field of most active reduction until it is itself reduced, and any irreducible impurities which may be present in the ore collect above the lead and are removed in any suitable manner. A high current density may be used and the reduction proceeds, therefore, with great rapidity. In operation the electrolyte is fused in a vessel and the charge of galena added, a cathode connection being made to the charge, preferably by means of a pool of lead upon which it rests. Anodes of carbon or graphite are located in the molten bath in proximity to the charge. As an alternative construction the lead ore may be placed in a basket or perforated holder, the cathode connection being made preferably to the central portion of the body of the ore. Upon the passage of the current the ore is reduced and the molten lead escapes through the perforations to the bottom of the fusion vessel. Since no oxygen is present in the bath or ore, anodes of retort carbon or graphite may be used without undue consumption. The sodium liberated from the molten bath combines at once with the sulphur of the ore, with the result that the electromotive force required for the decomposition of the electrolyte is correspondingly reduced. A certain amount of sodium sulphide is thus formed, which accumulates in the bath until it is sufficient in quantity to carry the current. Thereafter sodium sulphide alone is decomposed, sulphur distilling or burning around the anode, according as access of air is prevented or permitted. Since the sodium sulphide is decomposed at a lower electromotive force than the haloid salts, the power consumption decreases. Thereafter the electromotive force required is merely that necessary to overcome the ohmic resistance of the bath. Since the electrolyte is inert toward galena in all stages of the process, no metal exists in solution. This is important, as otherwise unavoidable losses of electrolyte would entail loss of metal.

Mercury Cathode Cell.—W. E. Harmon, 814,692, March 13. Application filed July 6, 1905. (Assigned to American Electrolytic Co.)

To secure the best results in producing sodium amalgam by electrolyzing common salt with a mercury cathode, the electrolyte should be kept free from an excess of hydrates, and it has been proposed to add acid to the electrolyte for this purpose. To accomplish the same end the inventor uses a small quantity of granulated carbon on the mercury surface (1 gram per square foot). This maintains the electrolyte acid.

Deep Etching of Zinc by Electrolysis.—O. C. Strecker and H. H. Strecker, 815,875, March 20. Application filed may 26, 1904.

In the electrolytic production of deeply-etched zincographs and relief-etchings for lithographic metal printing, troubles may arise from the use of impure zinc, resulting in an irregular solution of the zinc, so that a rough, unequally-bitten etched ground is obtained which takes the color and is printed off. This disadvantage is overcome by using a high current density, not less than 2 amps. per square centimeter of one-sided anode surface (maximum 12 amps.). A 10 per cent zinc acetate solution is used.

RECENT METALLURGICAL PATENTS.

Precious Metals from Zinc Ores.—In a patent recently granted (815,614, March 20) the late A. R. Meyer, proposes to avoid the expense of the double treatment, of which the first step is distillation of zinc and the second step a subsequent treatment of the residual matter for the recovery of the precious metals. In order to accomplish both results in one operation, the pulverized zinc ore is mixed with finely divided copper (or copper ore or lead or lead ore) and with carbon for reduction. The mixture is placed in a retort. The zinc distills off, while the copper acts directly upon the precious metal contents of the charge, extracting the same and combining therewith, forming a product either of metallic regulus or sulphides, forming matte, which may either gather in a mass, or, as is more usually the case, collect in globules throughout the charge in the retort.

Nickel. The well known Mond process consists essentially in heating nickel oxide in a reducing gas at a temperature between 350° and 500° C., cooling the material to 50° C. and subjecting it to the action of carbonic oxide gas, whereby the nickel is volatilized in the form of "nickel carbonyl" (our Vol. II., p. 201). In the original patent it is stated that while 50° C. is the preferable temperature for the carbonic oxide treat-

ment, yet it is possible to work at temperatures from 0° to 150° C. Carl Langer (815,717, March 20, assigned to Mond Nickel Co.) refers to the fact that on a manufacturing scale varying and low yields of nickel carbonyl have been obtained, for the reason that the range of temperatures from 0° to 150° is not at all permissible if the process shall be commercially successful. It is essential to keep the temperature carefully between 40° and 50° C. at atmospheric pressure. Merely cooling to 50° C. is also useless, since the reaction between nickel and carbonic oxide evolves heat so that the temperature would rapidly rise again. It is, therefore, necessary to apply artificial cooling. Langer describes apparatus for this purpose.

Precious Metals from Finely Divided Hydrometallurgical Products.—In refining the finely divided precipitates from cyanide, chlorine or hyposulphite solutions by the ordinary crucible method, considerable losses are encountered from "dusting." They may be avoided, according to C. W. Merrill (815,851, March 20), by the following process. The finely divided material is mixed with litharge, a soluble lead salt is added, and with the aid of a binding material—molasses or lime—briquettes are made. These can be introduced into the cupola without fearing any losses. In the cupola the litharge is reduced to metallic lead, whereupon fresh material is added periodically. The lead collects and contains the finely divided precious metals and is then cupelled off and recovered as litharge, to be used in subsequent operations. The precious metals are obtained nearly pure and may be cast or remelted into bars.

Modification of Cyanide Process.—T. B. Joseph (814,452, March 6) patents a leaching solution, containing water, sodium cyanide, hydrate of calcium, barium dioxide, ammonium bicarbonate, with the application of compressed air. The cyanogen in the solution extracts the precious metal, while the ammonium is intended to cause a better extraction of the silver as well as of the gold than could be obtained by the straight cyanide process. If there were no carbon with the ammonia in the solution, then the ammonia might decompose some of the cyanogen into its component parts and destroy its power to dissolve gold. If there are any of the ferrocyanides of zinc the ferrocyanide of copper, the zinc cyanide, the sulphur cyanide or the ferrocyanides remaining in this solution, any of them are also good solvents of the gold, as the barium and compressed air neutralize the injurious parts of sulphur and iron. Many of the injurious salts of iron, arsenic or sulphur are readily oxidized by the process or transformed into compounds insoluble or inert as to the solution.

Copper.—A. Elliott proposes the following process for winning copper from low-grade ores containing such quantities of bases (lime, iron, etc.) as to make them unavailable for leaching by acid solutions. He proposes to leach the ore with a non-acid, hot ferrous sulphate solution while air is passed through the solution. The copper, if in form of an oxide, will dissolve as sulphate, from which it is precipitated afterwards. The inventor gives the following reaction: First, action of air $6FeSO_4 + O = 2Fe_2(SO_4)_3 + Fe_2O_3$; second, leaching $Fe_2(SO_4)_3 + 3CuO = 3CuSO_4 + Fe_2O_3$; third, precipitation $CuSO_4 + Fe = Cu + FeSO_4$. The leaching solution is therefore regenerated.

BOOK REVIEW.

Cyanide Industry. Theoretically and Practically Considered. By E. Rosenhain and M. Loughlen. Translated by J. A. Lecher. New York: Wiley & Sons. Price $4.00.

The book comes at an opportune time, and fills in an adequate manner a real need. Beginning with a careful study of the chemistry of cyanogen and its compounds and derivatives, the reader is made acquainted in Chaper III. with the general properties of and with all methods of practical analysis for cyanides and cyanogen compounds. The end of Part I. is devoted to thermochemical data.

Part II. is a brief, and somewhat inadequate, treatment of the commercial and statistical sides of the subject. Here the statistics, though interesting, are out of date (the world's production of cyanide being stated no later than 1899, and no figures of consumption at all since the war in the Transvaal, for instance). Some information is given as to the actual and relative production, consumption and makers of cyanides, ferrocyanides, ferricyanides and sulphocyanides, but there is no connecting sequence to enable the general reader to reach a comprehensive grasp of all the bearings of this complicated industry.

Part III. is devoted to a very full and valuable account of actual industrial processes, used or suggested, for the manufacture of cyanides, ferrocyanides, sulphocyanides, cyonates and other salts of potassium, sodium et alia, finishing with some account of the Prussian blue industry. For this part of the book, which is about two-thirds of the whole, we have nothing but praise. A great deal of information hitherto unpublished, or only obtainable by searches in isolated records of technical "proceedings" and journals is placed at the reader's disposal.

Adequate treatment is given to three leading industrial methods in use for obtaining commercial cyanides. (1) From the gasworks by-products of ferrocyanide (chiefly by the aid of metallic sodium). (2) The so-called synthetic methods using metallic sodium, carbon and ammonia gas, of which Castner's process was the first, and the latest modification is the cyanamide process, due to the industry and enterprise of the Deutsche Gold-und Silber-Scheide Anstalt, whose American offshoot is the Roessler & Hasslacher Chemical Co., of Perth Amboy. (3) The nitric acid or nitrate and sulphocyanide methods, due to Raschen & Brock, and operated by the United Alkali Co., of Liverpool.

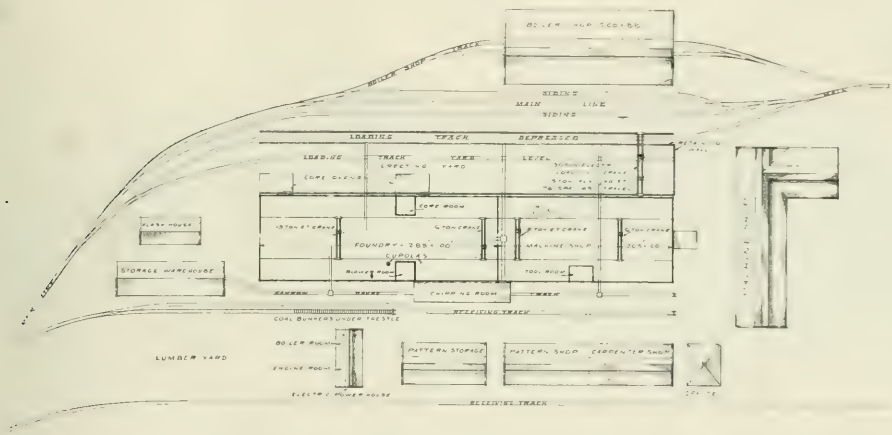
Part IV. treats of the *uses* of cyanogen compounds, and as far as it goes, is good. A very brief chapter of conclusions contains also a few useful comparative data of costs and values, and the general conclusion is drawn that the only real and abiding success attained amid the great rush of competitive processes rests with the methods using metallic sodium, either as a base for combination with ammonia and carbon, or as a reducing agent for ferrocyanide recovered from the spent oxides of gasworks. It is stated that the simplest of all methods, viz.: the fusion of ferrocyanide of potassium with metallic sodium, resulting in a double cyanide of 100 per cent quality (a method largely practiced by the Gas Light & Coke Co., of London, but attributed by the author to the American firm of Roessler & Hasslacher Co.), is economical when the cost of metallic sodium falls below 250 francs (French) per 100 kilos. (or 22.7 cents per pound). In fact, it would seem evident from the author's conclusions that the future of this industry largely rests on the commercial relations of the metal sodium. The concluding chapter finishes somewhat irrelevantly with some thermochemical and other tables.

The book concludes with a somewhat bulky and unnecessary reprint of some departmental summaries of metallurgical patents, very imperfectly related to the subject in hand. In this and other respects the book bears traces of amateur editorship, if not of too hasty compilation. In spite of these minor faults, and some small defects of printing and get up, it is an admirable work, and one which, coming as it does at a time when the expiration of the sodium monopoly is within sight, owing to the expiration of patents and other causes, and cheaper cyanide becomes, consequently, a possibility of the near future, it cannot fail to find a warm welcome in many quarters. We cordially praise the author's painstaking and able endeavors to present a text book of a class much needed.

New Works of the Traylor Manufacturing and Construction Company.

Several recent notable metallurgical plants designed and erected by the Traylor Engineering Co., of New York City, have brought this enterprising concern into the front rank of metallurgical interest; this is not surprising when it is known that the officers of the company are engineers, who have had a continuous practical experience of from fifteen to twenty-five years.

Last year, in our Vol. III., pages 122 and 163, we described at length the new mill of the Green Mountain Mining & Milling Co., of Silverton, Col., which is one of the most modern and efficient concentrating plants in this country. This plant is interesting not only on account of its compact design, which is due to the Traylor Engineering Co., but also on account of the extensive use of the Traylor centripact-screen. In one of our next issues we will give a description of a unique plant, also in Colorado, in which the final stage of operation



PLAN OF WORKS OF TRAYLOR MANUFACTURING & CONSTRUCTION CO.

will be closely allied with the latest developments of electrometallurgical engineering, while the crushing and leaching works were designed by the Traylor Engineering Co. These are only two isolated instances of the progressive work which the Traylor concern is now performing. A better idea of the extent of the business of this company may be had from the following brief description of their new works at Allentown, Pa.

Allentown was selected as the most desirable place for the works, after the resources of the country immediately adjoining New York City had been thoroughly tested. The company formerly had works at Belleville, N. J., but it seemed almost impossible there to solve the labor problem, since the class of workmen which could be depended upon for getting out the special line of products manufactured by the Traylor Co. could not be secured at that place. The Belleville works were therefore abandoned, and a special site was secured at Allentown with room for extensions. Entirely new works have recently been erected, where all the work of the company will be carried out in future. There are already a notable number of pig iron plants, foundries, machine shops, textile mills in and around Allentown, which, moreover, is the geographical center of the greatest cement industry in the world. The population of Allentown is made up mostly of Pennsylvania Dutch, who are exceedingly conservative in their views and [illegible] are good mechanics. [illegible] represented the best possible condition of the labor problem. On the other hand, the city is only [illegible] from New York City, and can be reached by three different railroads. It is 90 miles from Philadelphia, 5 miles from Bethlehem, 35 miles from Reading, and has direct railroad connection to several different lines with Pittsburg, the great iron and steel center.

The location of the works at Allentown has also brought the Traylor Co. into closer connection with the cement industry. The initial connecting link was the centripact screen, which we described in detail in our last volume, as noted above. While the screen was primarily designed for screening all kinds of ore products, for use in mines, mills and smelting works, it has now been found that it is quite as important in the manufacture of cement. The Traylor Co. has, therefore, decided to go into the general manufacture of all kinds of cement machinery, and by designing a modern and very powerful crushing roll to take the place of ball mills, heretofore used for reducing cement-making material down fine enough to pass a 10 mesh screen, the Traylor Co. has pushed its business into the cement industry, and it is expected that the use of the rolls will result in an appreciable saving. The company has also decided to build tube mills, kilns, sand-lime brick machinery, stone and rock crushing plants, etc.

The officers of the Traylor Manufacturing & Construction Co. are Samuel W. Traylor, president and treasurer; Bruce W. Traylor, vice-president; Frank W. Hopkins, assistant treasurer, and P. Edwin Van Saun, secretary and chief engineer, all of whom are engineers and have had long, continuous experience in the manufacture of their class of machinery.

The Traylor Engineering Co., of New York City, is made up of the same officers, and they own exclusively all of the stock of the Traylor Manufacturing & Construction Co., and all of the products of the Traylor Manufacturing & Construction Co. are handled exclusively by the Traylor Engineering Co. All contracts for plants are taken in the name of the Traylor Engineering Co. The two companies are, however, operated entirely independently of one another.

Mr. C. C. Knauss, one of the founders of the Bethlehem Foundry & Machine Co., and later manager of the Allentown Foundry & Machine Works, is the manager of the Allentown Traylor works.

In addition to the metal machinery business carried on by the Traylor Engineering Co., they act as consulting metallurgical and mechanical engineers for any one requiring their services. They are now designing some of the largest ore reducing plants in the United States. They have been closely identified with the designing and building of many of the most successful plants now in operation. At the present time they have under construction and nearing completion several large plants after their own designs.

The new works at Allentown consist of an office building, carpenter shop, pattern shop, pattern storage shop, power house, machine shop, foundry, flask house, storage warehouse, boiler shop, where sheet steel and blacksmith work are done, with large erecting floors and a lumber yard. The foundry is equipped with two large cupolas, one of which is probably the largest used in the vicinity of New York City, being of such a size as to make castings in single pieces weighing 50,000 pounds. The foundry is equipped with all the most modern devices, including electric traveling cranes, and, in fact, is built along the most approved designs, for the purpose of producing castings at the least possible cost, and is of such a size as to enable the company to have an output of from 40 to 50 tons per day. In this branch of their business alone the company will employ 150 to 200 men.

The machine shop and the foundry are each equipped with two traveling cranes running the full length of each shop, and covering the full width of the main working floors. One crane in each shop is of 6 tons capacity, while the other is of 15 tons capacity.

The machine shop is equipped along lines especially suitable for the production of the special class of machinery to be built. No expense has been spared in acquiring tools that are particularly adapted for their purposes, and many tools were made for the company's special purposes. This shop contains [illegible] machines weighing 75,000 to 100,000 pounds each, such as boring mills, open side planers and large lathes. All [illegible] to the use of the heavy overhead traveling cranes. [illegible] the handling of the product by machinery [illegible] a small consideration in cost of [illegible]. The [illegible] and other smaller machine tools have [illegible] for them [illegible].

The [illegible] built within it, and in the [illegible] there will be [illegible] room. Our [illegible] [illegible] [illegible].

The [illegible] machine [illegible] [illegible].

ALLENTOWN WORKS OF TRAYLOR MANUFACTURING & CONSTRUCTION CO.

company to load heavy machinery on outgoing cars, and to unload all incoming cars at a nominal cost. This ponderous crane is of the greatest importance in cheapening the cost of manufacture, especially with such products as are made by this company.

The power plant consists of two 204 hp. water tube boilers, an air compressor, and two Corliss engines, direct connected to two 100 kw generators. The current is transmitted to the different parts of their works for operating the machines. The entire plant is operated electrically, including all of the numerous hoists, large and small. A compound air compressor furnishes air for all of the chipping and riveting hammers, and to every part of the works where its use is required.

A general warehouse and storeroom, 100 feet x 300 feet, is now in course of construction along the railroad siding.

All of the buildings are strictly modern, being constructed with brick walls and roofs of slate. They are provided with ample light in every part.

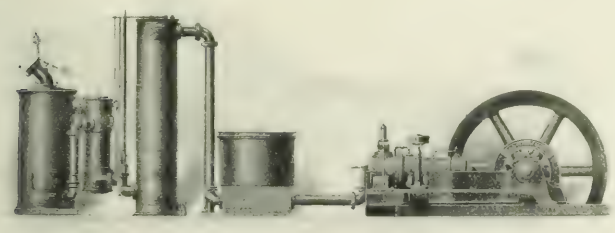
The company has a most favorable location in the matter of railroad facilities, having 1,800 feet along the railroad tracks, and a block of land consisting in all of about 10 acres. The haulage arrangements are perfect. They now have under contemplation plans for largely increasing the size of their plant in the near future. Including all the amount of floor space in the new works is now about 100,000 square feet.

Suction Gas Producers.

By Oskar Nagel, Ph. D

While in Germany, suction gas-power plants are a decided success, which can be seen from the fact that Koerting Bros., for instance, have sold over 4,000 plants in the last four years, some suction gas-power plants built in this country have proved a failure. The main reason that some of these American plants have not been successful is to be looked for in the gas engine. American gas engine builders are accustomed to work with highly efficient gases, such as natural or illuminating gas, and when building their engines for the use of producer gas they have just made some slight changes, which are not sufficient to insure good and continuous working with producer gas. A large number of American gas engines are built very light and working with high speed. A producer gas engine, on the contrary, has to be heavily built, and if to be used for continuous work it has to be a slow-speed engine. As producer gas is hardly ever absolutely pure the valves have to be so arranged that their working is not interfered with by small impurities. For places with changing load an extremely heavy fly-wheel with great momentum has to be used so as to help the producer over the first period of change. Furthermore, a good mixing valve has to be attached to the engine, as without this the economical running of the engine is impossible.

All these points have been carefully considered by Koerting Bros. in constructing their engines. Their mixing valve supplies the cylinder with a perfect combustible from the first revolution and greatly facilitates starting, which gives this engine an inestimable advantage over other types. The mixing



SUCTION GAS PRODUCER PLANT

valve is so constructed that the stream of incoming gas is split up into a number of small streams flowing at right angles to the incoming air, this mixture being again passed through another set of ports, which impart to it a whirling motion and insure perfect mixing of the two constituents before they are admitted to the cylinder.

A 75-hp. Koerting engine and suction producer, such as shown in the cut, is now running in the works of the De La Vergne Machine Co., New York, and consumes less than 1 pound of coal per brake horse-power-hour. The engine is a 4 cycle single-cylinder engine, running at 162 r. p. m. The producer plant consists of producer, evaporator, wet scrubber and sawdust scrubber. The producer is a sheet-iron shell, lined with fire-brick and provided with poke holes and double hopper, also with large fire and ash doors. The evaporator is of the tubular type, and is provided with an extremely well designed air inlet. The wet scrubber is of the ordinary type, and provided with three doors, while the sawdust scrubber contains two wooden trays, one covered with shavings and the other with sawdust.

It is probable [illegible] plant engineers, while thoroughly familiar with steam power, are not so well informed relative to gas power. [illegible] fore, ought to take special care in buying gas power plants to get thoroughly posted regarding the [illegible] engine and producer that is offered to them. In forming a decision the intrinsic value of the plant [illegible] considered, and not the more or less inconsiderable saving in the first cost.

If this had been done with the plants [illegible] this country, much disappointment and trouble would have been avoided.

Apparatus for Lifting Solutions by Compressed Air.

However beautiful and sound in theory [illegible] lurgical process may be, its successful carrying out on a commercial scale will mainly depend on the proper construction of the mechanical apparatus involved in its operation. Such little things like stoppage of a valve or leakage of a joint may cause innumerable troubles and the continued repetition of accidents, due to small mechanical defects, has more than once relegated a good proposition to the scrap heap. But even in commercially successful processes, the losses due to interruptions by mechanical accidents are not always small, and the cost of maintenance and repairs is often [illegible]. For this reason any decisive improvement in the construction of indispensable apparatus is always a matter of importance.

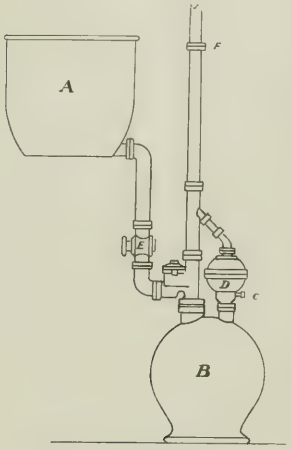
In the chemical and metallurgical arts special problems must be solved on account of the special properties of the materials and solutions to be handled. The manufacture of chemical stoneware apparatus—which in this country and abroad represents an industry the magnitude of which is perhaps not generally realized—is a special illustration of this necessity.

A problem with which this industry has been confronted for many years is that of lifting chemical solutions. Its importance is indicated by the large number of apparatus which have been devised and introduced into practice.

The Deutsche Steinzeugwarenfabrik (German stoneware factory) für Canalisation and Chemische Industrie in Friedrichsfeld, which is one of the leading manufacturers of stoneware apparatus, has paid special attention to this problem for a long time, and has placed on the market at least [illegible] or five essentially different designs, each representing a distinct improvement over the preceding one. The last apparatus, which is described below, is considered by the manufacturers to have reached the climax, and to fulfill all requirements of practice in a perfectly satisfactory manner. It has, therefore, been brought into commerce under the trade name "Friedrichsfelder Druckaufzug Ideal."

Its main features are great simplicity [illegible] to the fact that a single ground sphere of stoneware [illegible] the whole regulating device; perfect safety and ease of operation, since the apparatus is almost proof against breakage, and the number of joints which might cause leakage is reduced to a minimum; small, easy accessibility of all parts and [illegible] automatic operation without loss of compressed air. A type of [illegible] dent should occur. The most important advantage of the new system, however, is that it needs no special adjustment; depend-

ing upon the specific weight of the liquid. The apparatus may be used for liquids of any specific weight, and for lifting it to any height without requiring any changes. This means that if the apparatus has been used for lifting an acid of high specific gravity to a comparatively great height, it may be used immediately afterwards for lifting a lighter liquid to a much lesser height without changing anything in the apparatus.



APPARATUS FOR LIFTING SOLUTIONS BY COMPRESSED AIR.

The new apparatus may be used, for instance, to great advantage in connection with condensation towers, where it is necessary to repeatedly lift the dilute acid flowing off from the bottom of the tower up again to the top, in order to increase its concentration.

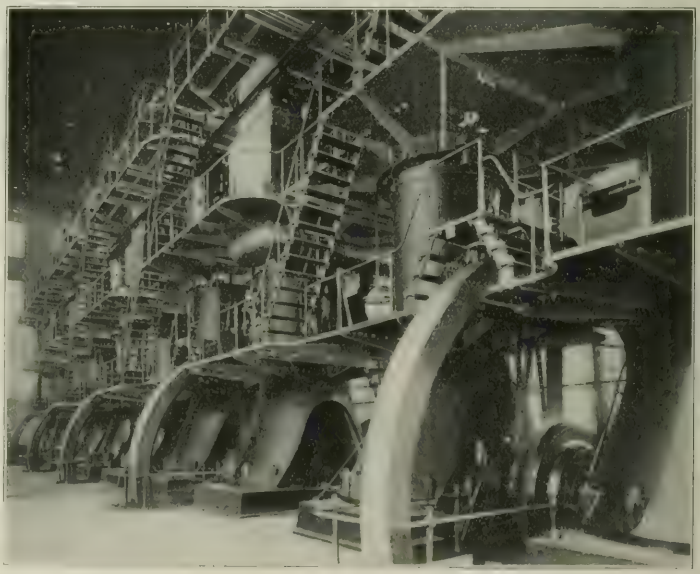
The construction of the lifting apparatus may be seen from the adjoining diagram. *A* is the reservoir containing the solution which is to be lifted. It passes through *E* and opens a valve at the junction of tube *E* and tube *F*, and enters into the vessel *B*, thereby driving the air from the vessel through a passage in *D* upwards into the tube *F*. The connection *C* to the compressed air reservoir is closed during this period. When the liquid reaches the top of *B* it acts automatically on a valve by which the passage is closed through which the air before escaped through *D* upwards, while simultaneously the connection *C* with the compressed air supply is opened. Compressed air now enters *B* and drives the liquid up through *F*, the valve at the junction of *E* and *F* being now automatically closed. The liquid is thus lifted upwards in pipe *F*, and this goes on until compressed air enters the pipe *F*. The valve in *D* then automatically reverses the two connections, shutting off the compressed air supply *C* and permitting the air to escape from *B* through *D* upwards, while simultaneously the valve at the junction of *E* and *F* is opened again. New liquid enters now from *A* into *B* and so on. The apparatus is built at present in four sizes. The smallest size is able to lift up to 800 liters (about 210 gallons) per hour, the second size from 600 to 2,500 liters (160 to 660 gallons) per hour, the third size 2,000 to 5,000 liters (530 to 1,320 gallons) per hour, and the fourth size from 4,000 to 10,000 liters (1,000 to 2,700 gallons) per hour.

In cases where hot liquids are to be lifted, or under such conditions where a stronger pressure is required than is generally sufficient, the company advocates the use of corundum instead of stoneware. Corundum has a resistivity against pressure and tension about 30 per cent higher than stoneware, and is also much less effected by temperature changes. This application of corundum appears to be a new and very interesting development, although mention was already made of it in a paper by Dr. Hans Goldschmidt before the International Electrical Congress in St. Louis (our Vol. II., p. 405).

The quantity of compressed air which is required in any case may be found, together with its tension easily from the following rules. The quantity of compressed air equals the quantity of liquid to be lifted in one operation increased by about 8 per cent. The required tension of the compressed air equals the specific gravity of the liquid multiplied by the height of lifting plus 1/10 atmosphere for every 10 meters of height of lifting when referred to water. For instance, we may take the case of the third size of the lifting appartus, the contents of which is 180 liters of liquid. If the specific gravity of the liquid is 1.4, and if it is to be lifted 12 meters high, there will be required 180 plus 8 per cent = 195 liters of compressed air and a tension equal to $1.4 \times 12 + 1.4 \times 1.2 \times 0.1$, that is about 2 atmospheres. The apparatus is introduced in this country by Messrs. F. Bertuch & Co., New York City.

Blowing Engines.

In our report of the Bethlehem meeting of the American Institute of Mining Engineers, in our last issue, we mentioned the large blowing engines, installed in 1905 by the Allis-Chalmers Co., at the Hokendauqua plant of the Thomas Iron Co. Since they are an excellent illustration of the recent tendency of steel and iron producers to install the most efficient and economical machinery available, a few additional notes on these blowing engines may be given here.



BLOWING ENGINES.

These blowing engines are of the well known vertical steeple type, having steam cylinders 30 to 80 and both air cylinders 84 inches diameter, all with a stroke of 60 inches. This type was originated by Mr. Reynolds, of the Allis-Chalmers Co.,

many years ago, forming the original for the modern engines now in the market. In these blowers (as in the standard or long cross-head class of the Allis-Chalmers Co., the air cylinders or "tubs" are placed above the steam cylinders, and are each of the same size; *i. e.*, 7 foot-diameter by 5-foot stroke. The two cylinders thus have a free air capacity of 750 cubic feet per revolution.

Engines of this large size each weigh nearly a million pounds, and are almost always vertical, since the weight of their enormous working parts would produce great friction if of horizontal design. The comopund steam cylinders, when run in connection with a condenser, blow the air with the least expenditure of steam, and the heat thus saved is available for other purposes. There are now a large number of these engines in operation in the great iron and steel plants of the country.

The adjoining photograph shows a typical line of the steeple-compound blowing engines built by the Allis-Chalmers Co.

Notes.

American Electrochemical Society.—At the March meeting of the board of directors the following gentlemen were elected to membership: H. J. Williams, Philadelphia, Pa.; Henry Lacroix, Geneva, Switzerland; Samuel A. Tucker, Columbia University, New York City; Wm. H. Nichols, New York City. At the April meeting the names of the following gentlemen will come up for election: Ellis W. Bacon, Philadelphia, Pa.; J. L. Wills, Brooklyn, N. Y.; Edward E. Free, Ithaca, N. Y.; B. G. Klugh, Margnette, Mich.; Philip C. Walsh, Jr., Newark, N. J.

Journal of Explosive Industries.—Vol. I. No. 1, of the *Zeitschrift für das gesamte Schiess-und Sprengstoffwesen* has been issued on Jan. 1, 1906. Dr. Richard Escales, in Munich, is editor, J. F. Lehmann's Verlag, Paul Heyse Strasse 20, Munich, are the publishers. The journal is bi-monthly, and the subscription price for this country is 26 marks. The journal is intended to be international in the sense that articles will be published either in German or French or English. The first number contains, however, only articles in German

Errata.—In Dr. Franz Meyer's article, published on pages 94 to 96 of our last issue, the following typographical errors should be corrected: Page 95, left-hand column, fourth paragraph, second line, read partition instead of partially; right-hand, seventh paragraph, first line, read Schmieder instead of Schurieder; page 96, fourth paragraph, eleventh line, read Cd instead of Co; thirteenth line, hence instead of these; fifth paragraph, fifth line, liquid instead of liquor.

Metallurgy and Electrometallurgy at Lehigh University.—Until 1898 the course in metallurgy at Lehigh University formed a part of the course leading to the degree of engineer of mines. Owing to increased demand and facilities for specialization, two separate courses were established in 1898, each of four years' length; one in mining leading to the degree of engineer of mines (E. M.) and the other in metallurgy, leading to the degree of metallurgical engineer (Met. E.). In 1902 a further step was made by establishing a separate course in electrometallurgy. There are thus two separate courses which are offered in the department of metallurgy, each of four years' length; one in metallurgical engineering, leading to the degree of metallurgical engineer (Met. E.) and the other in electrometallurgy, leading to the degree of electrometallurgist (El. Met.); both courses are under the direction of Prof. J. W. Richards. A full programme of the different courses is given in a recent pamphlet, entitled "Courses in Metallurgy," and published by Lehigh University, South Bethlehem, Pa.

Welding of Pipes by Thermit.—The Goldschmidt Thermit Co., of New York City, has recently issued an illustrated pamphlet giving instructions for welding wrought iron and steel pipes by utilizing the heat of the thermit reaction. The principles of the method of butt-welding pipes have repeatedly been described in our columns. The new pamphlet is specially interesting in its description of an improvement of the method, since instead of sheet iron molds, which have heretofore been used for that purpose, cast-iron molds have been substituted which considerably simplify the manipulation. The company has in stock the molds for the [illegible] wrought-iron pipes. This application of the thermit process has found considerable use in the welding of ammonia pipe lines on streets, as the welding can be done anywhere "in situ," while the perfect weld obtained in such cases by the thermit reaction always results in a very appreciable saving..

International Electrical Congress at St. Louis.—The total expenses of the International Electrical Congress at St. Louis in 1904 were $11,392.53, the total receipts from membership fees, sales of Transactions, etc., were $13,438.90. This excellent financial result was unexpected. As a matter of fact, an apparently inevitable deficit was turned into a credit balance by instituting at the right moment a canvass for the sale of copies of the Transactions. The executive committee of the Congress has now decided to offer the balance of $2,046.37 to the library of the American Institute of Electrical Engineers, as a fund in perpetuity commemorating the St. Louis Congress, the annual proceeds to be applied to the purchase of international electrical literature. Mr. W. D. Weaver was the resourceful treasurer of the Congress.

Refractories.—The high pressure at which metallurgical developments are now proceeding is indicated by the statistical figures for the consumption of refractories. The Harbison-Walker Refractories Co., in Pittsburg, reports that the orders received in February were 25 per cent larger in tonnage than those received in January, and nearly double the tonnage in September, 1905. This February's orders were the largest ever received in the history of the company during the past four years.

Testing Ores.—The Colorado Iron Works Co., of Denver, Col., has recently equipped its ore-testing plant with the most modern automatic machinery. The plant is of such a capacity that it is really an exceptionally well appointed mill, arranged in such a manner that tests can be carried out by any process, without handling the ore during any part of its course through the machinery, in exactly the same manner as it would be treated in a mill under actual service condition. These tests are carried out under the supervision of the experts of the staff of the Colorado Iron Works Co. An interesting illustrated description of this testing plant has recently been issued by the company in pamphlet form.

It is rumored that the **Automatic Refrigerating Co.**, of 22 Thames Street, New York, which company represents the merged interests of the Automatic Refrigerating Co., of Cleveland, Ohio, the Singer Automatic Ice Machine Co., of Bridgeport, Conn., and the Marshall Ice & Refrigerating Machine Co., of Boston, Mass., is about to remove their office, shop and refrigerating show plant to 630 Capitol Avenue, Hartford, Conn., where provision has been made for greatly increased manufacturing facilities. The Automatic Co. occupies a rather unique position in the refrigerating field in that it makes a specialty of small automatically-controlled equipment for which most of the 200 odd manufacturers of refrigerating and ice making machines do not care to bid.

Electric Furnace Process for Steel Making.— Mr. R. H. Wolff, 445 Broadway, New York, is the American representative of Dr. Paul Heroult for his electric smelting furnace and process. Mr. Wolff has been for many years a maker of high grade steels and wires in this country, and was during the last five years the foreign representative of the Crucible Steel Co. of America. While the recent Sault Ste Marie experiments

of Dr. Héroult on the reduction of pig iron from ore in the electric furnace (see page 124 of this issue) are most interesting, and will undoubtedly attract much attention, it is to be expected that in this country the chief application of the Héroult furnace will be in connection with the manufacture of steel, where it has undoubted advantages, as was repeatedly pointed out in this journal. It is also to be expected that the Héroult electric mixer (see page 30 of our January issue) will prove a very valuable adjunct in steel mills with large output of staple products, like billets, rails, structural steel, etc., since for these purposes it will be possible to get with the electric process a superior and more uniform quality on a more economical basis than heretofore.

Gas Producers.—Loomis-Pettibone gas generating plants are attractively described by means of diagrams and colored plates in a new pamphlet of the Power & Mining Machinery Co., Milwaukee, Wis. This system gasifies bituminous coal, wood, charcoal, lignite, etc., and produces a gas that is particularly adapted to the operation of gas engines. The water-gas made by this system can be used separately in various feeding work, such as forging, welding, tempering, etc. This pamphlet can be had upon application to the company.

Fire-Brick.—We have received from the Kier Fire-Brick Co., of Pittsburg, Pa., their illustrated catalogue on Salina fire-brick. The analysis of Salina fire-clay is 43.750 per cent silica, 40.966 alumina, 0.769 sesquioxide of iron, 0.095 lime, a trace of magnesia, and 14.410 combined water. Velvetry, stock hole and bridge tile for heating, puddling and air furnaces are a specialty of this company. Velvetry tile takes the place of the old 9-inch arch in the neck of the furnace. Stock hole tile takes the place of arches over the doors. Bridge tile is so constructed as to prevent the bridge from splitting and making a 6 or 13-inch bridge. Besides useful general information, the catalogue contains a long series of dimensional diagrams of tiles and furnace blocks.

Giant Copper Matting Furnace.—The Dominion Copper Co., of Boundary Falls, B. C., has just placed an order with the Traylor Engineering Co., of New York for a giant copper matting furnace. The furnace will measure 48 inches x 204 inches inside the jackets at the tuyere line. The water jackets will be of steel, flanged and riveted, with no seams or rivets exposed to the fire. They will be arranged in two tiers, one above the other. The furnace will be entirely self-contained. It is one of the largest ever designed, and will have a capacity of 350 to 400 tons a day. The Traylor Engineering Co. will also furnish a large blower to go with the furnace.

Glass Industry.—A recent consular report from Germany refers to the process of a Belgian inventor, Mr. Fourcault, whose patent is stated to have been bought for $952,000 by the European syndicate of plate-glass manufacturers. "The new invention draws the molten substance from the pot and conducts it between rollers lying side by side. Seventeen pairs of these rollers are built up tower like above the pot. The liquid mass cools on its way between the rows of rollers and comes out from them, polished on both sides, in any desired thickness (this being regulated by the relative position of the rollers), beautifully flattened and ready for use."

Concentration.—The North American Lead Co., Fredricktown, Mo., in making additions to its present plant, recently purchased two No. 3 Overstrom concentrating tables, two right and three left hand, and fitted with copper riffles. These tables, which are built by the Allis-Chalmers Company, Milwaukee, were described and illustrated in our Vol. III., p. 218. In addition to the new concentrating machinery, the North American Co. has also added to its electrical equipment in the purchase of a 150-kw. "Bullock" type "H" direct-current generator, with small motors, type "H" and "B," aggregating a hundred horse power. The electrical apparatus will be furnished by the Bullock works of the Allis-Chalmers Co., at Cincinnati.

Belt-Conveying Machinery.—We have received from the Robins Conveying Belt Co. a set of nine single pages, which by means of excellent illustrations facilitate greatly the understanding of the operation of belt-conveying machinery. The illustrations show typical applications of belt conveyors in actual practice in various metallurgical industries, and give a much better idea than a mere description could give. To mention only one example out of the whole series, Robins belt conveyors are very useful and economical as tailings stackers. In California, on gold dredges and at the large gold mines in the West and in South Africa, the Robins Conveying Belt Co. have installed a great number of their conveyors. This set of illustrated pages should go far towards getting almost as good an idea of the operation of belt conveyors in numerous cases as could otherwise be had only by inspection of plants in running order.

West Allis Shops of the Allis-Chalmers Co.—The progress of construction on the $3,000,000 extensions to the West Allis works of the Allis-Chalmers Co., in spite of delays of various kinds, is steadily being improved. Some idea of the unusual size of the new structure which will bring the total floor space for the entire plant to 1,513,000 square feet, may be gathered from the bill of material used in construction. Some of the largest individual orders ever placed by a Milwaukee concern have been given for material used in the building operations. Nineteen thousand barrels of Portland cement will be required for the concrete work, 785,000 feet of roof sheathing will help cover the works, and 850,000 feet of yellow pine and white oak will support the huge weight of structures and equipment. Even the scaffold lumber used only temporarily during construction will aggregate 10,800 pieces. Over 3½ acres of wire glass will be used in skylights and windows, while the weight of new equipment for the interior of machine shops will aggregate 3,080 tons.

Big Induction Motors for Anaconda.—Four of the largest induction motors ever installed in the West, and some of the largest in the country, with the exception of those installed at Niagara, will be used to furnish power at the $9,000,000 reduction plant of the Anaconda Copper Mining Co., in Anaconda, Mont. The Allis-Chalmers Co., of Milwaukee, will furnish this electrical equipment for the mammoth 8,000-ton concentrator and the Street Railway Company at Anaconda. The concentrator equipment will consist of four 1,200-hp. induction motors, three 300-kw. motor generator sets, together with the usual auxiliary accessories. This order is only one of many which will follow the determination of the Anaconda Co. to substitute Missouri River power, transmitted 90 miles at 70,000 volts, in place of the present power system installed in the largest smelter in the world. Each of the big induction motors will weigh over 40 tons, the weight of the entire equipment when ready for shipment being approximately 250 tons. The motors will be used in the concentration building, which is 600 feet long, through which a main line shaft extends the entire length. The motors will be rope connected to this shaft and will carry the entire power load.

Oil Engines.—The largest order ever placed for oil engines has recently been awarded to the De La Vergne Machine Co., of New York, by the Baldwin Locomotive Works, of Philadelphia. This is for engines aggregating over 3,300 actual horse-power. Some of these are to be installed in Philadelphia, and the remainder at their steel works at Burnham, Pa. The installation will consist of 125 and 250 hp. Hornsby-Akroyd oil engines, and they are to be used for direct connection to electric generators and to air compressors, while others will be used for operating machine tools by belt. To characterize the efficiency of the Hornsby-Akroyd oil engine it is stated that with crude or fuel oil at 2 cents a gallon, the cost of operation is ¼ cent per actual horse-power hour. The De La Vergne Machine Co. now holds the record of having sold the largest oil engine power plant, the largest ice plant and the largest oil engine power plant in the world.

Rock Crushing Machinery.—The Power & Mining Machinery Co., Milwaukee, Wis., have recently shipped a No. 9 McCully rock and ore breaker to the General Crushed Stone Co. for their plant at Rock Hill, Pa. This machine is to meet the most severe conditions of service ever before exacted of any rock and ore breaker—the crushing of the hardest known grade of trap rock. This No. 9 McCully will replace several machines of other make which were found unsuited for the arduous work imposed upon them. As a matter of fact, the material to be crushed is so hard that it practically ruined every machine heretofore used, through the breaking of the shafts, spiders, gearing, etc., as fast as new ones were put in. In addition to this No. 9 McCully breaker the Power & Mining Machinery Co. are also building for this same concern, for use in their plant at North Leroy, N. Y., a No. 10 McCully breaker, which is one of the two largest rock and ore breakers ever built, the first one having but recently been shipped to the Little Falls Stone Co., Little Falls, N. Y. In this connection attention may be called to catalogue No. 4 of the Power & Mining Machinery Co., since it deals with rock crushing machinery, and especially with the McCully gyratory rock crusher. The catalogue also contains illustrated notes on elevators, screens, hoists, etc., and sketches of complete crushing plants. In order to better handle their rapidly increasing business from the Rocky Mountain district, the Power & Mining Machinery Co. has opened an office at 312 Seventeenth Street, Denver, Col., which will be in charge of Henry F. Jurs, as district manager for Colorado, Wyoming and New Mexico.

Oxygen Generator.—The Roessler & Hasslacher Chemical Co. has now placed on the market an oxone generator of low-pressure type. It is a portable apparatus, for the instantaneous production of oxygen by means of oxone. The apparatus furnishes on an average 14 to 15 gallons of gas, and weighs, completely packed, about 8 pounds. The oxygen gas obtained is stated to be of 100 per cent purity. A circular recently issued by the Roessler & Hasslacher Chemical Co. describes the generator and gives instructions for its operation.

Engineering Publicity.—H. M. Baxter, engaged in engineering publicity with offices in the Lewis Block, Pittsburg, has recently added to his staff Chas. W. Brooke and Frank B. McConnell for the technical preparation and efficient distribution of catalogues, articles and advertisements for manufacturers of engineering practice. Mr. Brooke was formerly connected with the sales department of the Westinghouse Co., and Mr. McConnell was for a number of years engaged in illustration and catalogue work with the publicity department of the same company.

Asbestos for Wire Insulation.— Judge Archibald, of the Circuit Court of the United States for the middle district of Pennsylvania, has decided the infringement suit of Mr. Louis W. Downs *vs.* the Teter-Heany Developing Co., in favor of the defendant. The decision is that while the two processes of Downs and Heany have the same end in view—the insulation with asbestos of magnet wires—they are distinct and different and both may stand.

Wattmeter Suit.—The United States Circuit Court for the Northern District of New York, a few days ago, filed a decree enjoining the defendant in the case of the General Electric Co. *vs.* the Madison Gas & Electric Co., from using alternating-current integrating wattmeters or other motor devices embodying a provision for securing an approximate 90 phase adjustment by induced circuit. The infringement, which formed the basis of this suit, was the using of Sheefer alternating-current integrating wattmeters, manufactured by the Diamond Meter Co., of Peoria, Ill. This is the same patent which E. B. Latham & Company, of New York City, dealers in Sheefer wattmeters, was enjoined on Nov. 4, 1905.

Concentrating Plant.—The Colorado Iron Works Co., of Denver, Col., recently entered an order for a 100 ton concentrating mill, complete erected, for the Mowry Mines Co., near Patagonia, Ariz. The equipment includes Colorado Iron Works standard rolls, impact screens, simplex tables, vanners, spitzkastens and settling cones. This company also reports the sale of impact screens to the Atlas Lime Mining Co., Kimberly, Utah; Mine La Motte Development Co., Missouri; the Bertha Mineral Co., Pulaski, Va.; the Hermosa Copper Co., Hanover, N. M., and bullion cars to the Great Western Ore Purchasing & Reduction Co., Keeler, Cal.

Thermometers.—Messrs. S. L. Baily, Jr., and F. Stuhl have organized the Philadelphia Thermometer Co., with offices at Arch and Ninth Streets, Philadelphia, Pa. The company will manufacture thermometers, pyrometers, barometers, hydrometers, hygrometers and chemical and physical glassware.

Electric Pyrometer.—We have received from Mr. Wm. H. Bristol two bulletins on the construction and applications of his thermoelectric pyromter, which was described on page 115 of our March issue. After April 1 the offices of Mr. Wm. H. Bristol will be at 45 Vesey Street, New York City.

Election.—At the recent annual meeting of the Wellman-Seaver-Morgan Co., of Cleveland, Ohio, the office of general manager, which has been vacant since the death last June of Mr. Charles H. Wellman, was filled by the election of Mr. S. H. Pitkin, whose present title will be first vice-president and general manager. Otherwise no changes were made in the officers of the company.

Sale of Massena Power Plant.—The Pittsburg Reduction Co. has taken over the St. Lawrence River Power Co., which owns the large water-power plant at Massena, N. Y. The Pittsburg Reduction Co. has already had for several years a large aluminium plant at Massena, and was the principal customer of the power plant.

Price List of Chemicals.—A valuable feature of the price list of Messrs. Charles Cooper & Co. is that not only the prices are given but also indications whether the prices are advancing or declining.

Bulletins.—The Allis-Chalmers Co. has recently issued the following publications: Bulletin 1,049, on Bullock railway generators for direct current; instruction book No. 5,003, on the installation and operation of Bullock direct-current motors and generators; instruction book No. 5,000, on setting and operating style D Gates rock and ore breakers; Bulletin No. 1,203, on Allis roller mills; Bulletin No. 1,702 on carriage feeds for saw-mills; Bulletin No. 1,705, on transfer machinery for saw mills.

Calendars.—We have recently received several slightly belated calendars which are noteworthy in one or the other respect. The calendar of the F. J. Stokes Machine Co., in Philadelphia, is very neat, in green and white colors, and shows several illustrations of machinery and apparatus built by this company for the chemical and metallurgical industries. The calendar of the Laclede Fire-Brick Mfg. Co., of St. Louis, elaborates in greater detail the diabolic theme of last year's calendar of the same company, and is even more striking and more devilish.

Obituary.

Prof. R. Ogden Doremus.—With deep regret we note the death of Prof. R. Ogden Doremus, in New York City, at the age of 82. Member of old English and Dutch families of Manhattan Island, he graduated from New York University, and as early as 1848 had equipped a laboratory. He was one of the founders of the New York Medical College, the Long Island Hospital Medical College, and Bellevue Hospital Medical College, urging in each case the necessity of providing a chemical laboratory, where the students could have thorough chemical training. From 1853 to 1861 he was professor of natural history at the Free Academy, now the College of the

City of New York, and afterward was appointed to the chair of chemistry and physics in the same institution. He made the laboratory at Twenty-third Street and Lexington Avenue, a famous center for study, particularly in connection with electricity. His museum there was filled to overflowing with early electrical apparatus and instruments of great historical and technical value as bearing upon the early development of electrical industry. On one occasion, at the Academy of Music, in 1855, when lecturing, he took daguerreotypes of all the persons in the boxes by an arc light, and exhibited an induction coil with a 6-inch spark, a marvelous achievement in those days. His pioneér achievements in sanitation, toxicology, chemistry, etc., were not less notable. He was also a founder and president of the Philharmonic Society. One of his sons, Dr. Charles A. Doremus, is well known to our readers as a most distinguished leader in electrochemical developments.

Personal.

G. F. Brindley.—Our readers will be interested in learning that Mr. G. F. Brindley has resigned his position as works manager of the Niagara Electrochemical Co., and is going to Japan, where he will be associated with his brother, Mr. H. Brindley, who for several years has carried on a general engineering agency at Tokyo. Mr. G. F. Brindley has been intimately associated for many years with some of the most interesting of the electrochemical industries. He was born in Birmingham, Eng., in 1864, and in 1874 went with the rest of his family to Tokyo, where he was educated in the Imperial College of Engineering, taking his degree in engineering and chemistry. While at this institution he had the good fortune to study under such men as Milne, Alexander, Ayrton, Perry, Divers, etc. In 1883 he went to England, and then to California in 1886, but two years later returned to England, and there became associated with Mr. H. Y. Castner. As Mr. Castner's assistant Mr. Brindley worked on the manufacture of sodium by the distillation process, on the production of aluminium from the double chloride of aluminium and sodium, on the electrolytic production of sodium, on the manufacture of sodium peroxide, on the manufacture of cyanides on the electrolytic process for making caustic soda and chlorine, not to mention numerous other less important work done in Mr. Castner's laboratory. In 1896 Mr. Brindley came to this country, and became works manager of the Niagara Electrochemical Co., which manufactures sodium, sodium peroxide, etc., under the Castner patents. Within the last year Mr. Brindley has worked out successfully the difficult problem of producing fused sodium peroxide on a commercial scale, which has been put on the market under the name of "Oxone," and is used for generating oxygen by simple immersion in water. He has also been experimenting with Dr. R. von Lengerke in an interesting research on the regeneration of air by means of sodium peroxide. Mr. Brindley is a charter member of the American Electrochemical Society and was among the Niagara Falls delegation which attended the inaugural meeting in Philadelphia in 1902. His many friends here and in England will certainly join us in wishing him God speed and much success in his new enterprise.

G. F. BRINDLEY.

Digest of U. S. Patents.

Prior to July, 1902.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

Production of Ozone and Miscellaneous Gas Reactions.

No. 87,155, Feb. 23, 1869, William Elmer, New York, N. Y.

Purifies, warms or cools and ozonizes air for use in dwellings. The apparatus comprises a purifier, in which the air passes over shelves wet with a mixture of lime-water and potassium permanganate, to remove carbon dioxide and oxidize impurities. In winter, it is then passed successively through an air heater, ozonizer and thermobattery connected to the ozonizer. In summer, the chemical solution is cooled by ice, and thereby cools the air. The cooled ozonized air passes over one set of junctions of the thermobattery, while the other junctions are heated by a gas flame. The ozonizer comprises a metallic sphere surrounded by a horizontal metallic ring, which has points extending toward the sphere. The apparatus, termed a "Climozonator," is said to consist substantially of "an ozonizer, a therma and a frigidarium," and to "produce a pure, genial and invigorating in-door climate at all seasons of the year."

No. 100,736, March 15, 1870, C. F. Dunderdale, New York, N. Y.

An ozonizer consisting of parallel plates or concentric tubes of glass, silvered or coated with foil on one side. The alternate plates constitute opposite electrodes. They are arranged in a box lined with glass, and having an inlet and outlet at opposite ends. The electrodes may be of metal spaced by glass pieces.

No. 109,601, Nov. 29, 1870, C. F. Dunderdale, New York, N. Y.

An improvement on the preceding, the electrodes being provided with fine projecting points of metal, opposed to the flat surface of the adjacent electrode, or to other points thereon.

No. 167,844, Sept. 21, 1875, Thomas W. Lion, Brentsville, Va.

Alleges a beneficial effect in the manufacture of illuminating gas by subjecting superheated steam and highly heated vapor of naphtha, after mixing, to the successive action of a voltaic cell and a thermoelectric battery. The voltaic cell comprises a wire-gauze basket, suitably insulated, containing table salt and provided with platinum conductors, the chamber around the battery being packed with "laminated" carbon or graphite. The generation of electricity is alleged to follow the contact of the hot vapors with the elements of this battery.

No. 186,207, Jan. 16, 1877, J. G. Hunt, New York, N. Y.

Steam and vapors of petroleum are treated for the production of illuminating or heating gas, by subjecting them to the action of a battery "of any suitable character." One terminal appears to be immersed in the petroleum tank and the other grounded.

No. 254,424, Feb. 28, 1882, T. J. Yost, Mahwah, N. J.

Portable ozonizing case comprising a battery, coil and ozonizer. The ozonizer comprises a gas tube, near one end of which is mounted a small fan driven by clockwork. The tube is externally wound with wire, constituting one terminal, the interior terminal comprising a centrally-supported metal rod, carrying a number of metal discs with serrated edges, the rod being covered with insulating material between the points of contact of the discs. A larger form of device is described for use in the ventilating systems

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The Expiration of the Hall Patents.

On April 2, 1889, five patents were granted to Mr. Charles M. Hall for the production of aluminium, numbered 400,664, 400,665, 400,666, 400,667 and 400,766. All these patents relate to the electrolysis of alumina dissolved in fluorides. By their expiration on April 2 of this year the composition of the only electrolyte used in this country and abroad for the commercial production of aluminium has become public property. This does not affect, however, the commercial monopoly of the Pittsburg Reduction Co. for the production of aluminium in this country. While the electrolyte is public property, yet the method of heating the electrolyte by the electric current itself —which is unavoidable when working on a large commercial scale—will be protected by the Bradley patent till 1909. The outcome of the Bradley patent litigation, which was covered in detail on pages 519 and 560 of our first volume, now begins to become a blessing for the Pittsburg Reduction Co., since it has almost three more years to further strengthen its already extremely strong industrial position. The enormous increase in capacity of its works in recent years has only been sufficient to just meet the increasing demand. As a matter of fact, there is now almost a scarcity of aluminium in the market. While the boom of the automobile industry and the increased use of aluminium for electric conductors has much to do with this situation, yet it is certain that the better understanding of both the excellent qualities and the limitations of metallic aluminium has resulted in an increase of consumption on all lines, and that this increase will continue at about the same rate during the next years.

New Developments of the Fused Electrolytic Bath.

It is supposed that the chief trouble with the Salom lead reduction process was the smallness of the unit. This evidently has not been successfully overcome by the new continuous cell. The process was so attractive at first sight with the recovery of sulphur and the small amount of power used per ton of raw material, that the slowness of commercial development must be attributed to some cause, and this is the most likely one. Three different experimental metallurgists have endeavored to accomplish the same end as Salom had in view, but with the use of larger units by recourse to the fused electrolytic cell. In the Analysis of Current Electrochemical Patents in our April issue we noted a patent granted to Mr. Clinton Paul Townsend on this general subject of electrolytic lead reduction. In the Townsend electrolytic lead process the cathode employed is molten lead, on which rest the particles of galena. These particles are subjected to "cathodic reduction" while the sulphur is liberated at the anode. The action can also be described as the electrolysis of sodium sulphide formed in the cell. The sodium then reacts on the galena with the reformation of sodium sulphide. In either case the end

result is the same. Any impurities in the lead ore are removed "in any suitable manner." Such is the outline of the Townsend process which has much in it to be commended and in the recovery of sulphur as by-product. Two other similar processes are described in our current issue—the Betts-Valentine, in an article by Mr. Anson G. Betts, and the Swinburne-Ashcroft process, in an article by Mr. Edgar A. Ashcroft. In the Swinburne-Ashcroft process the decomposition of zinc sulphide in suspension in the fused electrolyte of mixed zinc chloride and sodium chloride is employed, and also the electrolysis of lead sulphide dissolved in a fused electrolyte of lead chloride. Mr. Ashcroft believes that, notwithstanding the cheapness of the old furnace process and also the "Scotch Hearth" process, that the Swinburne-Ashcroft process is commercially promising. We will comment on the very interesting work done by these two distinguished British electrometallurgists at greater length in a later issue, after the conclusion of Mr. Ashcroft's articles on his work with fused electrolytes.

* * *

Mr. Betts has made extended and most interesting metallurgical experiments on analogous lines. He gives an exhaustive analysis of the future prospects of his processes in the article on electrolytic lead smelting. In the short course of a few years, Mr. Betts has developed, in spite of obstacles incident to any new process and in spite of the amused tolerance of older metallurgists, his electrolytic lead refining process to a successful commercial basis, where he has two plants of fairly large size in successful operation, while the third one is soon to be started. Therefore, anything that Mr. Betts publishes is worthy of careful consideration. The object of his work was to dissolve galena from a high-grade lead concentrate in a bath of an easily-fusible electrolyte. The electrolyte selected was lead chloride, and thus the process is similar to the Swinburne-Ashcroft process. It was thought at first the process would be commercial, because of the recovery of the sulphur as a by-product, and also because of the fact that there would be practically no loss in lead. It was reckoned that this would give a saving of from $7 to $8 per ton over the old, tried smelting processes.

* * *

However, in spite of such a first favorable outlook the same old spoke in the wheel- "the accumulation of impurities in the electrolyte" (forcibly and clearly elucidated by Mr. David H. Browne in former issues of this journal) blocked this process, as it has blocked in the past many other promising electrolytic processes. Accordingly it was seen that some method must be devised to furnish the cell with a worked-up purer material. Just as in the Hall process, pure alumina must be made by a chemical process, so Betts' attempt to prepare a refined lead sulphide—practically a lead matte to feed to his electrolytic lead cell. There is an exact parallel between the aluminium process and Betts' proposed lead process. And again we have the evidence of the truth of the generalization that electrometallurgical methods are especially adaptable to the smelting reaction. Mr. Betts has deviated, however, somewhat from the general line of work, and has determined to make matte by electric melting, and thus to eliminate the impurities by a smelting process. He also proposes to combine the preheating of his charge by coal in a reverberatory furnace and finishing the matting in an electric furnace. Very careful thermochemical analyses are given of the two operations, and comparison is made with the costs by the old smelting process. An important by-product is, of course, sulphur, and the combined treatment has an additional advantage, due to increased recovery. Considered all in all, the Betts process for electrolytic lead smelting is quite attractive, and although it will be undoubtedly the subject of much criticism by other engineers, nevertheless, it is only right to make proper allowance for the success of the inventor in other lines. There is a striking similarity in a general way between the proposals of Townsend, Swinburne, Ashcroft and Betts. All of their proposals are interesting and illuminating, in the light of future possibilities, and also in showing striking analogies to other processes. We hope to publish in the course of time additional matter in this new field of electrochemical development.

The Electric Furnace for Steel Refining and Iron Reduction.

We are authoritatively informed that after some unavoidable delay in equipping the plant, the Héroult electric steel refining process is now finally in successful operation at the works of the Halcomb Steel Co., in Syracuse. The electric furnace was first started on April 4, with cold charging materials; a week later the regular operation began, superoxidized steel being supplied in molten state from a Wellman furnace to the electric furnace for refining. From the very start the operation has been satisfactory in every respect. There has been no hitch whatever, so that the success of this furnace, which is the first large electric steel furnace in this country, is complete. We offer our hearty congratulations to both Mr. Halcomb and Dr. Héroult.

* * *

In our last issue we gave an account of the successful outcome of the experiments made by Dr. Héroult at Sault Ste. Marie, with the cooperation and under the supervision of the Canadian Government, represented by Dr. Haanel. These experiments related to an entirely different metallurgical problem, namely, the reduction of iron from ore. This problem is, of course, a much bigger one and of vastly greater commercial importance than steel refining. On the other hand, it is certain—and we have repeatedly discussed the reasons why—that the electric furnace will be of very much less commercial importance for the problem of iron reduction from ores than in the steel industry. It can no longer be doubted that for refining steel, for making all kinds of high-grade steels, to compete with crucible steel, the electric furnace is specially well adapted, and in this field it will reign supreme within a very few years. There also seems to be a bright future for a large electric steel mixer, to secure uniformity in the product of an entire works, as proposed by Dr. Héroult and described on page 30 of our January issue. Further, it is not impossible that a suggestion made by Dr J. W. Richards, in his address to the American Electrochemical Society two years ago, will be realized in practice; namely, the use of electric heat as an auxiliary with our open-hearth steel furnaces to take off the "peak of the load," the electric current furnishing the last few hundred degrees of necessary temperature, while the combustion of gas furnishes the entire lower range.

As a matter of fact this same combination of gas and electric heating has recently been introduced on a fairly large scale with great success in an entirely different chemical industry in this country. On the other hand, it is certain that for the reduction of iron from ore the electric furnace can successfully compete with the blast furnace only in special localities under conditions which are especially favorable to the electric process and unfavorable to the blast furnace. These conditions are essentially cheap electric power and scarcity of coke. Under the conditions existing in Canada it is quite possible that localities may be found where these requirements are fulfilled.

* * *

Several points brought out in the Soo experiments are noteworthy, since they prove certain distinct advantages of the electric furnace process which should carefully be taken into consideration. First, there is no difficulty in using in the electric furnace ores of comparatively high sulphur content for the production of pig iron containing only a few thousandths of 1 per cent of sulphur. Further, charcoal and peat coke may be substituted for coke as the reducing agent. Finally, there is a possibility of utilizing to the utmost the calorific value of the fuel in a way impossible in the blast furnace. This possibility is indicated in the furnace construction of Héroult, described and illustrated on page 152 of our last issue. The fundamental idea is this: The gases leaving the blast furnace contain a considerable amount of carbon monoxide, and are now utilized outside of the blast furnace, most effectively in gas engines. The reason why carbon monoxide is necessarily contained in the blast furnace gases is to be found in the reaction between carbon dioxide and carbon, yielding carbon monoxide. As long as the fuel is mixed with the ore there is, therefore, no possibility whatever to avoid the formation of the monoxide, since the rising gases get into contact with elementary carbon. This is the reason why Héroult does not feed the reducing carbon together with the ore into the furnace, but feeds it through a hollow central electrode, so that it is introduced into the furnace at the base of the charge of ore which is not mixed with carbon. A mixture of carbon monoxide and carbon dioxide is then formed at the base of the furnace, and as this mixture of gases passes upward through the charge there is a reduction of iron ore with a simultaneous conversion of part of the carbon monoxide into dioxide. But the reducing power of the gas mixture diminishes, the higher the gases rise, and it would be impossible to change all monoxide into dioxide by this reaction alone. At a certain height, Héroult, therefore, blows air into the furnace, and thereby burns the remaining carbon monoxide to dioxide. The higher the point where the air is blown in, the greater the quantity of monoxide consumed for ore reduction, and, therefore, the smaller the heat produced by burning the monoxide. On the other hand, the lower the point of air blast, the smaller the ore reduction and the greater the heat developed. It is evident that in this ingenious way Dr. Héroult is enabled to get the full calorific value out of the fuel in the furnace itself. From a chemical and metallurgical point of view the method is undoubtedly sound, although it may be necessary to overcome some mechanical difficulties in practice. In the main experiments at Sault Ste. Marie the air blast was not used, although two short runs were made with air blast. It is unnecesary to emphasize here the entirely different character of the air blast in the Héroult electric furnace and in the blast furnace. The subject is extremely interesting, and further developments will be carefully watched by all progressive metallurgists. But, as we said before, the big commercial developments will come for the electric furnace in the near future in this country not in the pig iron field but in the steel industry. The great success of the plucky enterprise of the Halcomb Steel Co. is bound to act as an encouragement to others.

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The Internal Energy of Chemical Elements.

In thermochemical data we have always to do with the difference between two chemical energies. Thus the formation heat of HCl is the difference between the chemical energy of HCl and the chemical energies of H and Cl. For all practical purposes it is permissible to assume the chemical energies of the elements, in this case of H and Cl, to be zero. This is, of course, arbitrary, but is just as permissible as the assumption of any zero point in any other scale, and no error can arise therefrom as long as the elements themselves remain intact. In other words, as long as we do not know the absolute value of the internal energy of an atom, we may arbitrarily assume it to be zero. But the situation is different when we consider an atom to be a system of electrons, and take into account the possibility of its disintegration, as evidenced by the phenomena of radio-activity, if, according to Rutherford and Soddy's theory, radio-activity is a transmutation of an element into another element. The phenomena of radio-activity must, therefore, enable us to get at the value of the internal energy stored in chemical elements. We know now that these values are enormous, of an entirely different order of magnitude than the ordinary values of thermochemistry.

* * *

The practical impossibility of changing one element into another one thus appears in a new light. The change is practically impossible, because it would involve enormous quantities of energy, and we have not the necessary forces at our disposal. The old alchemist wanted to change cheap elements into more expensive ones; he simply looked at the matter. We know now that if such transmutation of elements was possible and spontaneous, it would pay on account of the energy evolved. Some figures from a recent paper of Soddy are suggestive. Let us assume that the element uranium, for example, which disintegrates to the extent of a thousand-millionth part annually, could be forced to disintegrate completely in the course of a year. From 1 gram of the element more than 1,000,000,000 calories could be evolved, and if it was possible to change this energy, not into heat but directly into electrical energy, this would be equivalent to more than 1,000 kw-hours, and would suffice to keep a 32-c. p. lamp burning continuously throughout the year. By the expenditure of about 1 ton yearly of uranium, costing less than $5,000, more energy would be derived than is supplied by all the electric supply stations of London put together. But the trouble is that with the forces at our disposal we cannot control the rate of disintegration of radio-active elements.

Ithaca Meeting of the American Electrochemical Society.

The programme of the ninth general meeting of the American Electrochemical Society, to be held at Ithaca, N. Y., from May 1 to 3, is as follows:

Headquarters for registry and information will be at the Chemical Laboratory of Cornell University, room 33, where members and guests will please register and receive their badges as soon after their arrival as possible.

All meetings will be held in the Chemical Laboratory, Morse Hall, which is at the northwest corner of the University Campus.

The mornings will be devoted to the reading and discussion of papers, and the afternoons and evenings to visits and social meetings.

TUESDAY.

On Tuesday morning, May 1, at 9.00 A. M., a meeting of the Board of Directors will be held at the house of Prof. W. D. Bancroft, 7 East Avenue.

The first session will take place on Tuesday morning, May 1, 10.00 A. M., in the Chemical Laboratory. After an address of welcome by Dr. Schurman, president of Cornell University, the annual business meeting of the Society will be held, at which the reports of Officers and Directors will be presented. The following papers will then be read and discussed:

W. H. Walker.—"An Instructive Laboratory Experiment in Applied Electrochemistry."

Lamar Lyndon.—"Electrolyte Density in Storage Cells."

C. L. Collens, 2d.—"Some Principles of the Resistor Furnace Design."

A. Van Winkle.—"Electrogalvanizing."

H. M. Goodwin and R. D. Mailey.—"Physical Properties of Fused Magnesium Oxide."

F. F. Colcord.—"Impurities in Electrolytic Copper."

G. I. Kemmerer.—"Cathodic Disintegration of Carbon in a Fused Chloride Electrolyte."

F. K. Cameron.—"The Movement of Suspended Clay and Other Similar Particles Under the Influence of the Current."

O. P. Watts.—"New Silicide of Molybdenum."

O. W. Brown.—"The Reduction of Metal Sulphides."

G. R. White.—"Laboratory Resistance Furnaces."

Tuesday afternoon will be devoted to an inspection of chemical laboratory, dynamo laboratory, engineering laboratories and shops.

On Tuesday evening, 8.30 P. M., the retiring president, Prof. W. D. Bancroft, will present in the Chemical Laboratory his address on "The Electrochemistry of Chemistry." This is to be followed by a complimentary smoker at the Town and Gown Club, Stewart Avenue, at 9.45 P. M.

WEDNESDAY.

On Wednesday morning a session will be held at 9.30 in the Chemical Laboratory, where the following papers will be read and discussed:

E. A. Sperry.—"Electrochemical Processes as Station Load Equalizers."

W. C. Arsem.—"The Electric Vacuum Furnace."

B. E. Curry.—"Electrolytic Corrosion of Copper Tin Alloys."

Maximilian Toch.—"Electrolytic Corrosion of Structural Steel."

C. F. Burgess and S. G. Engle.—"Observations on the Corrosion of Iron by Acids."

Isaac Adams.—"The Development of the Nickelplating Industry."

R. C. Snowden.—"Electrolytic Precipitation of Lead from Acetate Solutions."

C. F. Burgess and O. P. Watts.—"A Microscopic Study of Electrodeposits."

G. A. Hulett.—"The Cadmium Standard Cell."

M. LeBlanc.—"Electrolytic Chromium."

B. E. Curry.—"Electrodeposition of Bronze."

G. R. White.—"Ferromanganese Anodes in Caustic Potash."

On Wednesday afternoon a visit will be paid to the filtration plant, the hydraulic laboratory, the power plant in the gorge and the Ithaca Gun Co. The excursion will start at 2.30 from the Chemical Laboratory.

On the afternoon of Wednesday a subscription dinner will be served at 7.00 P. M. in the Dutch Kitchen.

THURSDAY.

The third session will be held on Thursday, at 9.30 A. M., in the chemical laboratory, when the following papers will be read and discussed:

M. deK. Thompson, Jr.—"The Free Energy of Some Halogen and Oxygen Compounds, Computed from the Results of Potential Measurements."

R. von Foregger and G. F. Brindley.—"Report on Experiments with Fused Sodium Peroxide for the Regeneration of Air in Submarines."

W. D. Bancroft.—"Lecture-room Switchboard."

G. R. White.—"Alternating-Current Electrolysis with Cadmium Electrodes."

J. W. Richards.—"The Electrolysis of Caustic Soda."

E. A. Ashcroft.—"A New Sodium Process."

H. M. Goodwin and H. A. Wentworth.—"Electrometric Measurements with Reference to the Dissociation of Fused Salts."

L. Kahlenberg and A. S. McDaniel.—"Differences of Potential Between Manganese and Lead Peroxides and Various Aqueous and Non-Aqueous Solutions."

Henry S. Carhart and F. W. Willard.—"A New Electrolyte for the Silver Coulometer."

E. S. Shepherd.—"Errors in Pyrometry."

John Nelson.—"The Effect of Oxides on the Adhesiveness of Electrodeposited Metals."

On Thursday afternoon an excursion will be made to the Remington Salt Co., with a trolley ride around the Loop and the Circle. The excursion will start at 2.30 from the Ithaca Hotel. The hotel headquarters will be at the Ithaca Hotel, State Street.

Ithaca may be reached either by the Lehigh Valley or by the Lackawanna Railroads.

The Regeneration of Air by Means of Fused Sodium Peroxide.

At a meeting of the Western New York Section of the American Chemical Society, held at Niagara Falls, April 3, Messrs. Brindley, FitzGerald and Bennie gave a demonstration of the regeneration of air by means of fused sodium peroxide. Mr. FitzGerald said in part as follows:

"It was originally intended that Dr. R. von Foregger should read a paper to-night; but as he, unfortunately, cannot be present he has asked me to take his place.

"When sodium peroxide is caused to react with water under suitable conditions it is decomposed, forming a solution of caustic soda and generating oxygen gas. Unfortunately, sodium peroxide in its ordinary form is not a very safe substance to handle, but under the name of 'oxone' a new and perfectly safe form of sodium peroxide can now be obtained. This substance is manufactured by the Niagara Electrochemical Co., the difficult problem of its production having been worked out by Mr. G. F. Brindley.

"In view of the action of water on 'oxone' it seemed reasonable to suppose that where moist air passed over it oxygen would be generated, and the caustic soda simultaneously produced would absorb any carbon dioxide contained in the air.

"Working on this hypothesis, Messrs. G. F. Brindley and

[1] Feerstenhag & Philipp, U. S. patent No. 788,256.

R. von Foregger carried out a number of experiments, and to-night we have one of these in progress before you. The box you see here is 150 x 70 x 70 cms. inside. It is tin-lined, and the lid provided with water-lute, which makes it perfectly air tight. The interior of the box is divided into two compartments by means of a wire screen. In one compartment is a wire cage, containing 'oxone' and a motor-driven fan, which causes the air to circulate over the 'oxone.' In the other compartment we have five rabbits. The lid of the box is provided with a tube passing to the interior, so that Mr. Brindley can draw off samples of the gas in the box and analyze them.

"We shall now stop the fan in the box, the experiment having been running, as you may see by the curve, for 1 hour and 50 minutes, and presently we shall observe a rapid increase of CO_2 and decrease of O. You will observe at the first part of the experiment how the oxygen percentage dropped and carbon dioxide increased before the fan was started.

"We believe that this experiment demonstrates conclusively that it is possible to regenerate the air in closed spaces by means of 'oxone,' and, moreover, the amount required is not large. Messrs. Brindley and von Foregger have found by experiment that one man requires very nearly 1-3 pound of 'oxone' per hour to supply him with oxygen for respiration and to absorb the carbon dioxide he eliminates. One other point in connection with the use of 'oxone' for the regeneration of air should be noted. You are aware that in the process of respiration various toxic substances are thrown off from the lungs, and that even where the oxygen and carbon dioxide are kept at a normal value in the air, the gradual accumulation of these toxic substances might have serious effects. The experiments of Messrs. Brindley and von Foregger indicate, however, that one of the results of passing the air over the 'oxone' is to cause the elimination of these toxic substances."

A short discussion followed Mr. FitzGerald's paper.

Electrometallurgy of Iron and Steel Before the Faraday Society.

The electrometallurgy of iron and steel was again the subject of discussion at the meeting of the Faraday Society on April 10. Three papers relating to this subject were presented, two being of French, the third of Italian origin.

The paper by Ernesto Stassano is a brief note on his rotating electric steel furnace in the artillery construction works of the Italian War Office, in Turin, Italy. The construction was described and illustrated in our Vol. III., p. 391. Three such furnaces, of 1,000, 200 and 100 hp. respectively, are being installed. They will be used for refining pig iron and melting scrap. But it is also intended to demonstrate the commercial possibility of making iron and steel in one operation direct from the ore.

The paper by Gustave Gin first states very briefly that the author's "canal-type furnace" (our Vol. II., p. 20) is now installed at the Plettenberg Works, Westphalia. These works were founded by M. Bruninghaus, who is associated with the Deutsche Elektrische Stahlwerke. Four photographs of the works are reproduced.

The author then introduces a description of his "furnace with canals and chambers," by the following words: "In the furnaces in which the canal is bent upon itself, the narrow cross-section of the canal is not convenient for the introduction of ore or scrap. Moreover, heavy currents at low potentials give rise to induction and other effects. To avoid these drawbacks I cause the electrothermic heating and refining to take place separately, the refining taking place in chambers which communicate with each other and with the leads by the canals for heating." However, "a difficulty which at present prevents the practical application of this furnace is finding a material for the bases sufficiently solid to prevent short circuits between the branches of the canals and the chambers."

The author then takes up his "combination furnace," which consists of three chambers: First, a melting and oxidizing crucible; second, a deoxidizing and recarburetting compartment; third, a final mixing chamber. This furnace is essentially the same as the type described in our Vol. III., p. 372. The author makes, finally, some general remarks on induction furnaces, the original invention of this type being ascribed to Ferranti (English patent of 1885).

The author of the third paper, which is the most interesting of the three, is Ch. A. Keller, engineer to the Keller Leleux Electrothermic Co., of Livet, France. He states that J. Holtzer & Co., of Unieux (Loire), have installed an electric steel plant which has just been completed. A steam-driven, 1,500-hp Westinghouse alternator gives a current of 20,000 amps., to be used in one Keller furnace. (According to the Canadian Commission Report, our Vol. II., p. 482, this is in principle identical with the Heroult furnace, which was repeatedly described in our columns, and varies only in details of construction.) The current is only for the finishing, refining reaction, since the steel produced in an open-hearth furnace is "run into the electric furnace immediately after the oxidizing melt, where the remaining operations would be completed by the current; 800 kilograms of metal could be operated on at one time, the operation being repeated three or four times in 24 hours."

The furnace at Unieux weighs about 50,000 kg.; it rests on a steel cradle and can be tilted. All the mechanical and electrical controls are obtained by auxiliary hydraulic motors. The steel produced by this furnace is intended for cannon and projectiles. No further metallurgical or mechanical information is given.

The author finally gives a preliminary account on experiments made by him at Livet on pig iron reduction in the electric furnace. "Apart from the elimination of sulphur, these experiments have shown that the Keller furnace (our Vol. II., pp. 483, 485) has an advantage over the blast furnace in the easier variation of the silicon and carbon content. This particular property of electric reduction would prove of great utility in the production of soft castings specially for mill work." The author has set up at Livet an electric furnace of 2,000 hp., which will be capable of producing about 20 tons of castings in 24 hours. It will be used specially for the industrial working of the process. The current strength will be 25,000 amps.

Recent Practice in the Treatment of Tin-Tungsten-Copper Ores.

At a meeting at the end of last year the Institution of Mining and Metallurgy had before them an interesting paper by Mr. F. Dietzch, on the "Treatment of Tin-Tungsten Copper Ores at the Clitters United Mines."

The paper began with the explanation that large quantities of tungsten and tin had been picked from the waste heaps of the old workings of this abandoned mine, and it was therefore decided to reopen it with labor saving machinery and the best appliances for separating the respective ores. It was initially proposed to work the mill as two separate systems, the one for the treatment of tin ores containing little tungsten, the other for treating tin ores containing large quantities of tungsten, but owing to the successful working of the magnetic separators, the two systems have been advantageously merged into one.

The arrangement of the mill is now as follows: The greater part of the ore arrives at the back of the mill and passes over two grizzlies, whose bars are 2 inches apart, the fines falling directly into the large storage hopper, while the coarse pieces fall on the platform, and from thence are fed into a 15 in. by

10-in. Blake rock breaker and delivered also into the same storage hopper. The ore is then passed by four Challenge feeders into the boxes of four batteries, each of five stamps. The weight of each head is 800 pounds, and they are dropped ninety-five times per minute.

The pulp is passed through No. 25-mesh gunmetal woven-wire screening.

A smaller portion of the ore is dumped into a storage hopper, and is, after the elimination of its fines by the automatic shaking screen, fed into a small 12-inch by 8-inch rock breaker. The ore then passes through a set of friction-driven rolls of 28-inch diameter, 12-inch face, and hoisted by the elevator to the vibro screen, from whence the fines are sent to a No. 6 ball mill, which crushes them through No. 30-mesh screening.

The coarse material collecting on top of the vibro screen returns to the rolls and is recrushed.

Prior to the introduction of magnetic separation, the second or smaller system was used for the treatment of tungsten-tin ores, and the place of the No. 6 ball mill was occupied by jigs, which received their pulp from the vibro screen after passing through a series of trommels. Following the extraction of the tungsten on the jigs, the tailings were recrushed in a separate five-stamp battery of the same construction as the other four above described. This separate battery is now supplied with mixed ores from the small rock breaker without having been crushed in the rolls.

The amalgamation of the two systems has permitted the plant to be better utilized, rendering the jigs superfluous in the general scheme. They will now be in reserve when re-elected, to be used upon such high-grade copper ores as contain the mineral in a sufficiently coarse condition to permit of their treatment by jigging. The rolls have proved of much use as an auxiliary to the No. 6 Krupp ball mill, as they increase the capacity of that mill considerably. The ball mill was experimentally erected to act as an auxiliary to the stamps, when it was found that although they could deal with the softer ores from Clitters, they were not of sufficient capacity when the harder ores of the Hingston Mine came to be included in the general treatment.

The combined pulp from the twenty-five stamps and the ball mill pass through three spitzlutten; two of these have three compartments of varying depths, and one has only two compartments. The product of each spitzlutten passes to a Buss swinging table, while the overflow of the three spitzlutten (fines) goes to a ten-compartment condensing and classifying spitzkasten, 50 feet long by 6 feet wide on top, and 5 feet 6 inches deep. Three pairs of these spitzkasten compartments supply their pulp through launders to three two-compartment distributing boxes, which in turn feed three double Lührig vanners.

The remaining four spitzkasten compartments supply a double vanner through a single two-compartment distributing box.

The overflow at the end of the series of spitzkasten, together with the tailings from the Buss tables, the overflow of the distributing boxes and the tailings of the Lührig vanners outside of the mill, pass into a second eight-compartment condensing spitzkasten, 40 feet long, 6 feet wide on top, and 5 feet 6 inches deep.

In future it is intended to collect the middlings from the tables and to recrush them in a wet ball mill, that is at present being used to recrush a stock of high-grade middlings, which had accumulated in the past. After recrushing, the table middlings, together with the overflow of the spitzlutten, will be treated on the Lührig vanners. At the same time, by using a No. 20 mesh screen, the first crushing of the ore will be made much coarser, and so decrease the production of slimes and increase the tonnage treated. It may here be remarked that the ball mill has been found to slime very much less than the stamp mill, and that even after recrushing in a ball mill, through an 80-mesh, the production of slimes was found to be comparatively small.

It follows that while the tables finally eliminated a very large proportion of the light waste in one operation, and only allow waste particles carrying some minerals to be caught as middlings, the quantity of these latter is by no means large, and can be easily dealt with in a small ball mill, so that as a net result not only is the advantage of not sliming the ore attained in the first instance, but the capacity of the mill is greatly increased by this form of graduated crushing.

On account of the limitations of space in our present issue, the conclusion of the abstract of this paper, as well as the account of the animated discussion which followed, must be reserved for our next issue.

CORRESPONDENCE.

Construction of Lampboard.

To the Editor of Electrochemical and Metallurgical Industry:

Sir:—A lampboard was recently devised and made for me by Mr. Angel, which is so simple and inexpensive, and at the same time so effective, that I think a description of it may be of interest to some of your readers.

The arrangement consists of a rectangular wooden frame, about 70 cm. long and $7\frac{1}{2}$ cm. wide, supported at either end about 15 cm. above the work bench.

On the two longer sides of this frame are stretched pieces of copper wire (*BB*) about 3 mm. in diameter, and connected to the binding posts (*AA*).

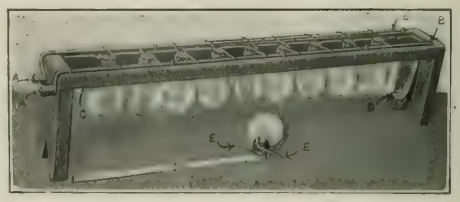


Fig. 1.—Construction of Lampboard.

The lamps to be used are supplied with two small pieces of hard copper wire (*EE*) 5 cm. long, soldered to their terminals, as shown in Fig. 1. By means of these wires the lamps are hung across the leads *BB*.

A second and smaller rectangular frame, pivotted at *C* and

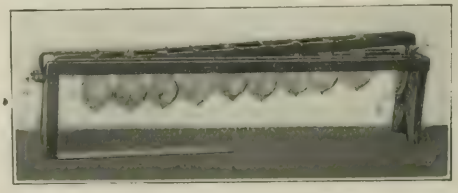


Fig. 2.—Only one Lamp in Circuit.

supplied with the small rack *D*, allows of any number of the lamps being thrown in or out of action. By raising the free end of this inner frame all the lamps may be successively cut out.

Fig. 2 shows the lampboard with all but the lamp nearest to *C* out of action.

Richard Seligman.

London, England.

Electric Lead Smelting.

By Anson G. Betts.

There are possibilities for electric lead smelting, especially for roasted lead concentrates, using electricity as a mere heating agent in place of coke in the ordinary lead reduction, as the greater cost for power than that for fuel in the blast furnace process would be offset by a lower loss of lead. Lead is a volatile metal, and lead sulphide is also quite volatile, so any process, the blast furnace process for example, in which large volumes of gas or air come in contact with heated lead, will suffer from loss of lead in fume. The electric furnace, by reason of the absence of large currents of gases, is bound to give a higher saving of lead.

But this paper will be confined to what might be termed *electrolytic* lead smelting. Not a great deal has been attempted in electrolytic lead smelting, except by Swinburne,[1] and possibly Swinburne and Ashcroft's well-known chlorine smelting process might be included under this head.

Dr. Wm. Valentine and I worked out the process which I shall describe, and it has since come to my knowledge that at least two other investigators[2] have made, or are making experiments similar to ours. In view of the fact that the process may be found to be very valuable under certain conditions, I think a presentation of our results and some calculations will be of interest.

It would not be a good plan to try to electrolyze lead sulphide directly for lead and sulphur, as the melting point is high, and lead sulphide is probably not an electrolyte, although a good conductor.

Electrolysis of Lead Sulphide Dissolved in Fused Lead Chloride.

The idea occurred to dissolve the lead sulphide in a bath, to form an easily melting electrolyte. A number of experiments were made, and for a time it appeared that we had a good process in the following:

A bath of lead was put in an iron or other suitable pot and melted lead chloride placed on top. A block of carbon or graphite dipped in the fused chloride as anode, while the lead was connected as cathode. Galena was added to the bath, and it dissolved rapidly, in large quantities, at temperatures well below a red heat. On passing a current, lead was reduced quantitatively, and sulphur came off from the anode, as vapor, if the bath reached the boiling point of sulphur, 440° C. The conductivity of the bath was extremely great. The voltage of decomposition is about .4 volt, but about 1 to 1.25 volts would be necessary to operate at a good rate of speed. The lead reduced was soft, free from iron and sulphur, the apparatus was durable, and with good galena, the results were excellent for a time. But eventually the cell became clogged up, on every occasion. It was evident that we had a good and neat process, if some way could be devised to make it continuous without too great expense. The power required to decompose a ton of lead sulphide would be about 14-hp. days, or a cost of $0.77 or $1.54 with power at $20 and $40 per horse-power per year respectively; 250 pounds of sulphur, or thereabout, would be saved, which would cover the power cost. The labor item, which would be for feeding ore to the pots and tapping out the lead and handling condensed sulphur, should not be over $0.75 to $1.00 per ton.

Then in cases where power was quite expensive, there would be the cost for keeping the pots hot with a fire underneath, which could be done most economically by producer gas, considering the resulting saving in labor and heat, but this would not cost to exceed 50 cents per ton. Allowing 50 cents for repairs, we would have a total cost of $3 to $4 per ton of galena, *saving practically all the sulphur*, or 250 pounds per ton, and *no loss in lead*. These last two items would mean a saving over ordinary smelting practice of probably as much as $5 to $6 per ton. So that it was evident that the process had great possibilities.

To overcome the clogging of the pots we tried various plans, such as dissolving out lime with hydrochloric acid in a preliminary treatment, and trying to fish the accumulated silica, etc., out of the pots. These methods were not successful, and it was found that *iron* was the great nuisance as it is in all such processes for reducing lead. The iron would reduce with the lead and make a thick scum. The hoped-for reaction $Fe + Pb Cl_2 = Fe Cl_2 + Pb$ does not occur. It appears to cause serious loss of efficiency, too.

There seemed to be no way to get rid of the iron probably contained as pyrites, and taken with the expense caused by lime, silica, etc., the process had to be abandoned. It was found afterward that Swinburne had already described the identical process in a British patent.

Fig. 1 shows the type of apparatus that should be used. The pots would be made of cast iron, lined on the sides with fire-brick, and could take a current of, say, 30,000 to 50,000 amps., with a voltage of 1.25, or thereabouts, corresponding with a daily production per pot of 3 to 5 tons of lead.

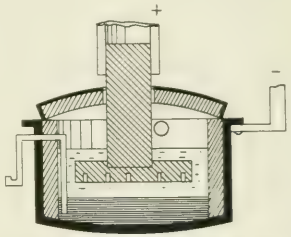


Fig. 1.—Apparatus for Electrolysis.

Such currents could be produced probably more easily and cheaply than the smaller currents that have been used heretofore. With the commutating generator the cost of the machine has been greater, the larger the current (for a given power output). With the unipolar generator, such as developed by Mr. Noeggerath,[3] the cost of the machine for constant output should be actually less, the greater the volume of current. The machine described in the *Electrical World*, if built for a current of 7,200 amps. at 42 volts, instead of 600 amps. at 500 volts, would have been cheaper and simpler. It seems probable that in electrolytic furnace processes we will soon use much larger units for 50,000 or 100,000 amps., perhaps, produced by unipolar generators, and this will mean larger furnaces with consequent less heat loss, and less labor, hence higher economy.

Zinc chloride was also tried as electrolyte, but lead sulphide does not dissolve in it apparently. It was thought that zincy lead ores could be treated with this electrolyte, but zinc chloride suffers from many objections. It is relatively a much poorer conductor than lead chloride, too volatile and expensive to make.

We then worked out our process from the experience gained with the above process and other similar work, and made provision for the elimination of gangue and iron, which is necessary for continuous operation with commercial ores. The process consists of two stages: first, fusion of the ore into the matte, high in lead sulphide, to get rid of lime, alumina, silica, etc.; and, second, electrolytic reduction of the fused matte under such conditions that the iron remains fused as sulphide and does not contaminate the reduced lead or clog the bath. This condition is obtained in melted sodium chloride, is used as electrolyte, with lead sulphide resting on a cathode at the bottom of the cell.

It is not difficult to produce with lead ore a furnace charge which will melt easily into a matte and slag. Some of the ores that we worked on gave the following on analysis:

[1]Engl. patent 10,329A, of 1897.
[2]U. S. patent of C. P. Townsend, 815,887, March 14, 1906. U. S. patent of Valentine and Betts, 816,764, April 3, 1906.
[3]Electrical World and Engineer, Feb. 4, 1905, page 230. Transactions American Institute Electrical Engineers, January, 1905.

No.	Ore.	Pb.	S.	Fe.	Insol.	Zn.	CaO.	Cu	MgO
1	Payne, B. C....	61.6	13.85	3.6	8.7	5.7	1.0	..	..
2	Missouri.....	72.5	13.3	1.45	1.25	..	4.65	.2	1.5
3	Bunker Hill and Sullivan, Idaho	50.75	12.4	13.75	6.0	..	1.35	..	..
4	St. Eugene, B.C..	52.38	11.86	12.9	5.5	3.0	.8	..	..

No. 1 gave a good melt when fluxed with 15 per cent of limonite (carrying 51 per cent iron and 13.55 per cent insol.) and 2.5 per cent limestone. The products were slag and matte of the following composition:

	Pb %	Fe. %	Insol %	Zn %	FeO %	CaO %	Wt. in Terms of Ore Used %
Matte.....	70.8	11.5	.8	5.1	..	..	78.7
Slag	12.12	..	44.7	6.7	23.4	3.3	51.7

In a good many of the melts metallic lead was produced in quite considerable quantity, which explains what would otherwise appear to be a heavy loss of lead. In this particular melt about 97 per cent of the lead is accounted for in the slag and matte.

No. 3 gave a good melt with 5 per cent of limestone. Percentage of lead which went into the slag 5. Matte recovered 68 per cent of weight of ore taken. Analysis of slag:

Pb	SiO_2	Feo	Ca	Percentage of Weight of Ore Taken.
5 pec ct.	46.20	22.6	8.33	22.75

No. 3 also melted without fluxes, but not so well. Matte recovered but not analyzed, 68 per cent of weight of ore taken. Analysis of slag:

Pb	FeO	SiO_2	CaO	Percentage of Weight of Ore Taken
9. 2 per cent.	24.2 per cent.	43.75	1.55	19.

No. 4 also melted without fluxes, giving 20 per cent of its weight of slag and 67.5 per cent of its weight of matte. Analysis of slag:

Pb.	Insol	FeO	CaO
13.1 per cent.	33.2	27.23	1.75

The mixture containing

Pb	Zn	Insol	S	Fe	CaO	H_2O	Cu
68.9	1.7	3.65	13.2	4.5	2.75	5	tr.

melted well with limonite (51 per cent Fe, 13.0 per cent Insol.) 12 per cent. limestone 2 per cent, and coal .6 per cent. Percentage yield of matte 79.4 per cent. of slag 9.1 per cent Analyses:

	Pb	Fe	FeO	Insol	CaO	S	Zn
Matte.....	67.73	10.6	...	tr.	tr.	19.83?	...
Slag	11.77	...	20.37	30.8	18.5	2.1	1.6

The coal was omitted in some melts, when no metallic lead was produced; 10 per cent of limonite, no lime used; yield of matte 86.4 per cent, of slag 16.2 per cent. Fusion not very successful Analyses:

	Pb	Fe	FeO	S	Insol	CaO	Zn
Matte.....	75.44	6.0	.	14.28	tr.	tr.	1.0
Slag	12.84	...	27.22	1.04	32.0	13.6	1.2

A great many such melts were made in large graphite pots, and we also tried melting in a small reverberatory furnace with gasoline burners The latter, however, was not successful, on account of low temperature in the furnace

For commercial work the melting could be done in blast furnaces, or better, in reverberatories, or still better, in an electric furnace With an electric furnace there would be practically no loss of lead (the lead in the slag could be easily recovered by blast furnace smelting), with a cost for power which can be calculated with a fair degree of accuracy. Taking the above melt with 12 per cent limonite, 2 per cent calcite, and 0.6 per cent coal, as an example, suppose the charge contains

	Per Cent.
Zinc as sulphide	1.5
Lead as sulphide	59.5
Sulphur	11.1
Iron as pyrite	8
Iron as siderite (in the lead ore)	3.3
Oxygen for same	.8
CO_2 for same	2.4
Iron as limonite	5.2
Oxygen for same	2.2
H_2O	2.0
SiO_2	4.4
Carbon	.5
CaO as limestone	3.4
CO_2 for same	2.6
	100.0

The calculation can best be made from the thermochemical values at the ordinary temperature, and adding to this the heat in the liquid products at 1,000° and the heat in the gases. The calculation was also made in an entirely different manner by taking the specific heats of the cold products, and adding the thermochemical values for the reactions that take place and the heat required for fusion. The result was not much different from the value obtained by the first method. The data used are from Dr. J. W. Richards' very useful "Metallurgical Calculations," published in the Electrochemical and Metallurgical Industry. Dr. Richards has very kindly supplied me with other data for use in the calculations. The heat in melted PbS at 1,015°, he has determined as 104 calories. The decomposition point of FeS_2 and S he has also determined at 565°.

As nearly as it is possible to tell, from the work that has been done so far, the reactions, and heat values for them, for this particular charge, are:

		Lb. Cals.
$CaCO_3$ (120 lbs.) + SiO_2	$= CaSiO_3 + CO_2$	— 32,400
$FeCO_3$ (137 lbs.)	$= FeO + CO_2$	— 42,300
FeS_2 (34 lbs.)	$= FeS + S$	— 560
2FeO (20 lbs.) + 2S + C	$= 2FeS + CO_2$	— 2,530
2FeO (40 lbs.) + 2PbS + C	$= 2FeS + 2Pb + CO_2$	— 7,300
FeO (43.5 lbs.) + SiO_2	$= FeOSiO_2$	— 11,400
$2Fe_2O_3$ (22 lbs.) + C	$= 4FeO + CO_2$	— 2,140
$9Fe_3O_4$ (83 lbs.) + C	$= 4Fe_3O_4 + CO_2$	— 710
CO_2 (20 lbs.) + C	$= 2CO$	— 20,995
Total for chemical reactions		— 96,115

Heat in Melted Matte and Slag. Calculation of the charge indicates that Fe_3O_4 was present in the matte, as it sometimes is, particularly in mattes high in iron. It hardly seems probable that there is magnetic oxide in these mattes, from the analyses given, and from other considerations. It is more probable that there is a slight error in the analyses.

Heat in melted matte at 1,000°:

		Lbs. Cals.
1,588 lbs.	19% FeS, ZnS, 200 5% Fe_3O_4, 400 76% PbS, 104	1,588 × 139..... 220,732

Heat in molten slag at 100° C.:

182 lbs. × 300 calories.....	54,600

Heat in gases. Assume average temperature of evolution 900°.

209 lbs. CO, 209 × .289 × 900	54,301
25 lbs. CO, 25 × .200 × 900	5,850
40 lbs. H_2O at 100, 40 × 646	24,040
	360,183
For chemical reactions	96,115
Total	456,298

If the matte contained no Fe_3O_4 but FeS instead, the extra heat required for formation of FeS from pyrite, iron oxide and carbon is somewhat less than the difference in the heat in

the melted materials at 1,000°, so that there is no material thermal difference either way.

It will be seen that some reduction in the heat bill would come from the substitution of iron oxide for part of the lime, and especially so if the iron oxide was free from silica, as the amount of slag necessary to form would be less. In this case the limonite, added as flux, carried some 25 pounds of silica, and meant the production of about 75 pounds extra slag, increasing the heat consumption altogether by about 35,000 pounds calories, as follows:

	Lbs. Cals.
Decomposition 27 lbs. $CaCO_3$ by silica	7,200
Heat in 12 lbs. CO_2 at 1,000°	3,720
Heat in 75 lbs. slag at 1,000°	22,500
Decomposition $FeCO_3$, unimportant	....
	33,420

As will be seen later the iron of the matte becomes in the process eventually available as flux without silica, and it is, therefore, proper to subtract from the above figure 35,000 calories, leaving 421,200 pounds calories.

Electric furnace efficiencies on a large scale may be brought to 75 per cent, especially when the temperature is relatively low, as in this case.

Therefore, we can estimate that 561,700 pounds calories are necessary to melt 1 ton of charge of the composition given, or 639,000 per ton of ore. Of course, this would be considerably less with some ores and charges, and greater with others, but an idea is at least obtained as to what is required.

One electric horse-power in 24 hours = 34,000 pounds calories, or 450-hp. hours per ton of ore. For 100 tons of ore per day about 1,880 hp. would be necessary. In some districts, with power at $20 per year, for example, Niagara Falls, the power bill would be $1.04 per ton of ore. With power at $40 per year, for which it could be probably be developed in lead districts like Idaho and British Columbia, the power bill would be $2.08 per ton.

It is easy to see how important economies of electric energy could be made for the above process. If carbon had been omitted from the charge, and a slag higher in lead made, the melting temperature would have been reduced, and less iron and lime needed for the charge. Allowing even considerable quantities of lead to go into the slag, would not be a disadvantage, as the proportion of slag produced is small, and there will always be enough lead in the slag to make it worth smelting in the blast furnace. The proportion of iron, and possibly zinc, in the matte is less, too, when carbon is not added to the charge, and this is a real advantage.

In case the ore is quite silicious, it might be better to flux the silica with lead, and resmelt the slag in a blast furnace, rather than add outside fluxes and increase the power consumption. If the ore is practically pure galena, like some Mississippi Valley ores, the amount of electricity required will not be much more than half the amount required in the example given.

For reverberatory melting, instead of electric furnace melting, and assuming a loss by volatilization of as low as 5 per cent of the lead, or 70 pounds per ton (valed at, say, 3.5 cents per pound = $2.45 per ton), and remembering that the coal for the reverberatory furnace and its handling costs something, it is apparent that with reasonably cheap power the electric furnace has a decided advantage.

A combination process is also possible. The melting furnace could be built in an analogous manner to a lead hand roaster with fusion hearth. The ore could be warmed up to the oxidizing point by direct heat from a fire, and then pushed into the electric hearth at the end and fused.

Probably three-fourths of the direct heating and volatilization could be done non-electrically, and this would leave to be supplied electrically perhaps half. If power was very expensive this combination of process would pay. It would be cheaper to install, as the electric and power machinery to be supplied for this last of the work would be only half as much, while the extra brick work required in the furnace would be relatively cheap.

The simplest electric furnace for this work appears to be the type of furnace, illustrated in Fig. 2. Of course, other types in which the heat supplied cannot raise any portion of the charge to an unduly high temperature may be used.

The furnace consists of a fusion hearth, on the bottom of which rests fused matte, with, perhaps, some lead underneath it. Leading out from the hearth is a resistance channel running back again into the hearth on the opposite side. An alternating current is supplied by two large electrodes.

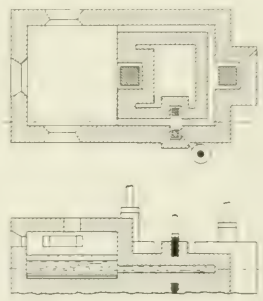


FIG. 2.—SECTIONS OF FURNACE.

To produce motion of matte (or lead) through the channel, a small cross-channel is built in with two smaller electrodes at the ends. A magnetic field is established at the intersection of the channels. When a current is passed through the cross-channel, motion is produced longitudinally in the main channel. Alternating current may be used, if an alternating current of the same frequency is also passed through the coil of the magnet.

The matte circulating through the channel becomes heated, and by establishing a continuous current the temperature on the hearth may be maintained. With this kind of furnace the consumption of electrodes in contact merely with metal or matte would be extremely small. The temperature at any one point would not be excessive, a desirable circumstance.

The labor involved in adding the ore to the electric furnace and tapping out the products ought to be a matter of perhaps 50 cents per ton. Repairs to a furnace of the type illustrated should be covered by 25 cents per ton. The cost for fluxes would vary largely, but as only 10 per cent need be used for many ores, and some of this is produced in the process, as will be seen further on, this would be an item of perhaps 25 cents per ton or less. The total cost would be for this operation, with power at $20 and $40 per year, as follows:

Labor	$0.50	$0.50
Repairs	0.25	0.25
Fluxes	0.25	0.25
Power	1.04	2.08
Smelting slag in blast furnace 10 per cent	0.40	0.40
	$2.44	$3.48

We still have to consider the electrolytic furnace operation for reducing the lead and recovering the sulphur. The theoretical voltage of decomposition of lead sulphide is .4 volts. This agrees with the value ascertained in the experiments. A somewhat higher voltage is necessary to drive the current through in the large volume desired to get a large output from the cells. External heating would save some electric power, but the more difficult construction of the pots, their larger size, and the greater amount of repairs, coupled with the fact that it would cost something to supply the heat by coal, puts such a procedure out of the question, unless power should be unduly costly.

The matte will be added at a higher temperature than the products are withdrawn, probably about 150° hotter, and this

will supply some heat to the electrolyzer. With larger furnaces and good heat insulation the voltage per vat should not exceed 2.4, and the current efficiency should be 90 per cent. Not all the lead may be reduced in the pot, as some will be tapped with the practically exhausted matte. Assume that the reduction will be 90 per cent of the lead. This figures out to 23.5-hp. days per ton of 75 per cent matte, or 20.7-hp. days per ton of ore. The 10 per cent of the lead tapped out with the matte will be brought back again as lead sulphide, so it will be better to add one-ninth to the amount of power, to save figuring the power for this lead at another time, so that the power per ton of ore would be 23.0-hp. days.

I think it quite possible that the voltage will be less than 2.4 volts per cell, especially with large cells.

Data: Hall's aluminium process is similar in some respects, and is operated in quite large cells, of 10,000 amps. capacity. The temperature is somewhat higher than in lead reduction, and the cells are lined with carbon, which is relatively a much better heat conductor than fire-brick, so the lead reduction ought to operate with considerably less heating by the current being necessary.

In the *Engineering and Mining Journal,* March 17, 1906, are given some data on the aluminium furnaces, as follows: Voltage per cells, 5.5; voltage required for electrolysis, 2.2; voltage for heating alumina and carbon, liquefying alumina and maintaining the temperature of the bath = 3.3.

To operate the 10,000-amp. cells, 14 pounds alumina and nearly the same weight of carbon per hour have to be added in the cold state.

	Lbs. Cals.
To heat 14 lbs. alumina to 900°, 14 × 900 × .29	3,654
To melt 14 lbs. alumina at 900°, 14 × 60 (assumed)	840
To heat 14 lbs. carbon 900°, 14 × .363 × 900	4,574
	9,068

Nine thousand and sixty-eight pound calories are equivalent to 0.48 kw-hours, or the e. m. f. supplied for heating materials and melting is 0.48 volts, leaving about 2.8 volts to maintain the temperature of the bath.

For a 10,000-amp. cell, reducing lead about 120 pounds of matte are added hourly, at about 150° above the temperature at which the products are removed.

Heat added hourly by matte and available for heating:

	Lb. Cals.
With 103 lbs. PbS, 103 × .12 × 150	1,854
With 14 lbs. FeS, 84 × .21 × 150	441
	2,295
	Volts.
Equivalent to 1.2 kw-hours per hour, or	0.12
Assumed voltage of operation	2.40
	2.52
Voltage of decomposition	0.04
Available for maintaining temperature	2.12
Plus 10 per cent loss of efficiency	0.04
	2.16

This is somewhat less than is supplied in aluminium making, but there are a number of considerations in favor of the lead process, namely, greater heat insulating power of fire brick than carbon, possibility of using several times larger cells in both processes, while the current density may be higher, meaning less heat loss, because less surface. The carbons will not be destroyed in the lead process, so that it would pay to make permanent contact and save something in power there.

Regarding the use of larger currents, in *Richards' Aluminium,* it is stated[5] that the Hall process used about 9 volts per pot with a current of 1,800 amps. In the article in the *Engineering and Mining Journal,* above referred to, the voltage with 10,000 amps. is down to 5.5. There is no reason to suppose that with 30,000 amps., say, the voltage would be still lower.

In Acker's sodium process, which is similar in many respects, 6 volts per pot is stated to be all that is required when all unnecessary resistances are removed;[6] 3.5 volts are required for decomposition and .59 volts to heat and melt the salt, leaving 1.9 volts to maintain the temperature. In Acker's process a current density of 2,750 amps. per square foot is used.

TABLE I.

Process	Current	Volts per Cell	Volts Available to Maintain Temperature
Hall	10,000	5.5	2.8–2.6
Acker	8,000 to 8,200	6.0–6.5	1.9 to 2.4[4]
Lead reduction	10,000 or over	2.0	2.2 on within assumptions

The reduction of the lead from the matte is carried out as follows: The fused (or solid) matte is put into a pot, as shown in Fig. 3, containing fused common salt. An iron casting imbedded in brickwork serves as cathode and container. It is lined internally with brick. The anodes are of carbon, much better of graphite. The upper part of the carbons and the metallic connection can be protected by a sleeve of fire-clay filled with cement. At one side is a lead well, at the other side a sulphur outlet, and at the rear a tap for matte, and if desired to clean the pot for the salt electrolyte. There is no perceptible action on the graphite anodes. A mixture of salts for electrolyte is less desirable than straight NaCl, which we used almost entirely. With the mixture, one is apt to be troubled with a solid crust of one of the components of the bath on the anodes, stopping the current. This can be obviated by using a higher temperature, but then the advantage of a mixture of salts disappears.

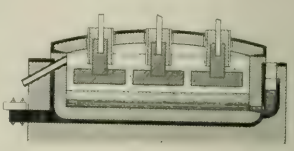


FIG. 3.—REDUCTION OF LEAD FROM MATTE.

With this kind of construction lead finds its way as far as the iron, through cracks in the brick-work, thus establishing the electrical connection to the bath of lead.

By using flues from a furnace built into the brickwork under the furnace, all loss of heat downward through the brick could be saved, and a lowering of voltage made, at the risk, however, of the pots cracking and lead running down into the flues. With this construction, if the pot leaks, no great harm is done. The greater part of the loss of heat will be laterally and upward, anyway. The pots could be heated by direct fire, and might last several months. It would be a case where experience would show for special conditions, which is cheapest, extra electricity, or new pots and cost of setting them.

It was discovered that the lead and iron sulphide of the matte do not dissolve in the electrolyte, so that the surface of the matte becomes cathode, and the action of the current on the start is to liberate chlorine at the anode, and reduce lead and form sodium sulphide at the matte cathode. After a short time sulphur chloride comes off, and then sulphur alone as sodium sulphide accumulates while the voltage drops.

The sodium sulphide formed does not easily dissolve in the salt electrolyte, but has considerable affinity for ferrous sulphide, and forms with it a fusible double sulphide. Yet there has been enough present in the electrolyte itself on long runs

[5] Page 383.
[4] Richards Am. Electrochemical Society, Vol. II., page 69
[6] Trans. Am. Electrochemical Soc., Vol. I., page 171.

to continuously evolve sulphur free from chlorine at the anode. Nevertheless, the solidified electrolyte on one occasion after solidification contained merely a trace of Na_2S. Probably on cooling the Na_2S separated as the salt solidified.

The tapped matte contained in one case 8.4 per cent Na_2S and 8.5 per cent NaCl. The residue contained Pb 19.4 per cent, Zn 5.1 per cent, Fe 30.5 per cent, and Ca 2.8 per cent.

Calculating the residue as sulphides the composition of the matte was

	Per Cent.
Na_2S	8.4
NaCl	8.5
PbS	22.4
FeS	48.0
ZnS	7.6
CaS	5.0
	99.9

On the assumption that the particular matte used contained 70 per cent lead and 9 per cent iron, which is not far from the truth, 48 parts of iron were added with 373 parts of lead, while 19.4 parts of lead remain, or a reduction of 96 per cent of the total lead. The remaining 4 per cent is not lost.

I mentioned that chlorine comes off with the sulphur if there is not sufficient sodium sulphide. As it is not desirable to produce chlorine with the sulphur it is necessary to return to the pots any sodium sulphide taken out with tapped matte, or replace it. This could be done by leaching the tapped matte and evaporating the solution for sodium sulphide. A consideration of the above figures shows that the amount of sulphide of sodium in the particular instance under discussion was 44 per cent of the weight of lead tapped, or only 22.6 pounds Na_2S per ton of fresh matte added. The amount is so small that the sodium sulphide could be produced directly by reducing salt-cake with carbon in a separate small furnace, to save leaching the matte tapped.

The matte tapped oxidizes very easily, and could be smelted in the blast furnace for the contained lead, along with the slag produced in the electric furnace, or better, it could be used as flux in the ore-melting process. The sodium sulphate and salt and iron oxide would be desirable fluxes.

It is not possible without actual large scale working to say just what the process would cost to operate, but it is possible to form a rough idea beforehand, and such a step is always a good one to take. It often saves laboring on processes which have no chance of success.

To add liquid matte to the furnace, handle the condensed sulphur, operate the pot, tap the lead and matte and dispose of them will require considerable labor. An estimate of 75 cents per ton of matte seems fair.

There will be no occasion for any great amount of repairs. We might roughly estimate repairs at 25 cents per ton of matte.

Twenty-five pounds of sodium sulphide, or to recover it from the matte tapped, might cost, say, 37.5 cents. Salt per ton matte, say, 8 cents.

	Per Ton of Matte.	
Power at $20 per hp.-year, .23 hp. days	$2.54	$1.27
Power at $40 per hp.-year, .23-hp. days	...	...
Labor	.75	.75
Repairs	.25	.25
Sodium sulphide	.38	.38
Salt	.08	.08
Roasting tapped matte (no fuel necessary)	.25	.25
	$4.25	$2.98
Per ton of ore (1 ton ore = .9 ton matte)	3.83	2.68
Cost per ton for melting operation (see above)	3.48	2.44
Total cost	$7.31	$5.12
Credit for 24.5 pounds sulphur at 1 cent	.245	.245
	$7.16	$4.97

The figures are estimated to cover the [illegible] and for comparison with modern [illegible] blast furnace melting, the cost of the [illegible] at not less than $4.00 per ton, and often considerably more.

It is seen so far that the new process will have a distinct competition, but the lead loss has not yet been considered. Simple melting in an electric furnace ought to give no more lead loss than melting in crucibles, in which case there is practically no loss. There is no opportunity for loss in electrolyzing the matte. The only chances for loss are in roasting the "deleaded" matte and smelting the electric furnace slag. Remembering that there is only a small portion of the lead, say 4 to 10 per cent in the matte and 1 per cent in the slag, it is not probable that the lead loss from wet assay will be as great as 1 per cent.

On the other hand, a lead smelter will lose considerable lead, say 3.5 per cent loss from fire assay, or 5 per cent from wet assay, or more. That would mean for the electric process a saving of 60 pounds extra lead, worth in this country not less than $2.40, and perhaps $3.30. This should be charged to the blast furnace cost, making it in the United States, say $7.20 or more.

There is another important consideration not to be lost sight of, and that is that the lead blast furnace requires other ores, and there are localities when no such ores can be had, and in that case the very important item of freight on the lead ore to a suitable locality must be taken into account.

Comparing the electric process with the blast furnace in a general way, the cost for labor ought not to be very different in the two processes, unless hand roasting is adopted, when the electric process has a decided advantage in this respect. With the electric process we have a cost for power of from $2.18 to $4.37, as against a cost for fuel and air blast in the roasting and reduction process of say, $1.50 to $2.00. The electric process will show no loss of lead when lead is determined by fire assay, and may come out a little ahead as against a loss of from 60-100 pounds lead per ton ore in the roasting and reduction process, and the electric process will also save about 200 pounds sulphur per ton of ore.

The roasting and reduction process can be operated successfully only where there is a plentiful supply of wet ores, while the electric process is independent in that respect.

I am very much indebted to Dr. J. W. Richards for much needed data and a number of valuable suggestions relative to the use of thermochemical and physical data for the ore melting process.

The Future of the Le Blanc and Electrolytic Alkali Works in Europe.

By JOHN B. C. KERSHAW.

An article by Hasenclever appeared in one of the German technical papers last year, discussing the future of the Le Blanc soda industry in that country. Hasenclever, who is the principal manufacturer by the Le Blanc process in Germany, considered the outlook for the industry in that country decidedly gloomy. Half the world's production of chloride of lime was stated by him to be already covered by the electrolytic alkali works. The ammonia soda works, which yield no by-products, were described as flourishing greatly, and are benefited rather than injured by the competition between the Le Blanc and electrolytic works.

The principal hope of the Le Blanc process, according to Hasenclever, lay in the discovery of new uses for sulphate of soda (one of the chief products), and in this direction the gradual growth of the new sulphur dye industry, in which

sodium sulphide was used, was expected to result in a large demand for sodium sulphate, the "salt cake" of the Le Blanc soda manufacture.

In view of the above pronouncement concerning the future of the Le Blanc alkali works in Germany, the reports of the chief companies carrying on the manufacture of alkali and chlorine products in the United Kingdom, for the year 1905, form interesting reading, especially for those who have a financial stake in the industry. The following are brief abstracts of the more important figures from the last reports of these companies:

The Castner Kellner Alkali Co. work the Castner electrolytic (mercury cathode) process at Weston Point, Cheshire, and their financial year ends on March 31. In their report for the year 1904-5 the directors state that the net profits[1] were £52,358, which, added to a carry over of £11,797, gave a balance of £64,135 available for distribution. Debenture interest and other charges absorbed £10,158 of this total, and the remaining £54,002 was divided as follows: Dividend, 4 per cent, £18,000; depreciation reserve fund, £15,000; plant patent and suspense account, £8,204; balance forwards, £12,798.

The Electrolytic Alkali Co. work the Hargreaves-Bird electrolytic process (diaphragm cell), at Middlewich, in Cheshire. Their financial year ends on Aug. 31. The report of this company for the year 1904-5 contains the following figures: Net profit, £4,746; carry over from 1903-4, £1,374. Total available for distribution, £6,120. Arrears of the dividend on the cumulative preference shares absorbed £5,601 of this amount, and £519 was carried forward to the next account.

The United Alkali Co. control the Le Blanc works in the United Kingdom, and their financial year ends on Dec. 31. The report of the directors of this company for the year 1905 has just been issued, and contains the following figures: Carry over from 1904, £43,762; net profit for 1905, £290,821; total, £334,584. This large sum is to be dealt with as follows:

Dividend on 7 per cent preference shares.....	£187,878
Transfer to reserve fund (depreciation account)...	50,000
Transfer to reserve fund (general account).....	40,000
Transfer to debenture redemption account.....	15,000
Carry forwards to 1906 account.....	41,706
	£334,584

Brunner Mond & Co. operate the Solvay ammonia soda process at Winnington and Northwich, in Cheshire, and their financial year ends on March 31. The report of this company for the year 1904-5 showed a net profit of £512,532. A dividend of 35 per cent for the year on the ordinary shares was paid out of this huge balance. £100,000 was written off the book value of the leasehold mines in Wales, £2,500 was applied to the reduction of the patents account, and a balance of £88,735 was carried forward to the next year.

A study of the above reports, in conjunction with those that have preceded them, and some knowledge concerning the inner history of the industry, leads the present writer to take a less gloomy view of the future of the Le Blanc alkali works than Hasenclever. In the writer's opinion all three types of works (Le Blanc, electrolytic and ammonia soda) are likely to exist side by side for many years to come.

The character of the chief manufactured products may change, and some indications of this change are, in fact, already to be noticed in the output of the Le Blanc and electrolytic works. Bleaching powder and caustic or carbonate of soda will no longer provide large profits for their makers by the Le Blanc or electrolytic processes. It is to the by products and special manufactures of each type of works, that one must look for the dividend of the future.

The following tabular statement, giving the average price of bleaching powder and 58 per cent soda ash in December of each year for the period 1895-1905, is of interest in this connection, since if studied in conjunction with the balance sheets of the various companies it serves to indicate the source of the profits of the three processes used in the manufacture of alkali and bleach.

Year.	Bleaching Powder.	58% Alkali.	Year.	Bleaching Powder.	58% Alkali.
1895.....	£7.00	£3.88	1901.....	£6.88	£4.88
1896.....	6.63	3.50	1902.....	6.25	4.50
1897.....	6.13	3.88	1903.....	4.00	4.50
1898.....	4.65	4.00	1904.....	4.08	4.50
1899.....	6.00	4.38	1905.....	4.50	4.50
1900.....	6.88	4.63			

The writer's conclusions regarding the future of the alkali and bleach industry generally may be summarized as follows:

The Le Blanc works will continue to produce caustic and carbonated alkali and bleaching powder, but in diminishing quantities, and the dividends earned by manufacture of these staple products (which once yielded large profits) will decline. But compensation for this loss will be found in the manufacture of sulphuric acid, sodium sulphate, hyposulphite, sulphide and other derivatives from these sulphur salts which cannot be manufactured by the processes of either of their rivals—and even the impurities of the pyrites burnt in the kilns for the lead chamber process of sulphuric acid manufacture, may be looked upon as an important source of revenue in years to come. Copper sulphate is another by-product, yielding its makers handsome profits, which can be most successfully made by the Le Blanc manufacturers. The Le Blanc Alkali Works will then depend more and more upon by-products, and less and less upon the staple products, soda and bleach, for their future profits. With this change will come the need for increased foresight and scientific supervision in their management and control.

The electrolytic works, when operating a good process, and when under wise financial management, will gradually take the place of the Le Blanc works as producers of bleaching powder, chlorates and other chlorine products. The manufacture of metallic sodium and of its derivatives, such as cyanides and peroxides, will also most probably fall into the hands of the electrolytic manufacturers. The cost of operating the electrolytic process of alkali manufacture is, however, greater than was at first estimated, and only the best processes and best equipped works will survive in the struggle for existence. The position of the electrolytic alkali industry in Europe at the present moment is of great interest in this connection, but limits of space will not permit any details relating to these works to be included in this article.

The ammonia soda works, both in this country and abroad, will continue to make enormous profits out of the manufacture of carbonate and bicarbonate of soda by the Solvay process. But, what is regarded by many as the best feature of these works, namely, the absence of by-products is, judged from another standpoint, a source of weakness. The ammonia soda works can never entirely take the place of the Le Blanc or electrolytic works, since neither sulphur nor chlorine products can be made a subsidiary and successful part of the manufacture of soda by the Solway process. All attempts hitherto made to extract chlorine successfully from the waste calcium chloride liquors of these works have failed, and the Hoepfner process of zinc extraction which has been in operation at Winnington for some years, cannot be considered to meet the demand for a satisfactory and profitable use for these waste liquors of the Solvay process. If such a process be ever found, and successfully developed, the forecasts given in this article will require revision, and their author may have to accept the fate of many another prophet, and to confess that in this world "nothing is certain but the unexpected."

London, England.

[1] All values in this article are given in English pounds sterling £

Laboratory of Applied Electrochemistry at Columbia University.

By Prof. Samuel A. Tucker.

For several years past practical work in electrochemistry has been conducted with a very modest installation placed in a portion of the Laboratory of Industrial Chemistry, but during the last year a very generous gift made possible the equipment of a laboratory devoted solely to the purpose of applied electrochemistry.

The present installation is situated in two rooms in the basement of Havemeyer Hall. Room 101 is 42 feet x 47 feet,

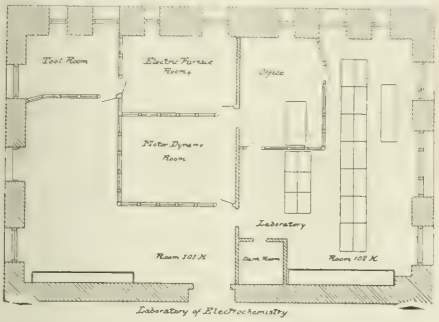


FIG. 1.—PLAN OF LABORATORY.

and room 102 33 feet x 47 feet, thus a considerable space has been provided for the use of this laboratory.

The rooms have a ceiling 25 feet high, which gives good ventilation, and the students' working desks are so situated as to have an abundance of light from long windows on two sides of the room in which they are placed.

Room 101 has been divided up by a wood and glass partition into smaller rooms, and these consist of a motor-dynamo room, electric furnace room and tool room. At present a portion of the vacant space shown on the plan of room 101

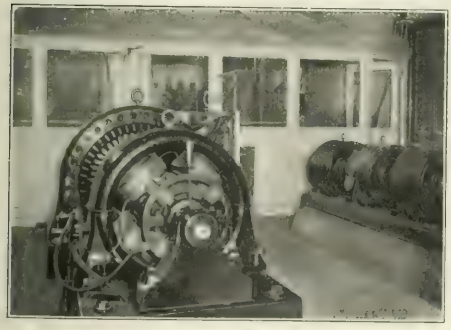


FIG. 2.—MOTOR-DYNAMO ROOM.

is occupied by some metallurgical furnaces, but it is expected that these will be removed, thus giving space for separate research rooms which are much desired.

In the motor-dynamo room are placed the generators for supplying the currents particularly adapted to the needs of the laboratory. These machines, together with the switchboard, were purchased from the Bullock Electric Co., of Cincinnati, and consists of two sets. The smaller comprise three direct-current dynamos directly connected to a 15 hp., 220-volt motor, which takes current from the power house mains. The three dynamos of this set have separately excited fields, and are wound to give 4, 10 and 25 volts, each a capacity of 30 amps. each. The current is supplied to the laboratory by a four-wire system, giving four [illegible]

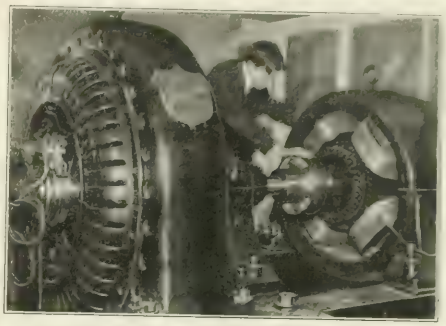


FIG. 3.—50-KW. MOTOR-DYNAMO FOR ALTERNATING CURRENTS.

By making intermediate connections, voltages of 4, 10, 14, 25 35 and 39 are available. The switchboard for this machine is provided with the necessary switches, field rheostats and overload circuit breakers. The reading instruments are Weston volt and ammeter, arranged with dial switches, with shunts for the ammeter, so that measurements may be made on each of the three dynamos. This switchboard is also equipped with a double-throw switch, so that direct current from either the 10 or 25-volt machine may be connected with the special busbars in the electric furnace room.

The other machine is designed to supply heavy alternating currents for electric furnace work, and consists of a 75-hp., 220-volt direct-current motor, directly connected to a 50-kw.

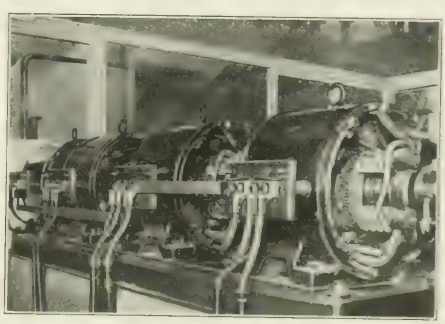


FIG. 4.—DIRECT-CURRENT MOTOR-DYNAMO SET.

alternator. The alternator is a single phase, 60 cycle machine with revolving field, excited from the 110-volt circuit of the building. The armature wires come out to a panel on the machine, with appropriate switches for connecting them in series, multiple or parallel; and as the maximum voltage of the machine is 100, voltages of 25, 50, 75 and 100, with a current of 2,000 amps. at the low voltage, are thus available. The leads from the alternator connect directly to the switchboard in the electric furnace room. Arrangements are also made for supplying alternating current to the students' desks from this machine. The motor dynamos are set on concrete foundations, and there is practically no vibration. The small direct-current set was installed in preference to a storage battery, because the maintenance of the battery is a consider-

expense and requires very constant attention, although the first cost is somewhat in favor of the battery. Movable storage batteries are provided to meet the requirements of special cases.

The laboratory is provided with twenty double working-desks or tables for students, giving an unusually large working space. The tops are of Alberine stone, and each section is supplied with gas, water, blast and suction. An oak shelf supports the switchboards, of which there is one for each student. These are of special design, enclosed in oak cases with glass doors, thus keeping switch and other contacts away from chemical fumes. Two switches are mounted on each board, from which connection may readily be made to any of the six fused terminals. The currents provided are from the low-voltage direct-current set as mentioned, and 110 volts direct current. These last terminals may be changed at will to alternating current, when the alternator is running, by means of a double-throw switch placed in the motor-dynamo room.

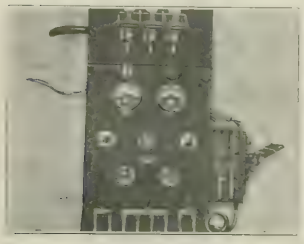


FIG. 5.—SWITCHBOARD FOR MOTOR-DYNAMO ROOM.

FIG. 6.—SWITCHBOARDS ON STUDENTS' DESKS.

The working desks are lighted by half-shade lamps placed over each student's switchboard. The measuring instruments are the Weston portable. One ammeter and one voltmeter are supplied to each student at the beginning of his work. These give three scale readings from zero to 3, 15 and 150 in 150 divisions of a volt, and in amperes from zero to 1.5, 15 and 150 in 150 divisions of an ampere by interchangeable shunts.

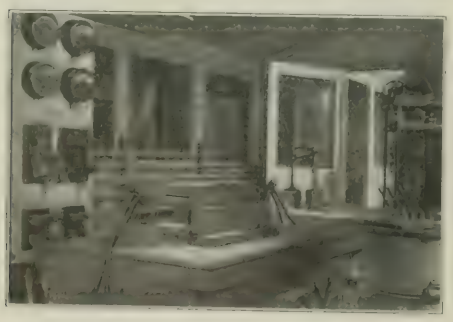


FIG. 7.—GENERAL LABORATORY ROOM.

Suitable thermometer and electrolytic tanks form part of the regular laboratory equipment. The electric wiring is carried through iron pipe in a wire subway to a master and the current is usually turned on and cut off at the head of one end of the laboratory, as well as at a central switch-table for the use of larger apparatus than can be conveniently handled on the regular students' tables.

A set of platinum electrodes has been purchased for electrolytic work, as well as a number of small motors for stirring purposes. These motors are fitted with change gearing to give three speeds. The timepiece of the laboratory is a sixty-day regulator, with seconds beat-pendulum arranged

FIG. 8.—LOW-VOLTAGE RHEOSTAT.

with electrical contacts, so that the time factor in certain experiments may be accurately determined. A thermoelectric pyrometer by Siemens & Halske, reading to 1,500° C., and a Warner optical pyrometer for higher temperatures, have been included, as well as two chemical balances.

The electric furnace room adjoins the motor-dynamo room, and a new cement floor has been laid here, as the usual asphalt floors of the building are most unsuitable for this work. This room is enclosed by a galvanized roof, in order to keep the dust from the machinery in the adjoining rooms. On one side of the room is placed the switchboard controlling the bus-bars, which are 16 feet long, and of which there are three sets.

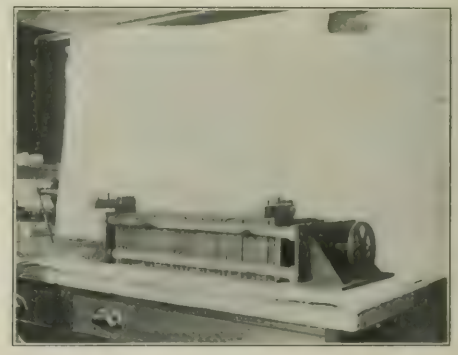


FIG. 9.—CARBON RHEOSTAT.

The bars are of aluminum, supplied especially for this purpose by the Pittsburg Reduction Co. The currents provided at these bus bars are 200 amps. at 10 or 25 volts direct current from the small motor-dynamo set, 300 amps. at 110 volts direct current from the power house, and alternating current up to 2,000 amps. from the alternator. The switchboard is equipped with heavy switches, circuit breakers and measuring instruments. Connections are made from the bus bars to the tables on which the furnaces are set by means of one or more cables depending upon the capacity of the furnace. The tables

are of wood, with fire-brick tops laid in cement. Separate switches, electrode holders, rheostats and measuring instruments complete the outfit for this part of the laboratory. The field rheostat for the alternator is operated from this room as well as from the motor-dynamo room, thus giving control of this machine while the furnaces are in operation.

The tool room is supplied with a 14-inch engine lathe, 16-inch shaper, sensitive drill and grinder, and a set of small tools used for repairs and for making appliances for the laboratory. Besides these tools, a heavy polishing and buffing head is installed in another part of the room, operated by an independent motor, and is used principally in electroplating work.

A brief description of some special appliances as used in the laboratory is mentioned, because these details are of importance to any one doing work in this subject.

For rheostats to be used in connection with low voltage and of a capacity of about 8 amps., the following was adopted: An iron frame work, 13 inches x 24 inches, mounted on a rectangular piece of Alberine stone, supports the porcelain insulators. The resistance wire is wound over these insulators, which are secured to the angle-iron cross pieces. The resistance wire is No. 15 german silver, and a sliding clip connection makes it possible to get fine adjustments of the current.

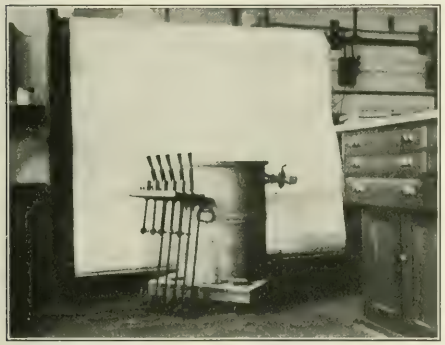


FIG. 10.—ARC FURNACE RHEOSTAT.

Another form of rheostat designed for heavier currents is made up of thirty-five carbon plates, 4 inches x 7 inches x ½ inch, arranged in a frame so that pressure may be applied by the hand-wheel screw. The resistance is 3.3 ohms, and forms a convenient and compact apparatus for use up to 25 volts. Too high a voltage is apt to cause arcing with consequent destruction of the plates.

For arc furnaces, a water-cooled resistance is in use, and has been a very practical form of rheostat. A slate top supports the bars and five switches, which are connected to copper bars, from which the resistance wire is strung. On each section there are three resistance wires, No. 17 1a, 1a, each 10½ feet long, spirial wound, each step being in parallel. The wires are water-cooled by placing in an ordinary galvanized ash can, means being provided for water circulation.

Each step has a resistance of about 1.5 ohms., and the carrying capacity, with all steps in, is about 400 amps.

A form of reciprocating stirrer for use in studying the deposits of copper under varying current density is shown. All connecting cables used in the electric furnace room are provided with brass tips and fit the various lugs and connections, which are all reamed to ½ inch holes.

The electrode holders are of iron for the most part, and will hold the electrode in a vertical or horizontal position at any desired elevation. The vertical adjustment is by a friction clamp, which fits any of the uprights provided on the electric furnace tables.

The laboratory has been designed to provide facilities for the study of practical electrochemistry which is taken by chemists and electrical engineers, and also research work which is offered to graduates for higher degrees.

The courses offered in the laboratory are as follows:

1. Practical Electrochemistry.—Electroplating, influence of current density, temperature, concentration, etc., electrolysis with molten electrolytes, the use of diaphragms, study of typical industrial processes, electric furnace practice. This course comprises five afternoons of laboratory during one term, or its equivalent in two terms, and two lectures a week during one term on the applications of electrochemistry in the arts.

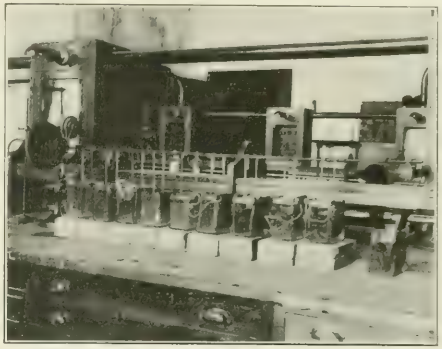


FIG. 11.—RECIPROCATING STIRRER FOR COPPER BATHS.

2. Practical Electrochemistry for Electrical Engineers.—Experiments in electric furnace work and the preparation of calcium carbide, aluminium, aluminium bronze, carborundum and metallic chromium; illustrations of the wet processes in electrochemistry. This course is given one afternoon during part of the second term.

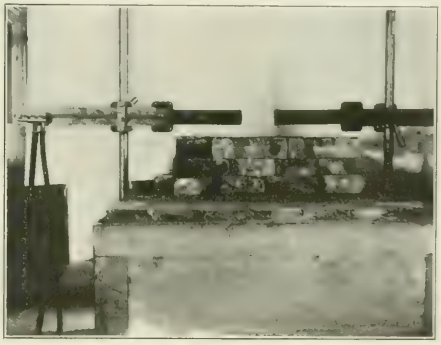


FIG. 12.—ELECTRODE HOLDERS.

3. Electrochemical Research.—Original investigation and research equivalent to 16 hours a week during one year.

4. Advanced Electrochemical Research.—Private study and research, original investigation and research during two years.

The following courses are in contemplation:

1. A practical course in the practice of electroplating and electrotyping, and the construction, maintenance and testing of storage batteries.

2. The preparation and study of the formation of typical organic compounds by electrolysis.

Thus far research work has been done and investigations have been conducted in the laboratory on the following subjects: Potassium percarbonate, potassium chlorate, ammonium and potassium persulphate, electrolysis of calcium chloride with reference to the formation of chlorate, electric furnace for laboratory use, ethylene from inorganic sources, production of aluminium in the electric furnace with the aid of calcium carbide, some hitherto unknown metallic borides, production of abrasive compounds in the electric furnace, the study of polarization by alternating currents, the reduction of sodium nitrate by electrolysis, preparation of metallic lithium, formation of spongy lead for storage battery plates, arc light electrodes and zirconium carbide, boron carbide, heat conserving qualities of certain refractory materials for electric furnace work, production of ferroalloys in the electric furnace, metallic calcium, kryptol and coke resistance furnaces, electrolysis of metallic oxides dissolved in boric oxide, electrolysis of chromium salts, measurement of temperature in resistance furnaces, melting points of refractory materials used in electric furnace construction, resistance furnaces.

Factory Scale Experiments With Fused Electrolytes—II.

By Edgar A. Ashcroft.

The principle of magnetic rotary agitation of the contents of the electrolytic cell, as described in my last article, has been successfully applied in two leading directions, and experiments have been more or less successfully conducted in others. The first of these applications is the direct electrolysis of sulphides (either mixed or singly). Several commercial processes are possible from this application of electrolysis accompanied by continuous mixing or agitation.

Fig. 3 represents the arrangement of plant for such a process, which consists in electrolyzing the ore suspended in any fused medium, such as halogen salt, between a solid indestructible anode and a fluid cathode, composed of the metal under recovery. Many forms and modifications of such a generally applicable process at once suggest themselves. In order to understand these more clearly the following reactions, all of which have been ascertained experimentally, should be studied:

Taking zinc chloride (most conveniently with a molecular proportion of sodium chloride, which for all practical purposes of reaction remains inert, although it substantially reduces the electrical resistance of the bath), we have results as follows:

Reaction A.—Zinc sulphide freely suspended does not react until a current is passed, when the net result of all reactions (probably very complicated in their intermediate forms), is that zinc sulphide is resolved into its elements, zinc and sulphur, which appear respectively at the cathode and the anode, thus:

$$ZnS + \text{current} = Zn \text{ (cathode)} + S \text{ (anode)}$$

If this operation is carried on at a temperature of about 450° C. the zinc is received molten and the sulphur distills as vapor, for the melting point of zinc and the vaporizing point of sulphur are nearly identical.

The original discovery of this reaction is due to J. Swinburne (British patent of 1896). This is a perfectly practical process for the treatment of pure blendes or those containing only siliceous gangues (such as crystalline garnets, quartz, etc.), which are not affected by the zinc chloride electrolyte.

The theoretical e. m. f. of decomposition for zinc sulphide is only 0.9 volt, and the practical working e. m. f. will probably not exceed 2.5 volts. This method of treating blende ores has the great advantage that no preliminary roasting is necessary. When sulphuric acid is to be manufactured from the blende simultaneously, it would be quite feasible to burn the sulphur, as it escapes from the cell by admitting air, and convey the SO_2 produced directly to the chambers (or other oxidizing device). In this way the expenses of fixing and operating either roasting plant or burners will be obviated, and the heat of combustion will be usefully employed in reducing the electrical energy necessary to maintain the heat of the fused charge. This process is very attractive at first sight, and has much to recommend it actually. But its limitations are greater than at first appear.

The desiderata for such a process of zinc recovery are:

1. Ores free from any harmful impurity which would contaminate the electrolyte.

2. Ores rich in blende and with little inert matter so as to avoid frequent cleaning out of the contents of the cell.

3. Cheap electrical energy near to the ores and the markets.

4. Efficient means for continuous agitation to prevent solid particles settling out on the cathode.

It is rather doubtful, in my opinion, in spite of the great apparent possibilities of such a process of blende reduction whether these conditions exist to-day or are at all likely to exist in future, except in a few small and isolated cases. The principal obstacles to commercial success are the impurities usually contained in the ore. For many ores contain both soluble gangue, such as horn blende and calcite, and also much iron manganese, etc., as sulphides, and few zinc ores which I have ever encountered are pure enough for practical success. Nevertheless, the study of these reactions is both interesting and may be important in other connections.

Reaction B.—Lead sulphide in a medium of fused zinc chloride reacts as follows:

$$ZnCl_2 + PS = ZnS + PbCl_2$$

This reaction (the discovery of which, I believe, is originally due to Ganelin and Swinburne simultaneously) has been made the basis of several processes which we are not concerned with here.

On electrolyzing such a mixture, lead appears at the cathode (as long as any lead is present in the electrolyte), and, as before, sulphur distils at the anode. As lead is the base we are dealing with here, it is quite practicable to substitute lead chloride for the zinc chloride of the last experiment, and it is quite indifferent to the practical result whether this is done wholly or in part or not at all. Lead sulphide dissolves in lead chloride, forming a dark-colored fusible mass. Here we are getting much nearer to commercial conditions, for many rich deposits of galena exist, and in fact the ores of commerce suitable for the reverberatory furnace reduction methods are almost perfectly pure galena. Precaution must be taken, however, to minimize any reaction between the metallic cathode and the suspended sulphide (forming lower sulphides, etc.), and the best means to this end is to keep the total sulphur contents of both as low as possible. A great advantage exists for this method when silver has to be recovered in small quantities, for by working with two sets of cells (or periodically changing the cathode metal, which comes to the same thing) all the silver can be concentrated in a rich bullion, from which it can be cheaply recovered by direct cupellation, so that much of the expense of the ordinary refining process is eliminated.

I am of the opinion that in spite of the cheapness of the furnace method with which it has to compete, the electrolytic reduction of galena by this method is quite a commercial proposition, and may some day be extensively applied. Three considerations will tend to help in this direction:

1. The growing scarcity of the purer galenas will bring more and more ore into the market, having some silver and some zinc contents (like the Broken Hill and Leadville lead concentrates), and the recovery of both zinc, silver, sulphur and lead are effected by the electrolytic method, whilst some of the silver and all of the zinc and sulphur in such concentrates is lost in the furnace process.

2. The growing cheapness of electrical energy

3. The device described in my last article for magnetically agitating the suspended particles of ore in the cell, which removes one of the great practical difficulties encountered by Swinburne and the writer when we first attempted to apply this idea as far back as 1899.

4. The progress of modern concentration methods which renders the complete elimination of iron (the most troublesome impurity) from the ore, in many cases now feasible. The theoretical e. m. f. for lead sulphide is only 0.4 volts, and the practical working e. m. f. when sulphur is also oxidized in the cell need not exceed 1.5 to 2.0 volts, so that, considering also the high ampere equivalent of lead, a very large output is possible from a given plant in comparison with other metals.

Reaction C.—Antimony sulphide (and many other sulphides, such as arsenic, etc., etc.,) reacts in a like manner with zinc chloride, thus:

$$Sb_2S_3 + 3ZnCl_2 = Sb_2Cl_6 + 3ZnS$$

Here a fortunate circumstance occurs, for the volatile chloride of antimony completely leaves the mixture (it vaporizes at 200° C. and fuses at 70° C.), and may be separately condensed, and thus a ready means is obtained for eliminating a troublesome impurity which frequently occurs in lead ores. On this reaction have also been based several processes, having for their object the treatment of antimony and antimonial gold ores, but the consideration of the reactions of these would lead us too far afield from our subject.

Reaction D.—Copper (cuprous) and iron (ferrous) sulphides react with zinc chloride, but not completely or sharply. (This appears to be one of those cases where reactions go half way and then attain a state of equilibrium and stop.) As neither ferric nor cupric salts can exist in the presence of the sulphide reducing influence, the problem of the reaction of these metals is greatly simplified and falls under the following *reactions E, F, G, H,* etc.

A whole series of substitution reactions take place between metals and fused chlorides, the following of which are typical:

E.—$PbCl_2 + Zn = ZnCl_2 + Pb$

F.—$2AgCl + Pb = PbCl_2 + 2Ag$

G.—$FeCl_2 + Zn = ZnCl_2 + Fe$

H.—The reactions of ferrous chloride with lead are especially interesting and important, as it has been found that they go opposite ways at different temperatures, and on this peculiarity the writer has based one method of eliminating iron in the mixed sulphide process. Thus, below 450° C.:

$$PbCl_2 + Fe = FeCl_2 + Pb$$

but at about 600° C. or less:

$$FeCl_2 + Pb = PbCl_2 + Fe$$

By making use of the latter reaction it is possible to remove all the iron from a mixture of fused chlorides by agitating the mixture to a temperature of 600° C. with metallic lead, and agitating it well. The iron alloys with the excess of lead below, and as long as the alloy is slowly agitated and the contents of iron does not exceed some 5 per cent, it remains freely fluid. Afterwards the alloyed metal thus obtained may be treated by a settling out and skimming process, whereby a caked alloy of 25 per cent iron contents and a remainder of nearly pure lead is produced. The rich alloy of iron may be melted in a cupola furnace and the lead recovered, and the nearly pure lead from which the rich alloy has been skimmed may be used again for precipitation of more iron. This process has importance when iron has to be removed from the fused bath as a preliminary to the precipitation of pure zinc, because iron can be much more cheaply removed from its alloy with lead than it can from its alloy with zinc. It is to be remembered, however, that for purposes of calculation in all such reactions an equivalent of electrical energy must be allowed corresponding to that required to precipitate the metal dissolved, or, in other words, corresponding to the atomicity of the iron removed.

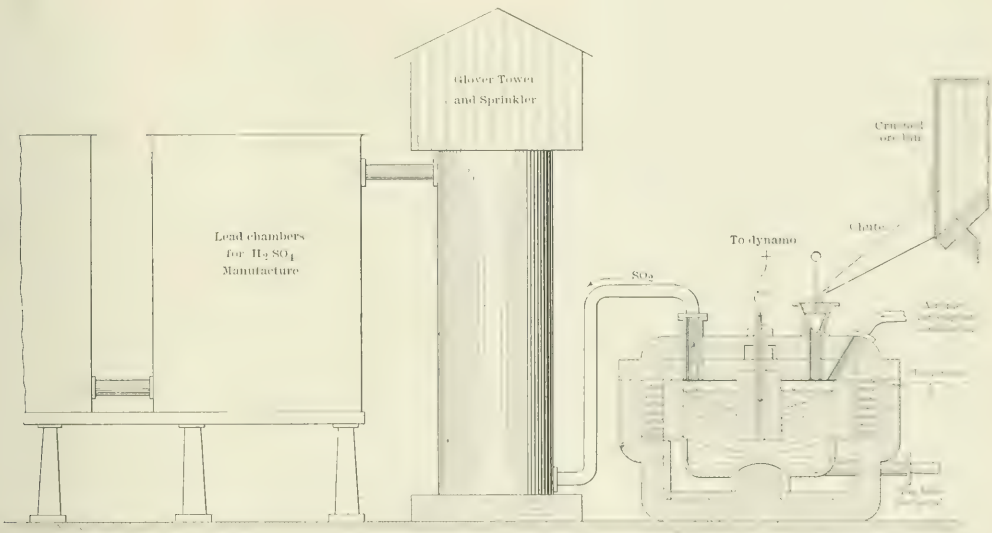


FIG. 3.—ARRANGEMENT OF PLANT.

Reaction I.—The more electropositive metals, of which sodium is the most important and typical, will practically precipitate all the others. Thus:

$$Na + PbCl_2 = NaCl_2 + Pb$$

$$Na + ZnCl_2 = NaCl_2 + Zn$$

Reaction K.—Finally, sodium sulphide introduced into a bath of either zinc or lead chloride behaves in a similar way to lead sulphide when placed in zinc chloride, the chlorine

going to the more electropositive base and precipitating a sulphide of the lower metals. Thus:

$$Na_2S + PbCl_2 = 2NaCl + PbS$$

or $$Na_2S + ZnCl_2 = 2NaCl + ZnS$$

The zinc and lead sulphides thus formed are of the light-colored, finely divided variety, and are easily maintained in suspension and freely acted upon by the anodic products of nascent chlorine in the electrolytic process.

I have no doubt at all that the proper way to chemically regard all these reactions is to consider the direct electrolysis of zinc (or lead chloride) as the primary reaction, producing

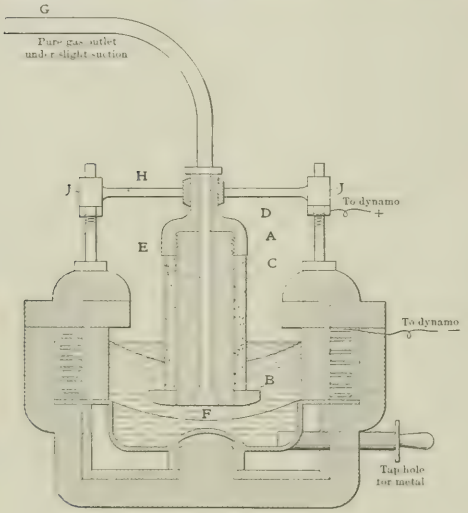


FIG. 4.—MODIFIED FORM OF ELECTROLYTIC CELL.

metal at the cathode and nascent chlorine at the anode which reacts with the suspended or dissolved particles of sulphide, setting free sulphur, reforming chlorides and simultaneously yielding up a quota of electrical energy. It was at one time supposed that iron salts were necessary as an intermediary for the exchange of chlorine to the sulphides, but this has been clearly proven not to be the case.

The second direction in which the principle of magnetic rotary agitation has found useful application is in connection with a new process of the writer's for cheap production of metallic sodium and some allied processes, making use of alloys with metallic lead and a double electrolytic apparatus with diverse electrolytes.

In my next article I will describe some applications of this double cell apparatus which was primarily devised for the sodium process, notice of which has already appeared in these columns.

In the meantime I will conclude this article with a description of a modification of the magnetic cell, which I have devised for use when (as sometimes happens) frequent and free access to the contents of the cell is necessary for economical working.

Fig. 4 illustrates such an apparatus, and the following references to the cut will explain its parts:

REFERENCES.

A. Graphitized carbon block.

B. Graphitized carbon shoe threaded to block.

C. Protecting shield of fire-clay tubulure.

D. Metal cap for carbon block.

E. Porcelain-lined copper tube, making electrical connection with carbon and serving for gas passage.

F. Spiral passages to assist projection of gas toward center of anode.

G. Outlet for gas (maintained under slight suction).

H. Yoke for support and adjustment of electrode.

J. Adjusting nuts.

By employing this form of modified anode and open electrolytic cell the gases (chlorine, etc.) can be caused to collect inwards in the center of the cell, and may be then withdrawn in a very rich and concentrated form, suitable for certain industrial processes where this condition is of great importance. The construction also affords free access to the cell for charging raw material and removing accumulations of dirt, etc., and allows of a ready adjustment of the position of the anode, both points of great importance in certain operations which I shall describe later.

NOTE.—By a slight oversight in Fig. 1 of my last article the lines showing the electrolyte in the zinc cell were omitted, and the words "tightened casting" in same figure should read "lightened casting."

Metallurgical Calculations.

BY J. W. RICHARDS, PH. D.,

Professor of Metallurgy in Lehigh University.

CALCULATION OF THE CHARGE OF THE BLAST FURNACE.

In the last instalment we discussed the balance sheet of materials entering and leaving the furnace, showing the distribution of the ingredients of the charge and the blast into the various products and by-products of the furnace. We did not there go into the question as to how the proportions of the charge are determined by the metallurgist in charge of the furnace. There are, however, very few factors of the charge which can be controlled at will. The blast furnace reduces practically all of the iron present in the ore into the pig iron, and, therefore, if the ore contains 50 per cent of metallic iron and the pig iron 90 per cent, it will take $0.90 \div 0.50 = 180$ parts of ore to furnish the iron in 100 parts of pig iron. The amount of ore to be used per unit of pig iron made is therefore fixed by the richness or poverty of the ore, and is not capable of variation. The amount of fuel used per unit of pig iron made is not fixed *a priori*, as is the amount of ore, but is governed by the caloritic requirements of the furnace while in operation. If the pig iron and slag run colder than they should, it is evident that more heat must be put in or developed within the furnace, which the manager promptly proceeds to accomplish by increasing the temperature of the air blast (if he can), or by relatively increasing the amount of fuel in the charge (which he always can). The ratio of the weight of the ore and flux in the charge to the weight of fuel used is called the burden of the furnace, and in practice it is usual to charge at one time a fixed weight of fuel, and to vary the burden according to the heat requirements of the furnace. Changing the burden is therefore only another expression for changing the relative amount of fuel used, and this is varied simply from observation of the temperatures of iron and slag and the conclusions then from as to whether the burden is too heavy or unnecessarily light. The amount of blast used is another factor relatively fixed per unit of pig iron produced. Blow more blast, and more pig iron is produced; blow no blast (bank the furnace), and no pig iron is made; the ratio is not quite exact, but it is quite nearly true that, other conditions being equal, the output of pig iron is nearly proportional to the amount of blast blown. Variations in the temperatures of the blast produce important changes, which will be separately discussed.

The amount of flux used is really the one factor in which the manager has the greatest freedom of action. The amount of this indispensable substance used is determined by many factors, and can be varied between quite wide limits without

fundamentally deranging the furnace. It is here a question of using sufficient flux in the charge to make with the unreduced constituents of ore and ash of the fuel a slag which shall be well-fused at the temperature of the furnace, shall carry off considerable sulphur, if much is present, and shall not corrode the lining of the furnace. These considerations are so important, and often so little understood, that we will discuss them more at length.

Calculation of the Flux and Slag.

From the balance sheet of problem 51 it will be seen that the metallurgist running the Swedish blast furnace at Herräng, used, per 1,000 of pig iron made, 1530.2 of ore, 115.8 of limestone flux, 682 of charcoal, and blew in 2416.8 parts by weight of blast. It may with safety be believed that the amount of charcoal used was the minimum which he found by experience necessary to keep his furnace at proper temperature; and most American blast furnace managers will wonder how he could get along with so little. The amount of ore used was the necessary proportion to furnish the iron. The amount of blast was probably all that could be gotten out of the blowing apparatus with which the furnace was provided. Finally, the amount of flux was that amount necessary to make a proper slag. Let us investigate the question as to how its amount was determined and the characteristics of the proper slag.

The ingredients of the slag produced in the case in question are, from the balance sheet:

	From Ore.	*From Flux*	*From Fuel*	*Total.*
SiO^2	69.6	3.6	1.3	74.5
Al^2O^3	11.6	0.4	...	12.0
CaO	34.0	62.2	5.9	102.1
MgO	14.8	0.2	0.7	15.7
FeO	1.2	0.2	2.0	3.4
MnO	9.3	...	...	9.3
K^2O	...	...	3.4	3.4
CaS	0.2	...	0.25	0.45
	140.7	66.6	13.55	220.85

And the percentage composition of the slag:

SiO^2	= 33.73	Per Cent.	FeO	= 1.54	Per Cent.
Al^2O^3	= 5.43	"	MnO	= 4.21	"
CaO	= 46.23	"	K^2O	= 1.54	"
MgO	= 7.11	"	CaS	= 0.20	"

The crucial question now presents itself, "What guided the metallurgist in choosing the quantity of limestone used, and in making a slag of the above composition?" The answer to this will develop the whole practice of fluxing.

Primarily, the fundamental guide in this matter is previous experience, as revealed in recorded analyses of slags which have worked properly. Such analyses have been made for a hundred years, and freely published; they show what compositions of slag have been found practicable and suitable in blast furnace practice. As reported by the chemists, the analyses show the varying percentages of SiO^2, Al^2O^3, CaO, MgO, FeO, MnO, etc., found in actual slags made in successful practice, and this information would be the metallurgist's guide in calculating the amount of flux to use to produce a good slag. On studying these analyses, we find that silica and lime are the predominating constituents of all blast furnace slags, the reason being that silica is the principal material to be fluxed, and that lime (from limestone) is the cheapest material which will flux it, and form a fusible slag.

Analyses of numerous blast furnace slags show the following variations of composition:

SiO^2	25 to 65	Per Cent	FeO	0 to 6 Per Cent
Al^2O^3	3 " 30	"	MnO	0 " 14 "
TiO^2	0 " 10	"	K^2O	0 " 3 "
CaO	12 " 50	"	Na^2O	
MgO	0 " 18	"	CaS	0 " 0 "

The above limits are not reached simultaneously by one and the same slag. The ordinary variations [illegible] as follows (according to Ledebur):

	SiO^2	Al^2O^3	CaO	MgO
Producing gray iron, using charcoal	45—55	4—5	45—25	
Producing gray iron, using coke	30—45	15—?	40—55	
Producing white iron, using charcoal	45—50	10—5	45	
Producing white iron, using coke	30—40	10—5	60—55	
Producing spiegeleisen, using coke	30	10	55—45	

In the latter case there is present 5—15 per cent of MnO.

Among these varied compositions, however, there are certain degrees of fusibility, and of suitability to the blast furnace's needs. The most easily fusible slags are those (considering only the main ingredients) which contain 35 per cent of lime, and in which, if alumina is present, each 1 per cent of alumina is balanced by the presence of 0.5 per cent additional lime. This rule gives good slags up to about 65 per cent of lime and alumina counted together. Another observation is, that with 33 to 40 per cent of lime in the slag, the amount of silica and alumina together being some 60 to 70 per cent, it is possible to change the relations of silica to alumina within large limits (*i. e.*, from 40 to 50 silica and 20 alumina to 25 or 35 silica and 35 alumina) with very little effect upon the fusibility of the slag. On the other hand, with a low proportion of silica in the slag, say 30 to 40 per cent, it is equally possible to change the relations of lime to alumina within large limits (*i. e.*, from 50 lime and 15 alumina to 35 lime and 35 alumina) with very little effect upon the fusibility of the slag.

The blast furnace manager usually decides upon the kind of slag he will make, on one of three or four assumptions:

(1) If there is little alumina present, and practically no magnesia, he usually assumes some ratio between the weights of silica and lime which he desires his slag to possess, and calculates the weight of flux necessary to make slag conforming to that condition. In charcoal furnaces, where there is practically no sulphur in the charge, the silica may be 1.5 to 2.0 times the lime present; in coke furnaces, where it is necessary to eliminate much sulphur, this ratio is usually 0.5 to 1.0.

Illustration. If a ton of iron ore carries 300 pounds of silica, how much limestone, which is practically pure calcium carbonate, will be required to slag it, neglecting flux necessary to slag the ash of the fuel?

Solution: Pure limestone is $CaCO^3$, or $CaO \cdot CO^2$, and carries 56 per cent of lime, which will go into the slag, and 44 per cent of carbone oxide (CO^2), which goes into the gases. Using x pounds of limestone, the weight of lime going into the slag is $0.56\,x$. If we assume the ratio of silica to lime = 1 for a coke furnace, and 1.75 for a charcoal furnace, we get the two equations and corresponding values of x:

$$\frac{300}{0.56\,x} = 1 \quad \text{whence } x = 536 \text{ Pounds}$$

$$\frac{300}{0.56\,x} = 1.75 \quad \text{whence } x = 307 \text{ "}$$

(2) If considerable magnesia is present it is usual to either count it simply as so much lime, or else to calculate the weight of lime to which it is chemically equivalent, and add this to the lime, calling the sum the "summated lime." The ratio of silica to lime is then used as the ratio of silica to lime plus magnesia, or of silica to summated lime. When there is considerable magnesia present the chemical summation should always be made. Small amounts of MnO, FeO, and K^2O or Na^2O are also chemically summated as lime. The chemical summation is based on the fact that one molecule of a base containing one atom of oxygen is considered the equivalent in fluxing power

of any other similar molecule; *e. g.*, CaO, MgO, FeO, MnO, K^2O, Na^2O are considered as equivalent; but as these molecules weigh differently, we have equivalent fluxing powers in the following weights:

CaO	56
MgO	40
FeO	72
MnO	71
K^2O	94
Na^2O	62

from which we conclude that since 40 parts by weight of magnesia is equivalent to 56 parts of lime, that, therefore, the lime equivalent of any weight of magnesia is $\frac{56}{40}$ of the weight of the magnesia. Similarly, we get the lime equivalent of these bases as follows:

CaO equivalent of any weight of MgO $= \frac{56}{40} \times$ weight MgO

" " " " FeO $= \frac{56}{72} \times$ " FeO

" " " " MnO $= \frac{56}{71} \times$ " MnO

" " " " K^2O $= \frac{56}{94} \times$ " K^2O

" " " " Na^2O $= \frac{56}{62} \times$ " Na^2O

Illustration: An iron ore contributes to the slag 350 pounds of silica, 12 of FeO and 60 of MnO. It is desired to flux it by using limestone containing 38.1 per cent lime, 13.6 magnesia, 3.4 silica, and 44.9 carbonic oxide (CO^2). How much flux must be used to produce a ratio of silica to summated lime = 0.8?

Solution: Letting x be the pounds of limestone used, the ingredients of the slag will be

SiO^2	$= 350 + 0.034\ x$
CaO	$= 0.381\ x$
MgO	$= 0.136\ x$
FeO	$= 12$
MnO	$= 60$

The lime equivalents of the MgO, FeO and MnO are:

CaO equivalent of MgO $= 0.136\ x \times \frac{56}{40} = 0.1904\ x$

" " FeO $= 12 \times \frac{56}{72} = 9.33$

" MnO $= 60 \times \frac{56}{71} = 47.32$

" CaO $= 0.381\ x \times 1 = 0.381\ x$

Summated CaO $= 0.5714\ x + 56.65$

The ratio of silica to summated lime is therefore:

$$\frac{350 + 0.034\ x}{56.65 + 0.5714\ x} = 0.8$$

Whence $x = 720$ pounds.

It is useful to note that the method of solution, calling x the weight of the whole and then getting expressions for the weight of each ingredient in the slag and the weight of the whole slag, is a very general solution which is applicable to any kind of assumed composition to which it is desired that the slag shall conform.

(3) If alumina is present in the slag-forming constituents of the ore it also may be reckoned with in several ways. It may be reckoned as so much by weight, and added in as such to the silica or the lime, or it may be calculated to its silica or its lime equivalent, and added into the summated silica or the summated lime. Here we touch on a question which has agitated blast furnace managers and theorists for a generation: Should the alumina be reckoned with the bases or with the acids; summated as silica or as lime? It would be presumptuous to set forth a dictum on a subject which has been so long and so ably discussed by some of the best iron metallurgists, but we will assume, as somewhere near the truth, that as far as the elmination of sulphur in the slag is concerned, alumina acts in slags low in silica as though it were lime, not in the proportions of its lime equivalent

$$\left(= \frac{168}{102} \times \text{weight of alumina} \right)$$

but rather in about the proportions of its simple weight. As far as fusibility is concerned, in high silica slags alumina increases the fusibility up to a certain point, above which it decreases it. It acts in these, therefore, like lime, and may be classed with the bases. In low silica slags, below 45 per cent, alumina acts like silica when considerable is present, and like lime when less is present; for instance, in a low lime, high alumina slag, alumina and silica may be substituted one for the other within wide limits, without materially affecting the fusibility of the slag; in a high lime, high alumina slag, alumina and lime may be substituted for each other within wide limits without sensibly changing the fusibility. To state the matter as succintly as possible, in slags low in silica (30 to 35 per cent), alumina reinforces the bases in the elimination of sulphur; in regard to fusibility, it acts like silica in a slag low in both silica and lime, and like lime in all other blast furnace slags.

Illustration: An iron ore carries 10 per cent of its weight of silica and 6 per cent of alumina. The limestone on hand contains 37.3 per cent lime, 13.3 magnesia, 3.3 silica, 44 carbonic oxide (CO^2), and 2.1 per cent alumina. How much flux is required per 1,000 parts of ore to make (*a*) a slag with 49 per cent of silica plus alumina; (*b*) a slag with 33 per cent silica; (*c*) a slag with summated silica = the summated lime?

Solution:

(*a*) Weight of SiO^2 in slag $= 100 + 0.033\ x$

Weight of Al^2O^3 in slag $= 60 + 0.021\ x$

Weight of CaO in slag $= 0.373\ x$

Weight of MgO in slag $= 0.133\ x$

Total weight of slag $= 160 + 0.560\ x$

therefore,

$$160 + 0.054\ x = 0.49\ (160 + 0.560\ x)$$

whence $x = 370$

(*b*) $100 + 0.033\ x = 0.33\ (160 + 0.560\ x)$

whence $x = 313$

(*c*) Silica $= 100 + 0.033\ x$

Silica equivalent of alumina $= \frac{180}{204}\ (60 + 0.021\ x)$

Summated silica $= 153 + 0.0515\ x$

Lime $= 0.373\ x$

Lime equivalent of magnesia $= \frac{56}{40}\ (0.133\ x)$

Summated lime $= 0.5592\ x$

therefore $153 + 0.0515\ x = 0.5592\ x$

whence $x = 300$

Returning to the slag resulting from the proportions of flux chosen in Problem 51, and containing

SiO^2	33.74 Per Cent	FeO	1.54 Per Cent
Al^2O^3	5.43 "	MnO	4.21 "
CaO	46.23 "	K^2O	1.54 "
MgO	7.11 "	CaS	0.20 "

We see that its percentage of silica is low, therefore it is adapted to produce pig iron low in sulphur; its percentage of alumina is low, and therefore its presence increases the fusibility of the slag, which would otherwise be rather deficient, because of the high lime and somewhat considerable amount of other bases. In such a slag, alumina would be summated with the bases

The ratio of silica to bases is $\frac{33.74}{66.06} = 0.516$

The ratio of silica to lime plus magnesia is $\frac{33.74}{53.33} = 0.632$

The summated lime $= 46.23$

$+$ CaO equivalent of $Al^2O^3 = \frac{168}{102}(5.43) = 8.94$

$+$ CaO equivalent of $MgO = \frac{56}{40}(7.11) = 9.95$

$+$ CaO equivalent of $FeO = \frac{56}{72}(1.54) = 1.20$

$+$ CaO equivalent of $MnO = \frac{56}{71}(4.21) = 3.32$

$+$ CaO equivalent of $K^2O = \frac{56}{94}(1.54) = 0.92$

70.56

Ratio silica to summated lime $= \frac{33.74}{70.56} = 0.478$

Problem 52.

In a blast furnace charge, consisting of 1530.2 pounds of ore and 682 pounds of charcoal, per 1,000 pounds of pig iron made, it is known from the balance sheet of above materials that they will contribute to the slag the following slag-forming ingredients. (See balance sheet, Problem 51.)

SiO^2	70.9 Pounds	FeO	3.2 Pounds
Al^2O^3	11.0 "	MnO	0.3 "
CaO	39.9 "	K^2O	3.4 "
MgO	15.5 "	CaS	0.4 "

The limestone at hand contains:

CaO	53.74 Per Cent.	Al^2O^3	0.32 Per Cent
MgO	0.17 "	FeO	0.18 "
SiO^2	3.14 "	CO^2	42.42 "

Required. The weight of limestone to be used to make (1) A slag containing 33.74 per cent of silica (2) A slag in which the ratio of silica to bases is 0.516. (3) A slag in which the ratio silica to summated lime is 0.478.

Solution: This problem embodies the conditions which confront the metallurgist when desiring to calculate the flux needed by any given furnace, and we have assumed certain working ratios to be aimed at in the slag, in order to elucidate the method of solution.

CONSTITUENTS OF SLAG.

	From Ore and Fuel.	*From Flux.*	*Total.*
SiO^2	70.9	0.0314 x	70.9 + 0.0314 x
MgO	11.0	0.0032 x	11.0 + 0.0032 x
CaO	39.9	0.5374 x	39.9 + 0.5374 x
MgO	15.5	0.0017 x	15.5 + 0.0017 x
FeO	3.2	0.0016 x	3.2 + 0.0016 x
MnO	0.3	0.0023 x	0.3 + 0.0023 x
K^2O	3.4		3.4
CaS	0.4		0.4
	154.2	0.5753 x	154.2 + 0.5753 x

(1) To make a slag with 33.74 per cent of silica we must have

$$70.9 + 0.0314\,x = 0.3374\,(154.2 + 0.5753\,x)$$

whence $x = 116$ pounds.

(2) To make a slag with ratio of silica to bases 0.516 we must have

$$70.9 + 0.0314\,x = 0.516\,(82.9 + 0.5439\,x)$$

whence $x = 116$ pounds.

(3) To make a slag with ratio of silica to summated lime 0.478, we must first summate the lime as follows:

Lime $= 39.9 + 0.5374\,x$

Lime equiv. of $Al^2O^3 = \frac{168}{102}(11.6 + 0.0032\,x) = 19.1 + 0.0053\,x$

" " $MgO = \frac{56}{40}(15.5 + 0.0017\,x) = 21.7 + 0.0024\,x$

" " $FeO = \frac{56}{72}(3.2 + 0.0016\,x) = 2.5 + 0.0012\,x$

" $MnO = \frac{56}{71}(0.3) = 0.3$

" $K^2O = \frac{56}{94}(3.4) = 2.0$

Summated lime $= 92.5 + 0.5463\,x$

therefore

$$70.9 + 0.0314\,x = 0.478\,(92.5 + 0.5463\,x)$$

whence $x = 116$ pounds.

COMPARISON OF FUELS, FLUXES AND ORES.

By properly utilizing the preceding principles, it is possible to compare different varieties of fuels, fluxes or ores with each other, and thus to determine their relative values to the furnace, as far as can be inferred from their chemical composition. (Some of the following methods are from a paper by Mr. F. W. Gordon; Trans. Am. Institute Mining Eng., 1892, p. 61.)

COMPARISON OF FUELS.

If different qualities of fuel are available it is possible to calculate which is the most advantageous to use in the furnace. The fixed carbon only is efficient for the furnace, and not all of that, because the ash of the fuel needs to be fluxed to slag, and a certain amount of the fixed carbon will need to be burned simply to melt this slag. Then the cost of the limestone to be used to flux this ash must be counted in, and finally part of the labor-cost of running the furnace must be charged against the slag. We can thus calculate the total charge against the fuel to supply one part of available carbon, which is the best basis upon which to compare different fuels.

Example. Two varieties of coke are available for a blast furnace, analyzing respectively:

	No. 1	No. 2
Fixed carbon	84 Per Cent	88 Per Cent
Volatile matter	1.0 "	1.0 "
Moisture	0.5 "	0.5 "
Ash	14.5 "	10.5 "

And costing respectively $4.50 and $5.50 per ton. The ash of the fuels analyzes respectively:

	No. 1.	No. 2.
Silica	55 Per Cent.	25 Per Cent.
Alumina	25 "	5 "
Lime	15 "	50 "
Magnesia	5 "	10 "
Ferric oxide	.. "	10 "

They are to be fluxed with the limestone of preceding problem, assumed to cost $1.00 per ton, and to make a slag carrying 40 per cent of silica and alumina together. Assume an average of 0.228 parts of fixed carbon necessary to melt down 1 part of slag, and that the manufacturing costs borne by the slag amount to $1.00 per ton. What are the relative values of the two fuels to this furnace?

Solution: The amounts of flux needed to 100 parts of each fuel burned will be found as follows, letting x be the amount of flux used:

	Slag No. 1.	Slag No. 2.
Silica	$4.95 + 0.0314\,x$	$1.50 + 0.0314\,x$
Alumina	$2.25 + 0.0032\,x$	$0.30 - 0.0032\,x$
Lime	$1.35 + 0.5374\,x$	$3.00 - 0.5374\,x$
Magnesia	$0.45 + 0.0017\,x$	$0.60 + 0.0017\,x$
Ferrous oxide.	$0.0016\,x$	$0.54 + 0.0016\,x$
Total weights.	$9.00 + 0.5753\,x$	$5.94 + 0.5753\,x$

Therefore, in case No. 1:

$$7.20 + 0.0346\,x = 0.40\ (9.00 + 0.5753\,x)$$

whence $x = 18.4$

And in case No. 2:

$$1.80 + 0.0346\,x = 0.40\ (9.00 + 0.5753\,x)$$

whence $x = -9.2$

The negative value in case No. 2 simply means that the ash of fuel No. 2 is more basic than the slag and, therefore, requires no limestone, but itself acts as a basic flux. Per ton of fuel burned, there would be required respectively 0.184 and — 0.092 tons of limestone, and the weights of slag would be 0.196 tons and 0.0065 tons. (Substituting values of x in the total weights of slags.)

The weights of fixed carbon necessary to smelt these weights of slag would be:

No. 1. $0.196 \times 0.228 = 0.0447$ Tons.
No. 2. $0.0065 \times 0.228 = 0.0015$ "

Leaving as available fixed carbon for the furnace in each case:

No. 1. $0.84 - 0.0447 = 0.7953$ Tons.
No. 2. $0.90 - 0.0015 = 0.8985$ "

From these figures the cost of 1 ton of available carbon furnished by each fuel, adding in manufacturing cost chargeable against the slag, is

No. 1.

Cost of coke, $4.50 ÷ 0.7953	= $5.658
Cost of limestone, $1.00 × 0.184 ÷ 0.7953	= 0.231
Costs against slag, $1.00 × 0.196 ÷ 0.7953	= 0.246
	$6.135

No. 2.

Cost of coke, $5.50 ÷ 0.8985	= $6.233
Cost of limestone, $1.00 × (— 0.092) ÷ 0.8985	= 0.102
Cost against slag, $1.00 × 0.0065 ÷ 0.8985	= 0.007
	$6.138

The two fuels, at the prices given, are therefore of almost exactly the same value to the furnace.

The method along the lines shown is general, for any desired composition of slag, or for use with any given limestone. The means of comparing the values of different fuels, by the cost of a ton of pure, available carbon, when furnished by each fuel, is an item which is useful when comparing the relative values of different fluxes or ores with each other.

Comparison of Fluxes.

If different qualities of flux are available it is very desirable to be able to calculate which is the most economical to use in the furnace. Any acid ingredients in the flux diminish very sharply its efficient fluxing power, because they must first be satisfied from the bases present in the same proportions as acid to bases in the final slag. The slag thus formed from the impurities requires further to be melted, and other costs are properly chargeable against it. The best comparison is finally obtained by calculating for each flux available the cost from it of pure net lime, or net summated lime, analogous to the calculation for pure net carbon in the case of fuels. Any ordinary condition may be imposed upon the slag which the furnace is to produce.

Illustration: There are available for a furnace two qualities of limestone, containing respectively:

CaO	53.74 Per Cent.	47.80 Per Cent.
MgO	0.17 "	4.61 "
SiO^2	3.14 "	5.12 "
Al^2O	0.32 "	3.36 "
Fe^2O^3	0.18 "	1.10 "
CO^2	42.42 "	37.55 "

The first costs $1.00 per ton, the second $0.80. Assume them smelted with fuel furnishing pure available fixed carbon at $6.135 per ton; that 0.228 tons of pure fixed carbon is needed to smelt 1 ton of slag; that manufacturing costs against slag are $1.00 per ton, and that the slag to be made in the furnace must have summated silica equal to summated lime. Compare the relative values of the two fluxes.

Solution: We will direct our calculations towards finding the net cost of 1 ton of pure available summated lime from each of the two limestones. The summated lime and silica in each flux are:

	No. 1.	No. 2.
Summated lime	0.5411	0.5502
Summated silica	0.0342	0.0808
Excess of summated lime	0.5069	0.4694

The weights of slag formed from the impurities present in each limestone will be:

	No. 1.	No. 2.
CaO (difference between amount present and excess of summated lime found)	0.0305	0.0086
MgO	0.0017	0.0461
FeO	0.0016	0.0090
SiO^2	0.0314	0.0512
Al^2O^3	0.0032	0.0336
Totals	0.0684	0.1494

The cost of 1 ton of pure available lime from each of these fluxes will therefore be, adding in costs chargeable against the slags formed by the impurities present:

No. 1:

Cost of limestone, $1.00 ÷ 0.5069	= $1.973
Cost of carbon for melting slag, $6.135 × 0.228 × 0.0684 ÷ 0.5069	= 0.188
Costs of running, chargeable against slag, $1.00 × 0.0684 ÷ 0.5069	= 0.135
	$2.296

No. 2:

Cost of limestone, $0.80 ÷ 0.4694	= 1.904
Cost of carbon for melting slag, $6.135 × 0.228 × 0.1494 ÷ 0.4694	= 0.465
Cost of running, chargeable against slag, $1.00 × 0.1494 ÷ 0.4694	= 0.318
	$2.487

The conclusion is that the poorer limestone, at $0.20 per ton less cost, is in reality costing $0.191 per ton more for pure available lime, or is in reality 8.3 per cent dearer than the first, instead of being 20 per cent cheaper.

The method of calculation here described is quite general for any compositions of limestone or other flux, and for any assumed conditions which the slag must conform to.

Comparison of Ores.

As the more complicated, we come to the comparison of various ores which may be at the iron master's disposal. Here a similar method of procedure is advisable. It can be calculated first, for a unit weight of ore, how much pure lime would be required to flux its impurities, how much pure carbon would be required to melt the slag thus formed, and to the costs of each of these would be added the handling of the slag. Each of these can be also expressed per unit of pure oxide of iron in the ore; and if to their sum we add the cost of ore necessary to furnish unit weight of pure oxide of iron we obtain the total costs per unit weight of pure iron oxide (Fe^2O^3). This is the basis on which different ores may then be compared. It must not be forgotten that one ore may, because of higher sulphur content, require the production of a more basic slag, so that the amounts of flux required and slag formed will be influenced by the condition necessary to impose on the slag in each case.

Illustration: The iron ore briquettes (of Problem 51) contained Fe^2O^3, 85.93 per cent; FeO, 3.96; SiO^2, 5.50; MnO, 0.63; Al^2O^3, 0.76; CaO, 2.23; MgO, 0.97 per cent. If these cost $4.40 per ton, and are smelted in a furnace making slag with ratio of silica to bases 0.516, and assuming 0.3 per cent of the iron, 82.7 per cent of the silica, and 96.6 per cent of the manganese to go into the slag, what is the cost per ton of pure Fe^2O^3 from this source, charging pure lime for fluxing at $2.296 per ton, pure carbon for smelting slag at $6.135 per ton, and requiring 0.228 tons of carbon for one of slag, adding also manufacturing costs at $1.00 per ton of slag?

Solution: The slag-forming ingredients from 1 ton of ore briquettes are:

$$\text{FeO } 0.003 \times \frac{72}{56} \times \left[\left(0.8593 \times \frac{112}{160}\right) + \left(0.0396 \times \frac{56}{72}\right)\right] - 0.0008 = \text{Tons}$$

MnO 0.966×0.0063	= 0.0061	"
CaO	= 0.0223	"
MgO	= 0.0097	"
Al^2O^3	= 0.0076	"
SiO^2	= 0.0455	"
and the bases to satisfy the silica present must be $0.0455 \div 0.516$	= 0.0882	"
But, sum of bases already present	= 0.0465	"
therefore, pure lime to be added	= 0.0417	"
and total weight of slag	= 0.1337	"
Efficient Fe^2O^3 in 1 ton of ore	= 0.8592	"
plus Fe^2O^3 equivalent of reduced FeO $= \frac{160}{144} \times (0.0396 - 0.0008)$	= 0.0432	"
Total available Fe^2O^3	= 0.9025	"

Cost of 1 ton pure available Fe^2O^3 from these briquettes

Cost of ore, $4.40 ÷ 0.9025	= $4.875
Cost of pure lime, $2.296 × 0.0417 ÷ 0.9025	= 0.106
Cost of carbon for melting slag, $6.135 × 0.228 × 0.1337 ÷ 0.9025	= 0.207
Costs chargeable against slag, $1.00 × 0.1337 ÷ 0.9025	= 0.148
Total	= $5.330

A similar calculation is possible with any ore of any given composition, and making any assumed quality of slag. The costs of 1 ton of pure ferric oxide thus calculated will give the relative costs of the iron obtained from these different sources, and therefore indicate the relative values of the different ores to the blast furnace manager.

Problem 53.

Assume a blast furnace manager to use the ore of preceding illustration, furnishing pure Fe^2O^3 [illegible] per ton, and to use with it fuel furnishing pure [illegible] per ton, there being required for reduction and melting the iron produced and furnishing it with carbon, 0.66 tons of pure available carbon per ton of pig iron produced, and the pig iron containing 96.656 per cent of iron. The running costs of the furnace are $3.00 per ton of pig iron produced [illegible] against slag not included.

Required: The cost of the pig iron per ton.

Solution:

Fe^2O^3 required $100 ÷ 112 \times 0.9656$	= 1.3808	Tons
Cost of the ore, $5.336 × 1.3808	= $7.368	
Cost of fuel, $6.135 × 0.66	= 4.049	
Cost of manufacturing (share against pig iron)	= 2.000	
Total cost	= $13.417	

Storage Batteries.

A paper, recently presented by Dr. Rudolf Gahl before the Colorado Scientific Society, deals with the scientific principles of electric storage batteries and contains much that is interesting. The author discusses many points in a novel way, basing his discussion throughout on the principles of modern physical chemistry. Dr. Gahl is now a consulting chemist and engineer in Denver, Col., having been formerly connected with the two most prominent storage battery factories in this country and in Germany. He has carried out a large amount of research work in connection with storage batteries.

The first part of the paper is a general discussion of the possibilities of storing electrical energy in form of chemical energy. Dr. Gahl derives several axioms from practical experience. For instance, that a practical accumulator cannot be based on gas reactions at present; that an electric storage battery must contain only one liquid, and then, of course, no diaphragm; that soluble cathodes for storage cells are not reliable (which excludes the zinc-lead cell).

The discussion thus narrows down to cells containing insoluble cathodes and insoluble anodes. The electrolyte must not be decomposed by the current into gaseous products, it must have a high conductivity; it must be cheap. Among the acid electrolytes sulphuric acid stands alone. Since only barium and lead form insoluble salts with sulphuric acid and since barium has to be excluded, being not reducible by the electric current, the lead cell is the only practical acid cell. For alkaline cells potassium and sodium hydrate are suitable electrolytes, while iron and cadmium may be used for cathodes and the oxides of nickel, iron, silver for the anodes. For the support of the plates lead must be used in acid solutions, and nickel in alkaline solutions.

Dr. Gahl then gave a concise review of the present situation of the lead storage battery. He described the methods of Plante and of Brush-Faure.

"Plante plates are used either as positive plates or as negative plates. One advantage of using these Plante plates for the negative, that is, for the spongy lead plates, is that after a while, owing to reactions which have not been studied thoroughly enough, the spongy lead has its pores closed and solidifies into solid lead. Of course, there is a decrease in the capacity of these plates according to the change in their

structure. A remedy is available in such a case, which is to use the plates which have been negative before, as positive plates, the effect of a long series of discharges and charges on the positive Plante plates being just the opposite from its effect on the negative plates.

"The solid body of the positive plate is slowly but constantly attacked by the combined effect of sulphuric acid and current, and the capacity would increase considerably unless through the action of the gas bubbles developed on the charge of the battery some lead peroxide would be loosened and fall off. But, nevertheless, they are not apt to lose their capacity as quick as the negative plates, and on reversing the polarity and using the negative plates as positives and the positive ones as negative ones, the newly-made negative plates start in with quite a high capacity and the positive plates regain their former capacity through the attack to which positive plates are always exposed.

"The grids in American Plante plates are cut mechanically by machines. The leading Plante plates in Europe are cast plates, which do not seem to gain favor in this country, possibly because casting of such fine things as these battery plates requires a high development of the art of casting, to which art the younger American factories have not paid enough attention.

"Positive Faure-Brush plates are not made by any prominent plant in this country, except in small sizes for automobile work; in fact, it is very difficult to make them, but it is possible, for some good plates of this type are manufactured abroad. Negative Faure plates are made in this country by the National Battery Co., Buffalo, N. Y., and by the Electric Storage Battery Co., Philadelphia, Pa. The negative Brush plates formerly had the same disadvantage which the Plante plates still have, of losing their capacity after some time of service through shrinkage, but it has been found that by giving the paste used for manufacturing these plates a certain composition, that tendency for shrinkage can be overcome, and that it is even possible to manufacture plates which expand and become more and more porous after they have worked some time. The expansion, and accordingly the porosity of the plates, had to be kept down to prevent falling off of the active material.

"Recently another type has been brought on the market by the Tudor companies in Europe and by the Philadelphia Electric Storage Battery Co. in this country, which permits the use of material of high expanding power, the falling off of active mass being prevented by a kind of perforated lead wrapper, cast together with the grid. This plate is called box plate, and means a vast improvement over the negative plates made before, and it evidently seems to solve the problem of negative plates, for now the negative plates have a longer life than the positive plates, whereas, formerly the life of the battery was always limited by the negative plates."

For the choice of the best concentration of the acid several considerations are of importance. The higher the concentration, the higher the e. m. f. and the greater the amount of available energy per unit of weight. On the other hand, the condenser acid, or sulphuric acid has a maximum near 1.2 specific gravity, which is about the standard in practical use. Most important, however, is the regard for the life of the plate, since a variation in one direction is harmful to a negative plate and a variation in the other direction to a positive plate.

The capacity of the storage plate is limited, not by the absolute amount of the acid in the cell, but the quantity of acid in the immediate neighborhood of the working plate. It is most important to make the plates as porous as possible, so as to enable a difference of concentration between the battery where the acid becomes exhausted during discharge.

"The construction of the best plates which I mentioned before, is practically the result of investigations in the line, in a way that the qualities of lead peroxide do not permit these plates to be made as porous as would be desirable for increasing their capacity. As lead peroxide has a tendency to disintegrate in fine particles, it is not possible to make a porous positive plate without endangering its life very much. This is the reason why the construction of an efficient and at the same time durable positive plate presents so many difficulties; in fact, is the main difficulty at present in raising the life of the lead storage cell to the life of other electrical machines, as, for example, a dynamo or motor."

After a short explanation of the delecterious effect of the presence of iron and platinum in the electrolyte on the capacity of the cell, Dr. Gahl discussed very briefly the possibilities of the alkaline cell, in which field "Edison, the great American inventor, seems to have been at least partly successful."

Both cadmium and silver make very good plates for high capacity, although the e. m. f. of the combination is only about 1 volt. For the combination of iron with nickel the voltage is somewhat higher, but probably owing to the small conductivity of the oxides, it is not possible to construct plates of any thickness, even when the conductivity of the plates is increased by the addition of some good conductor like graphite or some metallic substance. The problem of making suitable iron and nickel electrodes is, therefore, chiefly a mechanical one.

"But the problem of a light alkaline cell will enter a new phase, when it will be commercially possible to use cadmium. It does not seem to be doubtful to me that as soon as large enough quantities of cadmium are discovered the problem of a light accumulator for automobiles will be brought very near its solution, at least for cases where the price does not cut so very large a figure as to exclude the application of metals like cadmium and silver.

"Perhaps it is left to Colorado to make this development possible by supplying sufficient amounts of cadmium."

Notes on Electrochemistry and Metallurgy in Great Britain

(From Our Special Correspondent.)

The Hermite Electrolytic Hypochlorite Plant at Poplar.

The first account which has appeared in an English journal of this installation, which has been installed by the Public Health Authority, is that in the *Lancet*, the celebrated medical journal. It is, therefore, not surprising that electrochemical details are not so full as might be desired. For instance, the specific gravity of the solution and the respective proportions of sodium chloride and magnesium chloride are not stated, neither is the current density mentioned. The electrolyzer, which is arranged on a cascade principle, apparently consists of forty eight cells, arranged in four tiers of twelve cells each. The solution passes through glass tubing from cell to cell, and from stage to stage, the strength increasing until, at the outflow, the solution contains about 5 grammes of available chlorine per liter. The output is 185 gallons per day, the voltage at the tank terminals is 230 volts, and the current 15 amps., from which data it may be calculated that the yield of available chlorine is 72 grammes per kw hour. Now, by no stretch of imagination (even of an inventor's elastic kind) can this be considered favorable on the side of electrochemical efficiency. In the March issue of Electrochemical and Metallurgical Industry are chronicled outputs at this chlorine strength exceeding 900 grammes of available chlorine per kw hour.

On the score of stability the resultant output is far superior to that chronicled by Sir Henry Roscoe in his investigation of the Old Worthing installation. Some tests in the *Lancet* laboratory showed a deterioration of 5 per cent in two weeks

for a solution containing 5.68 grammes of available chlorine per liter, and slightly higher losses for weaker solutions.

The Institution of Mining and Metallurgy.
Cyanide Process.

The papers read before this institution at its last meeting, March 15, included a "Note on the Ammonia-Copper Cyanide Process," by Mr. E. Le G. Brereton; a paper on the "Cyanide Treatment of Cupriferous Tailings by the Sulphuric Acid Process," by Mr. W. S. Brown. There was also a paper on "Earth Temperature on the Witwaters Rand Gold Fields and Their Relation to Deep Level Mining," by Mr. H. F. Marriott.

Mr. Brereton's paper is a recitation of some experiments, from which he concludes that "under certain conditions cuprous potassium cyanide in an ammoniacal solution is broken up with the liberation of an alkaline cyanide." Omitting the details of the actual experiments the following are the author's detailed conclusions:

"I. That there is always free cyanide (as indicated by $AgNO_3$) in the solution when the blue solutions are decolorized. II. The free cyanide indicated by $AgNO_3$ increases in the solution on standing. III. The removal of free cyanide by $AgNO_3$ tends to cause the solutions to regain the blue color, the precipitate of silver iodide dissolving, and a free cyanide again making its appearance. IV. That a large excess of ammonium hydrate in the solution is necessary if this reaction is to be carried to its limits. When an ammoniacal copper solution is titrated with a potassium cyanide solution, free cyanide is indicated by silver nitrate at the earliest stages of the titration.

"In the above experiments the change of the solution from colorless to blue, and the quantity of silver which has passed into solution, presumably as a double cyanide, is clear evidence of chemical changes taking place in the solution."

Mr. Brereton is of the opinion that in a strongly ammoniacal solution containing also an ammonium salt Cu_2Cy_2xKCy (assuming that such a compound is formed, which seems probable), is unstable unless an excess of free alkaline cyanide is present. If this excess of free alkaline cyanide is removed a reaction takes place of possibly somewhat the following order:

$$2(Cu_2Cy_2.2KCy) + 12NH_4OH + 8NH_4NO_3 + O = 4Cy_4NH_{12} + Cu(NO_3)_2 + 4KCy + 4NH_4Cy + 14H_2O$$

After an ammoniacal copper solution has been decolorized by a potassium cyanide solution it appears that chemical changes are going on in the solution for some considerable time (over four days) with the liberation of a free alkaline cyanide.

This may be explained by the breaking down of the cupric cyanides to cuprous; but to do so a more complex reaction is necessary than the one represented by the equation suggested by Mr. Sulman for the decolorization of an ammoniacal copper solution by potassium cyanide.

Mr. Brereton suggests that compounds of the x (NH_3) y $(CuCy_2)$ z (Cu_2Cy_2) class are first formed, which pass to the x (KCy) y $(CuCy_2)$ z (Cu_2Cy_2) class as the solution becomes colorless. Then if the latter class of colorless compounds takes some time to pass to its cuprous limit, namely, Cu_2Cy_2 $2KCN$ with the liberation of cyanogen, which in turn reacts with the ammonium hydrate, forming ammonium cyanide and cyanate, it accounts for the results observed.

Mr. Brown's paper is interesting on account of the practical information of the sulphuric acid process in comparison with the ammonia process described last year by Messrs. Jarman & Brereton.

The ore treated in which the copper is present as carbonate came principally from the Cobar Chesney Mine, Cobar, N. S. W., and was crushed in that company's old battery many years ago.

Separate acid vats were provided for the preliminary acid treatment. In the operations under review, these vats were shallow, rectangular wooden vats, each holding about 25 tons of tailings. The [illegible] vats held 75 tons of tailings, so that the contents of three acid vats were treated and subsequently loaded into one cyanide vat. On an acid vat being loaded, 10 to 12 tons of dilute H_2SO_4 [illegible] as the vat was full of solution, covering over the ore for some [illegible] the bottom valve was opened and the spent solution allowed to drain off slowly through a launder to the copper precipitation boxes. When the solution had drained [illegible] to show the ore on top of the vat, the bottom valve was again closed for an hour or so and the acid allowed to remain in contact. This first solution when drained off never showed free acid. On again opening the bottom valve, the balance of the acid solution was allowed to percolate, usually at the rate of about 2 tons an hour, and flowed directly through the copper precipitation boxes to the sump for making up [illegible] with H_2SO_4 or to waste.

The acid solution was followed immediately by a first-water wash, equal in tons to the original solution, which flowed into the vat from a tank above at the same rate as the vat was draining. This wash was followed by a clean water wash of about half the quantity, and the vat was allowed to drain for discharge, the total time of treatment being 48 hours. All the acid solution and wash passed through the copper precipitation boxes, and was either used over again or run to waste, depending on the supply of water available. The final wash was always clean water. When these operations were complete and the vat ready for discharge, 0.5 per cent to 0.8 per cent of lime was distributed over the surface and discharged with the sands, ensuring a fairly intimate mixture.

The copper was precipitated from solutions by passing through two boxes 10 feet x 3 feet x 4 feet, each divided into four compartments as in the ordinary zinc boxes. These boxes were filled with scrap sheet iron, obtained locally in the shape of old tins which had been burnt for recovery of the solder, and were to be had for the cost of carting. The precipitation of Cu on the large surface of iron thus exposed was practically perfect. On the other hand, all free acid entering the boxes was consumed at the expense of the iron.

Fifteen tons of weak KCy solution were first applied to displace the approximate 15 tons brought as moisture from the acid vats. This first 15 tons carried little gold. As a rule, about 12 tons would come through carrying only traces, and the other three tons would vary, according to the relative perfection of the displacement. In any case, the first 15 tons were carefully isolated in a special sump, and eventually used as a final wash for the outgoing residues. This solution passed slowly through a zinc box, and then through about 20 cubic feet of packed charcoal. The zinc precipitation was not effective unless free KCy was present, showing defective displacement in the vat, in which case there would be high gold values in the solution, but when no free KCy was present and but little gold, the charcoal always caught some, if not all. If assays showed that more than a few grammes of gold were still present, and precipitation had been imperfect, the solution was again passed through a freshly made-up zinc box, with addition of KCy, before being finally used up as a wash.

[illegible] the 150 tons treated during this period [illegible] ounces of gold are thoroughly shown as recovered. The general recovery over a period of some months was about 1 per cent to 1½ per cent in excess. The average value of the sands treated is shown as 1 dwt. 4 gr. to 1 dwt. 11 gr. [illegible] leaving 1 dwt. 17 gr. in the residues, equal to 40 per cent to 45 per cent extraction. The average theoretical strength of acid required per ton is shown as 40 lbs. per cent H_2SO_4, [illegible] 9 lbs. per ton acid. The theoretical quantity of acid called for is 0.888 pounds [illegible] pounds per ton treated, or 25 pounds excess of the actual amount required for the ores treated. The quantity consumed was [illegible] pounds of 60 per cent H_2SO_4, equal to 30 pounds per ton treated, or 25 pounds excess of the actual amount required for the ores treated. The copper precipitated through the zinc boxes [illegible]

equal to 1.44 tons for 450 tons, 7.1 pounds per ton. Cyanide consumption is shown as .072 per cent, or 1.6 pound per ton.

From costs taken over several months:

	Per Ton.
Scrap iron averaged	1.5d
Lime averaged	3.6d
Zinc averaged	2.0d

The quantity of acid used—4.5 pounds per ton in excess of consumption on the ore—is open to criticism, and could to a great extent have been obviated by a better arrangement. Instead of passing the acid solution direct to the copper precipitation boxes, where the excess of free acid was consumed by the iron, it might have been sent to a sump and restrengthened for a second application, or used as a preliminary wash on the next vat, and only passed through the precipitation boxes when showing no free acid. Under the circumstances in which the work was performed it was considered economical to sacrifice acid and iron for other considerations.

The cement copper recovered was readily disposed of, although, perhaps, at a low price, since the total tonnage was small. Shipments averaged about 60 per cent Cu, the balance being iron and silica.

No serious difficulty was found in getting a reasonable extraction of the gold from the acid-treated sands. The first experiment showed a possible extraction of only 65 per cent, with six days' treatment, but this was found to be due to lack of aeration. While experimenting with various oxidizers it was discovered that sufficient aeration could be obtained by applying the solutions as before described. In practice the solutions were well circulated in the sumps, and the usual provision made of allowing the pump suction to draw a little air.

The working solutions seldom showed over 0.3 per cent Cu, and usually were much lower. When, as latterly, more slimes had to be treated in the charges, conditions were not so favorable, and more Cu had to be dealt with in the cyanide treatment. The zinc boxes then required careful attention, and the endeavor was to get as clean a precipitation of copper, as well as of gold, as was possible. Lead acetate was freely used, and the boxes freshly made up at close intervals. No cyanide solutions were at any time discarded. When the solutions were foul they were always high in gold contents, and as the precipitation of the gold was accompanied by precipitation of the copper, re-aeration restored their working usefulness.

The Capitalistic Side of British Electrochemistry and Electrometallurgy.

Last April the present writer reviewed the capital invested in, and interest on, the British electrochemical industries from the unclassified information thereon contained in the indispensable (in this direction) *Garcke's Manual of Electrical Undertakings*. Dividing these into the same four groups as last year the changes are as follow:

(*a*) Makers of electrolytic alkali and bleaching powder—The share and debenture capital of the Castner-Kellner Alkali Co. now stand at £675,000, some £25,000 of debenture having been redeemed. A dividend of 4 per cent was again paid last year. The Electrolytic Alkali Co., whose share capital amounts to £302,000, and debenture capital at £20,850, again paid no dividends on their ordinary shares, but owing to an improvement in trade conditions were able to pay part of the arrears of dividends on their preference shares.

(*b*) Makers of primary and secondary batteries—Taking only these firms which devote their sole attention to this matter, the aggregate capital is about £622,800, the number of firms being sixteen. The Chloride Electrical Storage Co. again paid 8 per cent on their ordinary share capital; also a debenture debt of £20,000 has since been redeemed. The Electrical Power Storage Co. also paid the same dividend—6 per cent—on their ordinary shares as last year. The D. P. Battery Co. again paid 20 per cent on their small ordinary share capital of £10,000. The Hart Accumulator Co. paid 12½ per cent on their £40,000 of ordinary share capital, 6 per cent on their preference capital of £3,000.

(*c*) Electrometallurgical undertakings—The aggregate capital invested in these is £2,117,000. A decline due to the British Aluminium Co. having written off £90,000 of share capital is balanced by increases in other directions. In regard to this section it is rather difficult to define the difference between it and section (*d*), which embraces a number of smaller firms, the work of some of which is partly metallurgical.

(*d*) Miscellaneous undertakings number eleven, and have an aggregate capital of £444,000, there being no notable change since last year.

Market Quotations During March.

The chief feature of the month has been the high prices reached by both tin and copper. The former fell from £165 to £162 between the first and fifth, and then rose to £169 by the thirtieth, and has since gone higher. Copper, which opened at £78.5, has twice touched £84, and closed only a fraction easier at £83.15.

In regard to iron, there have been no marked fluctuations, Cleveland warrants having closed at £2.8.7½. Lead has been easier, and has closed at £16 for English ingot. Zinc sheet is fetching £28.15 per ton. Platinum is quoted at £4 11.6 per ounce, at which price dealings are very much restricted.

As to chemicals, nothing calls for comment save that copper sulphate has advanced to £25 per ton. There is a marked all-round activity, which is somewhat hampered by the high price of copper.

London, April 7.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

Electrochemistry.

Calcium Cyanamide. The April issue of *Harper's Monthly Magazine* contains a well written and interesting summary, by R. S. Duncan, of the work which has been done in recent years towards the fixation of atmospheric nitrogen for the production of fertilizers. The whole problem has been discussed repeatedly in our columns and at length in our last issue. Prof. Duncan's article contains, however, several new incidental notes. A new use of cyanamide is mentioned, namely, its employment for the case-hardening of steel, and it is stated Loewe & Co. continually use large quantities of cyanamide in the manufacture of tools and of arms for the German Government. By the action of acids upon calcium cyanamide one obtains dicyandiamide, which is now made by the ton, "and much of it is sold to the dye industries for a purpose that cannot be imagined by the manufacturers. Still other quantities are sold to manufacturers of explosives, owing to the fact that when mixed with other substances it lowers the temperature in the gun-barrel." By reaction of cyanamide with water artificial urea is produced, tons of which have been

sold to manufacturers of pharmaceutical preparations. Finally, sarcosin unites with cyanamide to yield creatine "one of the actual substances of human muscle found in the extract of meat. The author makes the following prophecy: "We may look forward with a very reasonable assurance to the creation of as many factories for the fixation of elemental nitrogen as we have smelting furnaces for unfixing elemental iron."

Copper Refining.—In the Stockholm *Teknisk Tidskrift*, B. Carlson, who visited this country somewhat over a year ago, discusses the electrolytic refining of copper in the United States. The description of the various refineries contains nothing new, but the following notes contain some information which, although well known to refiners, does not appear to have been published before. An English translation of the whole article may be found in *The Mining Journal* (London) of Jan. 20:

Of the impurities of the anode copper, gold and platinum pass entirely over in the slime; only a small quantity of silver (7-10 grams per 1,000 kilograms) pass over to the cathodes, when the composition of the electrolyte remains normal. Considerably larger quantities of silver are found in the cathodes when the acidity of the electrolyte diminishes.

To guard against the loss of silver, a small quantity of hydrochloric acid (0.003 to 0.006 per cent of Cl), is added to the electrolyte when the water employed is not already sufficiently acid. If the addition of hydrochloric acid is too great, a danger of short circuiting arises, owing to the cathode copper depositing irregularly and forming needles on the cathode. The chlorine is recovered from the slime as chloride of silver. The amount of arsenic in cathode copper is not allowed to exceed 0.001 to 0.002 per cent. In the anodes arsenic occurs partly as metal and partly as copper arsenite, which remains insoluble in the anode slime, and only for a small part is made soluble by secondary processes. Metallic arsenic is dissolved as arsenic acid, and is not deposited on the cathodes as long as the solution is sufficiently acid and not saturated. Experiments have been made to obtain refined copper, absolutely free from arsenic, by adding up to 20 per cent sulphate of ammonia. A frequent renewal of the electrolyte is, however, more practical. The above mentioned slight addition of hydrochloric acid, in presence of arsenic and also of antimony, advantageously hinders the tendency of these metals to deposit on the cathodes at higher densities.

The small addition of hydrochloric acid is one of the most important advances made in the electrochemical refining of copper; by this the now usual employment of high-current densities is rendered possible. The antimony passes completely in the solution, but is not deposited so long as the electrolyte remains normal. Selenium and tellurium, as far as is known, exercise no influence worth noting on the cathode copper, and lead passes entirely with the anode slime. Nickel remains in the form of sulphate in the solution. When the acidity of the electrolyte is too low, or when the current density is too small, the cathode copper is often so brittle that it can be pulverized. This particularity is caused by the high percentage of protoxide of copper in the deposited copper. The impure electrolyte removed from the baths is utilized for copper sulphate. Its free acid is neutralized by treating with copper granules and air, whereby the greater part of the ferric sulphate is deposited as ferrous sulphate. The solution is repeatedly boiled until, on cooling, copper sulphate crystallizes from the remaining liquor. Copper is separated with iron, and the liquid is run off. At chrome the electrolyte is only partly neutralized, and during crystallization receives a strong sulphuric acid solution. This is again heated, and sulphate of iron, nickel and copper are precipitated. The warm acids are filtered and allowed to cool, and arsenic acid crystallizes in large quantities. At the same time antimony antimonic acid are precipitated. The acid is used for treating the anode slime.

As is well known, the electrolyte must be kept in circulation, otherwise copper and sulphuric acid would rise at the anode and the impurities are deposited on the cathodes. The circulation is obtained by allowing the electrolyte to flow from an elevated vat, in which it is heated to nearly 50° by the heat which is placed over the other, and is then pumped back. Blowing air into the electrolyte at definite intervals aids the deposit of impurities, and especially [illegible] bottom of the bath in colloidal solutions. Otherwise these suspended impurities deposit mechanically at the cathodes. The anode slimes consist of [illegible] and partly oxide, arsenic and sulphur compounds, 5-15 per cent silver, and 0.01-0.7 per cent gold; to this must be added small proportions of iron, bismuth, lead, antimony and arsenic compounds. As a rule the slime is treated in a phosphor bronze centrifugal, where the larger grains of copper, which represent about 10 per cent of the slime, are separated and then taken to the anode furnaces. The mother liquor is led back to the electrolytic department, and the slime is mixed with 95 per cent sulphuric acid at 66° and heated by steam, during 12-14 hours in lead lined vats. By this process two-thirds of the arsenic, all the iron and bismuth, and a large part of the copper are dissolved. After settlement of the residues the solution is pumped to the vitriol works. The slime is boiled with water; this causes the precipitation of the copper and silver present from the sulphate of silver in the solution. If the slime does not contain sufficient copper, small quantities of raw slime are added.

The remaining slime, rich in precious metals, is first decanted and then filter-pressed, and dried in cast-iron vessels over a furnace. Silver containg 1-2 per cent gold and about 2-3 per cent copper is recovered by smelting in reverberatory or muffle furnaces with sand and soda. The slag, containing 1 per cent silver and 8-10 per cent copper, goes back to the raw copper furnace. The silver is refined, either by the electrolytic process, or treated in the ordinary way with concentrated sulphuric acid, and worked up to silver 0.998-0.9985 purity and gold 0.933-0.996 purity.

Electrodeposition of Copper Upon Iron.—In the January issue of the *Journal of Physical Chemistry*, O. W. Brown and F. C. Mathers give an account of a long series of experiments which lead them to recommend the following bath: Sixty grams copper sulphate, 50 grams sodium hydroxide, 159 grams sodium potassium tartrate, 1,000 cc. water. The best working conditions for this bath are a current density at the cathode of 0.1 to 0.5 amps. per square decimeter and a maximum density per square decimeter at the anode of not over 1.04 amps. If at any time a green precipitate should begin to form on the anode, a little caustic soda—3 or 4 grams per liter—should be added. The substitution of potassium hydroxide for sodium hydroxide and the warming of the bath were found to be detrimental to its working. Rolled, electrolytic and cast copper anodes work equally well. In all quantitative experiments of the authors, the anode corrosion had the theoretical ampere-hour efficiency if the anode discharge potential had a value of 0.3 volt or less, but whenever the value changed to [illegible] volt or greater, the anode corrosion became very poor. The tartrate baths are superior to the cyanide baths, in that they are non-poisonous, do not give off an offensive gas during electrolysis, will work at a higher anode density, give as good cathode deposits, require a lower voltage and yield a current efficiency of 100 per cent at both electrodes. The bath requires the addition of small amounts of caustic soda from time to time. However, it is not inferior to the cyanide bath in this respect, as the latter requires the addition of potassium cyanide.

Regulating the Voltage at the Terminals of Electric Furnaces. The April issue of the *Electric Journal* contains an interesting illustrated article, by H. R. Stuart, on potential regulation for large electric furnaces. The article discusses in

detail the design of regulators made by the Westinghouse Co., which were already described in an article of FitzGerald in our Vol. III., p. 9, and following.

Distillation of Metals.—H. Moissan has recently made some interesting experiments which are described in the *Comptes Rendus* of March 19, and abstracted in the London *Electrician*, April 6. These experiments relate to the liquefaction and distillation of metals. The iron family require considerable energy in the electric furnace, but even tungsten and molybdenum may be regularly distilled. Boron and carbon, on the other hand, pass direct from the solid to the gaseous state. Brought to the temperature at which carbon evaporates, titanium is liquefied. The author has distilled 17 grammes of titanium in 6 minutes. A difficulty lies in the avidity with which titanium combines with nitrogen. The author draws some conclusions with regard to the temperature of the sun, which is known to contain titanium. "On account of the large amount of heat radiated by the sun, it appears probable that it contains some solid matter as well as gaseous matter. At atmospheric pressure, no solid matter can exist at temperatures superior to that of the electric arc; *i. e.*, 3,500°. Under the pressures available within the body of the sun there will be much solidification, so that bodies can remain solid at temperatures higher than that of the arc. It follows that the temperature may be higher, and that at the surface of the sun it may amount to the 6,590° given by Wilson."

Blast Furnace Gas for Electrolytic Work.—In the London *Electrical Review*, of March 23, W. H. Booth refers to the fact that the large electrochemical industries have so far looked for cheap water-power, and that for this reason Niagara Falls and the upper Savoie have become centers of electrochemical activity. The author points out—what has been often emphasized in our columns—that for certain electrochemical and electrometallurgical work the blast furnace, running as it does 24 hours per day, may become a valuable power producer, the blast furnace gases, of course, to be used in gas engines coupled to electric generators. The author remarks that a British aluminium company hopes soon to have 30,000 hp at work, an amount of power that would be obtained from less than 2 per cent of the blast furnaces of Great Britain.

Metallurgy.

Iron and Steel.

Properties of Cast Iron.—Dr. Borcher's *Metallurgie*, having now extended its field to cover "die gesamte Hüttenkunde," and Dr. F. Wüst, of Aachen, having taken over the assistant editorship of the journal, with special oversight of iron and steel, the first number under the new regime, Jan. 8, contains very appropriately a long communication from Dr. Wüst himself, "mit 3, Abbildungen, 8 Kurven und 4 Tafeln." There are really some good points brought out in the article. Dr Wust investigated twenty varieties of cast iron, of each of which careful chemical analyses were made for graphite, carbide carbon, hardening carbon, silicon, manganese, phosphorus and sulphur, in almost every case the iron and carbon amounted to 99.0 per cent, the carbon varying from 2.56 to 4.82 per cent. With these alloys of iron and carbon very carefully made cooling curves were obtained. They showed, in general, melting points from 1,112 to 1,149 C., independently of the per cent of carbon, between 2.91 and 4.76. The conclusion is drawn that with 3 per cent and over of carbon a minimum melting point of about 1,130 is reached. Between 1,130 and 700 the cooling curve falls regularly, but at 700 a sudden evolution of heat takes places, corresponding to the temperature at which iron carbide is insoluble in iron, the whole mass settling into an intimate mixture of cementite and ferrite crystals. The change is visible in both gray and white iron, in amount corresponding to the quantities of perlite contained in each. The change takes place between 692° and 719°, but no law connecting these irregularities with the composition was discovered.

Phosphorus and Carbon.—Fettiveiss, in the metallurgical laboratory at Aachen (*Metallurgie*, Jan. 22) has reinvestigated the influence of phosphorus on the dissolving power of iron for carbon, in order to control Stead's researches of 1900. He improved on Stead by taking a pure highly carburized iron and adding to it ferrophosphorus, thus eliminating excess of insoluble carbon, and by analyzing the iron for phosphorus after the tests; whereas Stead added phosphorus to pure iron and then tried to saturate it with carbon, and did not control the test by subsequent analysis. The results are not far from Stead's, showing a steady decrease of 0.5 per cent of carbon for each 2 per cent of phosphorus present, and confirming the supposition that the phosphorus forms Fe^3P, containing 15.8 per cent phosphorus, and that since this can hold only about 0.25 per cent of carbon, the total carbon present must steadily decrease to nearly zero as the phosphorus approaches 15.8 per cent.

Manganese Alloys.—Levin and Tamman (*Zeitschrift für Anorganische Chemie*, 47, 136—from *Metallurgie*, Jan. 22) have made a much needed investigation of the melting points of iron-manganese alloys. Using soft steel with 0.26 per cent impurities, melting at 1,550°, and 99.4 per cent manganese, melting at 1,245°, the melting points were:

10	per cent	manganese	1,525°
20	"	"	1,483°
30	"	"	1,442°
40	"	"	1,380°
50	"	"	1,340°
60	"	"	1,288°
70	"	"	1,275
80	"	"	1,260
90	"	"	1,250°

An inspection of the curve shows that the melting point of the steel is reduced pretty regularly 4 .4 for each per cent of manganese present, until 60 per cent is reached, and thereafter close to 1° for each additional per cent up to pure manganese. It must not be forgotten that these are the carbonless alloys.

Deoxidation of Calcium.—C. Quasebart, of Aachen, relates, in *Metallurgie* for Jan. 8, his fruitless endeavors to alloy calcium with iron. He melted soft steel in a plumbago crucible, tied a piece of calcium to a soft-iron rod, with iron wire, and dipped it into the melted steel; the carbon in the steel fell 0.37 per cent, but no calcium was left in it. Throwing pieces of calcium into an ingot mould, while pouring 0.87 per cent carbon steel into it, likewise produced no alloy. A piece of calcium was bored out, a round iron cylinder tightly pressed in, and the two heated together 60 hours at 750°; not a trace of calcium diffused into the iron. A soft iron cylinder, filled with pieces of calcium was heated 3½ hours to 1,400°, but no calcium alloyed with the iron. A similar cartridge was put into melted cast iron with equally negative results. The investigator concludes that he has proved the universal negative, concerning the ability of iron and calcium to alloy with each other.

Lead.

Alloys with Arsenic.—Prof. K. Friedrich, of the old *Freiberger Bergakademie*, is continuing his good work on the metallic alloys. Jan. 22 number of *Metallurgie* contains a ten-page description of the combinations of lead with arsenic in increasing amounts up to 34 per cent, with nine interesting photomicrographs. Pure lead, melting at 327°, has its melting point reduced by arsenic until the 2.5 per cent arsenic alloy melts at 292°, forming a well marked eutectic, which appears also in all the other alloys investigated. Above 3 per cent arsenic the melting point rises regularly, becoming 563 at 34.4 per cent, with no intermediate indication whatever of any compounds such as Pb^5As, Pb^4As^3, Pb^3As^2, $PbAs$, or Pb^2As^3, which have been spoken of by Descamps. The latter must have had only mixtures. Alloys of lead and arsenic may exist above 34 per cent arsenic, but as far as can be predicted it is not likely.

COPPER.

Pyritic Smelting.—A paper by R. C. Alabaster and F. H. Wintle, before the Institute of Mining and Metallurgy, gives an account of the practice of pyritic smelting at the Tennessee Copper Co. and at the Ducktown Sulphur, Copper & Iron Co., at Ducktown, Tenn. The two-stage process is employed, namely, the ore is first concentrated to a matte giving 10 per cent of copper, and this matte is resmelted in another furnace to a matte containing about 45 per cent of copper. The enriched matte is bessemerized. In the concentration furnace 280 to 300 short tons of the 10 per cent matte produced in the first furnace are smelted every 24 hours. The Ducktown Sulphur, Copper & Iron Co. also employs the two-stage process. In the first stage there is made a matte containing 20 per cent of copper, the slag running 0.37 per cent in copper and containing 32 per cent of silica, 38 per cent iron and 8 per cent lime. The matte is concentrated to 50 per cent copper, the slag containing 0.6 per cent copper. The coke consumption is 8 per cent of the matte.

The authors think that for the treatment of an ore of the nature of the Ducktown ore the two-stage process is the most advantageous, and they recommend three or more rectangular water-jacketed blast furnaces, approximately 200 to 240 inches long, and not more than 44 inches or less than 40 inches wide. Wider furnaces have a tendency to run cold and form crusts. Good results would be obtained by using 20,000 cubic feet of free air at 30 to 35-ounce pressure with an ore column 12 feet high. The first concentration should produce a matte carrying 10 per cent copper and a slag with not more than 0.25 per cent copper. It is necessary to practice low concentration in this stage so as to keep down the slag losses. The matte should be treated in a second furnace with the necessary amount of silica for fluxing off the iron. The slags, containing about 0.6 to 1 per cent of copper, should be retreated in the first furnace. The composition of the slag to be formed depends upon special conditions, but the aim should be to produce an ideal slag of a mixture of bisilicate of lime and singulosilicate of iron in equal parts. Alumina is the only substance which interferes with pyritic smelting, since it forms a most infusible silicate and necessitates a higher consumption of fuel. Zinc, if its percentage is below 6 per cent in the ore, does not interfere with the process. The authors think that hot blast is unnecessary when the sulphur in the ore exceeds 20 per cent. Where the sulphur is below that percentage and coke is expensive, while other fuels, such as wood, oil, gas, etc., are comparatively cheap, hot blast may probably be employed with advantage. For all ordinary propositions, however, the authors unhesitatingly recommend the use of cold blast.

GOLD.

Fine Grinding and Cyanide Process.—A paper by F. C. Brown, published in the January issue of the *Bi-Monthly Bulletin* of the American Institute of Mining Engineers, gives a review of the cyanide treatment of ores in the Ohinemuri district of New Zealand. According to the author, extremely fine grinding (so that about 90 per cent will pass a 200-mesh screen) will become a very important feature in this district in the near future. A review of present practice is given The author shows the following advantages arising from the system of very fine grinding in tube mills when compared with coarser crushing. First'y, the dissolving of the gold and silver in the ore takes less time; secondly, weaker cyanide solutions can be used, and, thirdly, a better extraction can be obtained. From many experiments in treating concentrates by the cyanide process the author has found that the following points are of great importance: The concentrates must not be allowed to become decomposed by being left exposed to the air. Exceedingly fine grinding is necessary in most cases, and usually the concentrates must be reduced to slimes. Agitation must provide for a good supply of air Weak cyanide solutions will give a good extraction, provided the grinding is fine enough. Since all these requirements would be obtained by treating the mineral particles; *i. e.*, the concentrates, together with the sand, it would appear as if the special operation of concentration could be dispensed with in some cases, which would result in a great saving in cost of plant and operating expenses.

ALLOYS.

Aluminium-Magnesium.—Louguinine and Schukareff publish in *Revue de Metallurgie for* January, a very carefully prepared article on the heat of combination of these two metals to form alloys. They tested alloys corresponding to the formulæ, $AlMg$, Al^2Mg, Al^4Mg, $AlMg^2$, $AlMg^3$, by dissolving them in hydrochloric acid in a calorimeter, and comparing the heat evolved with that of the solution of equal quantities of aluminium and magnesium unalloyed. In no case could they detect any differences, showing that the two metals alloy without evolution or absorption of heat. The two data which remain as the useful result of their work, are the heats of solution of aluminium and magnesium in $HCl.10H^2O$. These are 4,612 calories per gram of aluminium, and 4,510.9 calories per gram of magnesium, corresponding to 124,524 per atom of Al. and 108,262 per atom of Mg., which represent 41,508 calories and 54,131 calories per chemical equivalent of the respective metals.

Analysis of Current Electrochemical Patents

ELECTRIC FURNACES.

Pressure Furnace.—H. N. Potter, 814,727. March 13. Application filed July 23, 1903.

Dr. Potter, who has done a great deal in the past in working out the design of electric tube furnaces, patents here a tube furnace for use in such processes in which gases or vapors are to be heated to a high temperature and to be subjected simultaneously to a high pressure. The ends of the heating tube are clamped between terminals of carbon which are connected to the outside cylindrical wall of the furnace. This is made in two tubular parts, separated by an insulating gasket. The space between the outside wall and the inner heating tube is filled with an insulating refractory packing material. The current passes from one half of the outside wall, which forms one pole, to the carbon terminal at that end, then through the inner tube to the carbon terminal at the other end, and leaves the furnace through the second half of the outside wall. For details of the design, which is very carefully worked out, the reader must be referred to the patent specification.

Vacuum Furnace. H. N. Potter, 814,726. March 13. Application filed July 23, 1903.

The purpose of this furnace is just opposite to that of the type described in the preceding abstract. It is designed for highly heating refractory materials under reduced pressure, or for executing certain high-temperature chemical reactions, where the speed of the reaction or its efficiency is improved by reduced pressure. The material to be treated is packed between the water-jacketed outside walls of the furnace, and a tube in the center of the furnace and the air between the particles to be heated is exhausted by means of a pump through the central tube or otherwise. This furnace, like the one in the preceding abstract, is of highly finished mechanical design, and many details of the construction are [illegible]

[illegible]

Iron from Ore. R. H. Aiken, St. [illegible] Mo. Application filed June [illegible] 1903.

Pulverized [illegible] ore [illegible] both of [illegible] with an [illegible] [illegible]

bath is electrolyzed, the precipitated iron serving as cathode. As much as 20 per cent of Fe_2O_4 can be dissolved in a basic silicate of proper constitution. The process is continuous, and the cell may be made large.

Aromatic Alcohols and Derivatives.—C. Mettler, 815,548, March 20. Application filed Nov. 2, 1904.

Aromatic alcohols and their ethers are produced by "exposing the aromatic esters in a dissolving agent, capable of conducting the electric current in the cathode space of an electrolytic apparatus, using for the cathode a material of specially high cathodic tension, then neutralizing the cathode liquid, then separating the reaction product and finally distilling." The following is an example: The cathode space of a cell is charged with a solution containing 300 grams of ethyl ester of benzoic acid, 350 grams of concentrated sulphuric acid, 500 grams of alcohol of 96 per cent, 100 grams of water. (In place of water a correspondingly dilute solution of sulphuric acid can, of course, be employed.) The anode space, separated by an earthenware cell, contains sulphuric acid diluted with water. It is known that the electrolytic reduction of substances, which are not readily and easily reduced, can be facilitated by employing for the cathode a metal having a high cathodic tension. Therefore, the cathode is made of pure sheet-lead, which is suitably prepared according to Tafel's prescription (*Ber. d. Deutschen Chem. Ges.*, 33; 2,215), and pure lead is employed for the anode. A current of 7 amps. per 100 square centimeters cathode-surface is sent through the cell; but the strength of the current can be materially varied without disadvantage. By means of cooling, the temperature is kept between 20° and 30° C. When no more hydrogen is absorbed, the cathode liquid is neutralized with about 30 per cent soda-lye. Two layers are formed, of which the upper one consists of alcohol and the products of reduction of the ethyl ester of benzoic acid. This is separated, distilled and fractionized. The benzyl-ethyl ether boils at 184°, the benzyl alcohol at 205°.

Aromatic Alcohols.—C. Mettler, 815,193, March 13. Application filed Oct. 11, 1905. (Assigned to Badische Anilin und Soda Fabrik.)

With reference to the process, described in the preceding abstract, the inventor states that the free carboxylic acids themselves can also be reduced in an analogous manner, giving rise solely to the alcohols. It is necessary to employ a cathode of a metal of high cathodic tension like lead. The course of the reaction is illustrated by the equation $C_6H_5.COOH + 4H = C_6H_5.CH_2OH + H_2O$. One example given is as follows: Introduce into the cathode compartment of an electrolytic apparatus a solution of 200 parts of benzoic acid in 400 parts of concentrated sulphuric acid and 1,400 parts of alcohol. The cathode consists of sheet-lead, prepared according to Tafel. The anode consists of pure lead, and the anode compartment is filled with dilute sulphuric acid. Send through the liquid an electric current of from 6 to 12 amps. per 100 square centimeters of cathode surface and cool by means of water, so that the temperature remains between 20° and 40° C. As soon as no more hydrogen is absorbed neutralize the cathode liquid with caustic soda and extract the alcohol, which is formed by means of ether. The acid of benzyl alcohol boiling at a temperature of 201° centigrade is almost quantitative.

Production of Ammonia.—J. V. Lloyd and E. C. Broadwell, 815,928, April 3. Application filed Sept. 3, 1904.

A crucible forming the cathode contains a fused bath of a borate of an electropositive metal (potassium, sodium, manganese, chromium, molybdenum, tungsten, vanadium, etc.) or a mixture of such borates with a carbon rod as anode in the center. On account of the considerable small surface of the anode rod, the anode current density is high and an intense heat is created at the carbon anode. The anode is surrounded by a pipe suspended into the fused bath, and through this pipe nitrogen or a nitrogen-bearing gas is passed into the bath. "The electrolysis produces anion, boric anhydride and oxygen at the anode, where the intensely-heated carbon acts as a reducing agent to chemically reduce the anion to boron. The pipe serves to prevent the boron thus formed from floating over to the cathode, and the nitrogen introduced into the pipe combines with the boron to form boron nitride. The metal of the borates will be deposited at the cathode." After sufficient boron nitride has accumulated it is subjected, while separated from the metals, to steam at a temperature of 600° C., or above, when the reaction takes place $2BN + 3H_2O = B_2O_3 + 2NH_3$. The temperature of the bath is maintained at about 1,000° C.

Purifying Water.—H. C. Bailey, 814,764, March 13. Application filed May 23, 1901.

A casing with insulating cover contains a plurality of spirally-arranged metallic strips or plates between the bottom and the cover. The inner terminals of these spiral strips are arranged near the center of the receptacle, while the outer ends extend to the sides of the receptacle. Channels are thus formed between the spiral strips. Water is passed through these channels in a rotary movement toward the center, where it enters a secondary channel in the reverse direction, flowing toward the periphery of the receptacle to an outlet opening. On the passage through the channels the water is electrolyzed, one side of the channel forming the anode, the other the cathode.

Electroplating Apparatus.—Guy L. Meaker, 815,027, March 13. Application filed Dec. 23, 1904.

The apparatus is intended for plating small articles. These articles rest on a horizontal tray with an open, foraminous or reticulated metallic bottom, which forms the cathode. Immersed within the electrolyte above the tray are the anodes, consisting, of course, of the metal to be deposited. The tray is agitated and vibrated in such a manner that the articles on it are constantly shifted in one direction and simultaneously turned so as to be coated on all sides.

Electrolysis of Water.—W. F. M. McCarty, 814,155, March 6. Application filed Nov. 8, 1904.

Apparatus for carrying out the process mentioned on page 113 of our March issue. The electrolytic cell is divided into two compartments by means of a solid diaphragm, which is perforated, short glass tubes being inserted in each perforation.

Batteries.

Storage Battery.—F. Monterde, 814,064, March 6. Application filed July 24, 1905.

A cylindrical lead vessel forms one pole of the battery. Within it are provided a number of concentric open-ended cylindrical electrodes provided on both sides with Y-shaped pockets to contain the active material.

Battery.—E. W. Schneider, 810,384, March 27. Application filed Aug. 11, 1905.

In a battery of several dry cells one cell may be thrown into or out of the generating circuit of the other cells by means of an end-to-end reversal. The object is to use this special cell only when the others have become nearly exhausted. By being connected in series with the others it then serves to boost the voltage.

Storage Battery.—H. B. Hallock, 814,691, March 13. Application filed May 15, 1905.

Details of construction of a storage battery with two electrodes in horizontal position one above the other and separated from each other by means of a porous separator. The lower electrode is the positive pole element, and its grid fits upon the inside surface of the inclined sides at the bottom of the jar.

Active Material for Storage Batteries.—E. L. Oppermann, 815,028, March 20. Application filed April 11, 1905.

In order to produce an exceedingly porous and at the same time hard and tenacious active material, the usual active oxide is mixed with animal hair or wool, ground or powdered so that the length of any individual hair is not more than one-sixteenth of an inch. The hair should form from ½ to 1½ per cent of the mixture.

RECENT METALLURGICAL PATENTS.

Treatment of Sulphide Ores with Chlorine.—C. E. Baker and A. W. Burwell (816,061, March 27) patent a process of treating sulphide ores which is analogous to the first step in the Swinburne-Ashcroft process (our Vol. I., p. 412). The crushed, dry sulphide ore is charged into a pebble mill "and subjected therein with or without the application of heat to chlorine, in quantity sufficient to combine with the metals only of the ore, thereby separating sulphur in a physical state which will depend upon the temperature." In the revolving drum the particles of ore are finely ground, thus exposing fresh reacting surfaces to the action of the chlorine. The liberated sulphur remains mixed with the gangue, and may be recovered by distillation, etc. At the low temperature requisite for the distillation of sulphur no appreciable amount of metallic chlorides is said to be reconverted into sulphides.

Gas Producer for Zinc Furnaces.—C. Ellis (816,973, April 3) points out that when gas producers for zinc furnaces are operated with steam, it is in practice impossible to completely decompose the steam, which results in an adulteration of the producer gas by water vapor. This steam tends to deaden combustion in the furnace, and is said to have a disintegrating effect on the retorts used in the distillation furnace. C. Ellis, therefore, proposes to generate practically dry combustible gas in a gas producer, in which a deep bed of coal-fire is brought to a state of incandescence by the air of a blast produced by a fan. The generated dry combustible gas is burnt around the retorts or muffles by admixture of air preheated from the waste products of combustion. The products of combustion from the furnace preferably pass through a regenerator of the continuous type. A portion of the products of combustion is diverted either before or after its passage through the regenerator, and is returned to the gas producer for the generation of combustible gas.

Converting Zinc Sulphate Solutions into Zinc Oxide.—C. E. Dewey (815,516, March 20) patents apparatus for this purpose. The process consists of two steps; the solution (in which a quantity of fine coal is placed) being first evaporated and then calcined. Accordingly, two independently-revoluble cylinders are provided, being mounted in axial alinement. Their adjacent extremities are open, and in such proximity to each other that the heat may readily pass from one to the other. In one cylinder the calcining operation, or the removal of the sulphuric acid from the zinc sulphate, is effected, while in the other the solution is evaporated to dryness. The calcining cylinder is located in proximity to a fire-box, from which the heat and products of combustion pass directly into one end of the rotary cylinder, while the heat and products of combustion pass from the calcining cylinder into the evaporating cylinder, and thence into a dust chamber located at the extremity of the evaporating cylinder remote from the calcining chamber.

Aluminium-Tungsten Steel.—S. Parfitt (818,044, April 17) patents the manufacture of a steel of low carbon content, which is claimed to have a high tensile strength and not to be affected in the presence of strong solutions of sulphuric acid and similar corrodents. The iron is melted in an open-hearth furnace and decarbonized in the usual way, so as to contain not more than 0.1 per cent of carbon, [illegible] per cent of aluminium and 6 per cent of powdered tungsten are added.

Articles of an Iron-Hydrogen Alloy.—G. W. Gesner has formerly patented several processes [illegible] and hydrogen. He now (815,491, March 20) uses [illegible] for agglomerating the powdered [illegible] bodies. The powdered alloy is [illegible] hydraulic pressure into the body of desired form, which is then [illegible] at a temperature sufficient to cause the component particles to fuse or partially fuse together without melting them into a liquid mass. For instance, plates ¼ inch thick are maintained at about 2,000° F. for 2 hours. "The alloy is a material of great value for many uses, such as the construction of [illegible] trochemical apparatus, chemical containers, electrical conductors," etc.

Purifying Blast Furnace Gas.—For the utilization of blast furnace gas in gas engines nothing is more important than its purification and removal of all dust. This is done according to Q. Bent (815,674, March 20) by centrifugal force in two chambers, through which the gas is passed in succession. Both chambers contain each a fan. The gas is passed first upwards through a smaller chamber, where, by means of the fan, it is diverted from its upward course and thrown outwardly toward the periphery, whereby the heavier dust particles are separated and escape through four openings into downward pipes. The gas with the lighter dust particles, not yet removed, passes onwardly into the upper large chamber, where the lighter dust particles are removed by means of a large fan.

Refractory Material for Electric Insulation.—D. M. Steward (816,270, March 27) patents a highly refractory fireproof material, which is at the same time a good electric insulator and is unaffected by cold or moisture. The chief constituent is magnesium silicate or aluminium silicate, found in nature as steatite; 100 parts by weight of finely comminuted steatite and 20 parts of water glass are mixed to a plastic mass of about the consistency of workable putty. The mixture is brought into the desired shape by strong pressure, and the water contained in it is expelled in a drying kiln. The article is finally hardened by subjecting it to a temperature of from 1,500° to 2,000° C. for 5 to 8 hours, depending upon the bulk of the mass. Laboratory tests with minute article have shown the best results to be obtained by a 5-minutes' exposure to a temperature of about 1,500° C.

BOOK REVIEW.

METALLURGICAL CALCULATIONS. By Joseph W. Richards, A. C., Ph. D., Professor of Metallurgy in Lehigh University. Part I: Introduction, Chemical and Metallurgical Principles, Problems in Combustion. New York: McGraw Publishing Co. 201 pages. Price $2.00.

The problem of the modern engineer is to carry on [illegible] industrial enterprises at a maximum efficiency. To attain such a worthy object accurate knowledge of the science that [illegible] all material phenomena is indispensable. In all the technological world there has been a constant but slow increase in this knowledge. Thus, we have seen the millwright evolved into the mechanical engineer and the old "bell-hanger" replaced by the college trained electrical engineer. However, the progress in metallurgy has been slow from the rule of thumb man to the technical engineer, especially in the United States. We except, of course, a number of brilliant original thinkers. The chief reason for this slow development is the lack of accurate metallurgical data on which to base metallurgical calculations. The advance made has been, as empirical, manly and noble endeavor has nevertheless been inefficient and wasteful.

In the past five years the mining [illegible]

and other recording instruments, together with the spread of knowledge of modern physical chemistry, has resulted in a partial remedy of this bad condition. At the psychological moment, Prof. Richards has collected, systematized and elucidated all the fundamental chemical, physical and mechanical principles as well as empirical new data together with the old. The result is the book under discussion.

Since the main part of the book is a reprint of the articles which appeared in this journal from March, 1905, to March, 1906, no review of its contents is needed at this place. As a valuable new addition the appendix should, however, be mentioned, containing the statements of twenty-three more problems from practice, without solutions, but with a statement of the final answers.

Judging from the enthusiasm with which the Metallurgical Calculations were received in serial form by our readers all over the world, we are sure that they will be even more welcome in the more convenient and very handsome book form, and will produce much good in the direction indicated by the author in the following words of his preface:

"If the rule-of-thumb is to be replaced in a metallurgical process by scientific operation, the change must be based on experiments, classification of results and calculations therefrom. The principles involved are physical, chemical and mechanical; the scientific metallurgist must master these, use them as tools, and overcome brute nature by their skillful employment.

"Every metallurgical problem is an exercise in pure logic and mathematical reasoning; the premises are observed facts—all that can be learned of the process by direct observation and measurement; the conclusions desired are everything which can be deduced from the premises by hook or by crook, by direct logical process or by inference. In this way data and information are obtained which cannot be directly observed or measured, and which are of the most essential value for thoroughly understanding the process."

Producer Gas and Gas Producer. By Samuel S. Wyer, Mc. E. New York: *Engineering and Mining Journal*. 296 pages. Price $4.00.

This most excellent treatise fills a large gap in technical literature which had previously been only partly closed by the smaller work of Sexton. The rather too numerous (28) chapters are devoted principally to physical and chemical laws, other commercial gases, classification of gas producers (intelligently done), use of steam (very practically handled), efficiency (rather too brief), descriptions of pressure producers, suction producers, by product coke ovens, power plants and gas poisoning.

There is more general information on these subjects within the covers of this book than in any other that we know of; at the same time, we note the absence in many cases of specific data which would be valuable, such as, for instance, the quantity of tar, ammonia and benzene recovered, the approximate chemical composition of the tar and its calorific power. Such details might be added in many places, to the improvement of the value of the book to the practical man. The thermal and physical calculations are very clearly explained, but from the standpoint of the mechanical engineer, the chemist has many short-cuts which are absent from Mr. Wyer's work, which would facilitate greatly the calculations if introduced with proper explanations.

Altogether, the work can be highly recommended to students and practical men as the best treatise there is at present on this special subject.

Practical Electrochemistry. By Bertram Blount, F. I. C. London: A. Constable & Co. Third impression. Second edition, revised and enlarged. Price $3.25.

That this is the third impression of Mr. Blount's now well-known work is the best evidence that, as the author states in his preface to the second edition, the book has been found acceptable by a small but "discriminating" public. As the title implies the book necessarily deals with a very wide range of subjects, embracing processes for electrolytically winning nearly every known metal; processes for separating and recovering new metallic products and finally a series of organic separations and combinations which the modern practice of electrolysis has brought into use.

The strong point of this book, to our mind, has always been the systematic application which the author makes throughout his work of electrothermal and thermochemical data and laws. This gives to the subject a most useful connected sequence and plan, and the author evidently feels the value of this forcibly, as he is very fond of impressing upon the reader that "there is no magical efficacy in electrolytic processes *per se*," but that the boundary line of their utility is found in their economical factors; a somewhat self-evident truth one would imagine at this time of day.

The book begins with a theoretical discussion of the principles of electrolysis, with special reference to the dissociation theory. This subject receives adequate, if not luminous, treatment, and the author's style is thoroughly readable.

Section II. deals with the winning of copper, lead, gold, silver, nickel, cobalt, antimony and zinc, all from their aqueous solutions. There is some slight defective arrangement here, several fusion processes for lead, zinc, etc., are treated, which do not properly belong to the section under the main classification adopted. The other orders of classification also appear to be purely arbitrary and leave something to be desired.

In this section (as also in other parts of the book) we find much matter that is out of date. Old matters, like Borchers numerous unworked or abortive designs being frequently rehashed. The principal additions of value are a short amount of the Tommasi and Salom lead processes, the Phœnix zinc process and a few minor matters, but no account at all is given of such actual and importnat operations of recent date as the Betts refining process now at work at Trail and St. Helens, and soon in South Chicago.

Section III. nominally deals with all igneous solution processes, but actually is confined to aluminium, magnesium and sodium. With the exception of a brief notice of a few recent developments, like Ashcroft's sodium process, we do not find that any fresh information has been produced here.

Section IVV. deals with the electric furnace proper in all its branches. This subject receives able treatment from the electrothermal side, and constitutes one of the striking values of the book. The statistical and practical account of the various industries, however, is somewhat weaker, and such important subjects as phosphorus receive altogether inadequate treatment.

The same remarks apply to Section V., which deals adequately with the recent developments in iron and steel smelting, and to Section VII., which treats of alkali and chlorine processes. Section VI., in our opinion, had been better omitted altogether. Section VIII., the electrolysis of organic compounds, is a valuable addition to the work, more especially as it has not been customary hitherto in electrolytic text-books to touch on this subject. Section IX. and last, deals with "power," and gives the author another opportunity to emphasize his favorite viewpoint of the thermochemical side of his subject.

Altogether an admirable text-book, and one well worth keeping up to date. It bears evidence of thoughtful and scholarly compilation. Its weak points are those nearly unavoidable in such cases, and arise from limited opportunities and limited experience in the "works" direction. Such limitations are often almost inseparable from the lot of professor or laboratory man. We cordially welcome and praise Mr. Blount's latest effort.

Reduction Plant of the Vanadium Alloys Company.

The uses of vanadium were formerly quite restricted, vanadic acid being employed chiefly in calico printing, in the manufacture of aniline black and in the coloring of glass. But

FIG. 1.—REDUCTION PLANT OF VANADIUM ALLOYS CO.

this rare element achieved an eminent practical importance when it was found that as an addition to steel it is able to give it certain extremely valuable properties.[1]

It is well known that the revolution in workshop practice, due to the introduction of high-speed tool steels only a few years ago, has brought about a new and highly important industry—the manufacture of ferroalloys. This commercial evolution has been the subject of numerous articles in our columns during the past years, since it represented one of the most interesting current electrometallurgical developments. It is, of course, well known that in the case of highly refractory oxides and for the production of high percentage alloys, the electric furnace is indispensable.

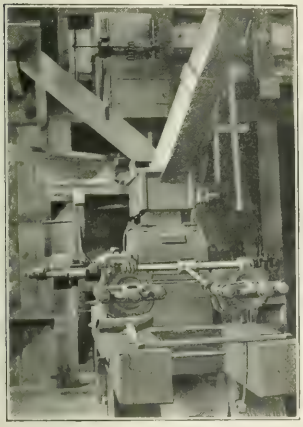


FIG. 2.—ROLLERS.

In this development Europe has formerly been leading, although in the Willson Aluminum Co. this country has the pioneer producer of ferrochrome.

Ferrovanadium has been produced in the past only in small quantities; none whatever on a commercial scale in this country. Most of the ore that has been taken out in the past from the deposits in Western Colorado was shipped to Paris, and there worked up into the ferroalloy.

It is with great pleasure, therefore, that we can record in this article the successful starting of a ferrovanadium plant on Colorado soil. It is the first mill of its kind, built not only in America but in the world, and credit must be given to Mr. Tyndal Rynard, of New York, for his untiring efforts in building this first commercial plant, whereby the value of vanadium can be demonstrated on a large scale as can only be done [illegible]. The general offices of the Vanadium Alloys Co. are at 25 Broad Street, New York City. The reduction works are located at Newmire, Col. Mr. Tyndal Rynard, who is a well-known member of the American Electrochemical Society, is the general manager of the company. Dr. Waldemar Lee, assistant to Dr. Borchers at the Institute of Technology at Aachen, Germany, has charge of the [illegible] equipped works laboratory. The present capacity of the reduction works is 40 tons of ore per day.

The ore worked in the mill runs from 2.5 to 6 per cent of vanadium. The supply of vanadiferous sandstone at the disposal of the company is almost inexhaustible. The company controls mines covering a territory along the San Miguel River of over 12 miles. For descriptions of it reference may be made to the report of Dr. W. F. Hildebrand on vanadiferous minerals in the *American Journal of Science*, August, 1900.

Fig. 1 gives an outside view of the mill at Newmire, Col. Fig. 3 shows one of many mines, 2 miles from the mill. The illustration shows the limestone cap above the ore, which occurs in the form of a blanket vein about 9 feet high.

The process consists essentially of three steps. The first is crushing and roasting with a salt.

The second step is leaching with water and precipitating, by means of ferrous sulphate (green vitriol), a vanadate of iron which contains 50 to 75 per cent V_2O_5, and is free from silica and other impurities.

The third step is the reduction of this vanadate of iron in the electric furnace. The final product of the process, as carried out by the Vanadium Alloys Co., is ferrovanadium containing 25 to 50 per cent vanadium with less than 1 per cent of carbon.

The crushing, roasting and leaching plant was designed,

FIG. 3.—MINE.

built and equipped by the Traylor Engineering Co., of New York City. Although the first works of its kind, the mill was so carefully planned and constructed that as soon as started a few weeks ago it worked to perfect satisfaction. The adjoining illustrations show several interesting details of the plant. For the original photographs we are obliged to Mr. Tyndal Rynard and to the Traylor Engineering Co.

Fig. 2 shows the rollers in the crushing mill. There are three hoppers shown, one for the ore, the second for the salt, and

[1] An excellent summary of researches in this field was given by Dr. Hans Goldschmidt, in our Vol. III., page 169.

the third for the oversize from the screens. The first and the second hopper are worked intermittently. The Traylor centripact screen, which was described in our last volume, pages

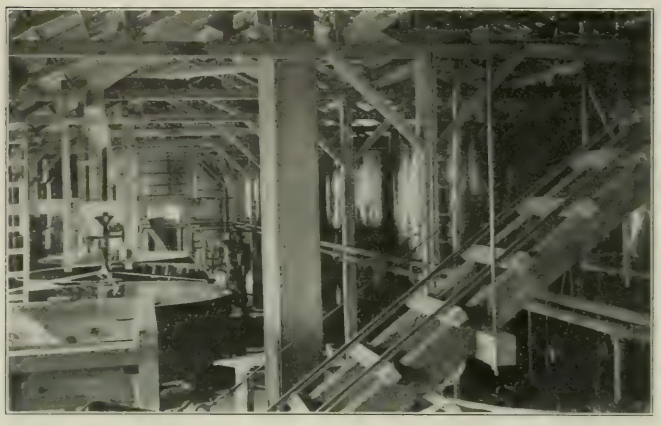


FIG. 4.—LEACHING PLANT.

124 and 163, has here again proven a most valuable piece of apparatus.

After being crushed and roasted the ore is brought to the cooling floor, which slopes both ways to a line, which is inclined downwards to a box discharging into the elevator. The roasted ore is dumped on the brick cooling floor, and then by means of a hose connected with a pump, the material is washed into the elevator box and taken up to the launder, having a draw-off for supplying any of the leaching tanks by means of an automatic distributor.

The elevator is shown on the right of Fig. 4, which, together with Fig. 5, gives a view of the leaching plant.

The leaching tanks are fitted with filters, and connections

FIG. 5.—LEACHING TANKS.

are provided so that by means of pumps the solution may be drawn from any leaching tank to either of the two sump tanks and also from either of the latter to any of the strong solution tanks or weak solution tanks. In general, the connections between the different tanks and pumps are very conveniently arranged, so as to meet the requirements of practice. Since the solution from the leaching tanks will leave a mud residue in the strong solution tanks, it is necessary to avoid drawing this off with the solution to the precipitating tanks. For this purpose a special simple but very effective means is provided.

A double flange is fitted to the bottom of each of the strong solution tanks. The under half connects with the pipe line beneath the floor, going to a Montejus tank. Into the top half of the flange a 2-inch pipe about 2 feet long is screwed. Over this 2-inch pipe a bell is allowed to hang. This bell is made of a 4-inch pipe, 2 feet long, with a cap to raise and lower it. This bell is hung over the top of the 2-inch pipe, so that its bottom edge will be at the level it is desired, to allow the mud to settle before cleaning, so that the clear solution will never be drawn off below this desired point. This clear solution is allowed to flow by gravity to the Montejus tank, which is operated in the well-known manner by compressed air to supply the solution to the precipitating tanks.

It is to be expected that with vanadium now available in any quantities for practical purposes, it will soon cease to be considered a rare metal by steel men. The Vanadium Alloys Co. will then reap the fruit of its progressive pioneer work.

A New Slag Car.

A new and interesting type of slag car has recently been designed by the Power & Mining Machinery Co., of Cudahy, Wis. The two views shown here are of a tenton slag car, of which fifteen were recently built by the above firm for the new Garfield plant of the American Smelting & Refining Co., near Salt Lake City. Some idea as to its size can be gained from the over-all dimensions, which are 15 feet long and 6½ feet high. Each car weighs 27,000 pounds empty, or 47,000 pounds when filled with slag.

The bowl is made in five sections, the bottom being made in one piece with four quarters forming the top. This allows of any worn or broken part being easily replaced. These sections are securely bolted together and tied to a steel supporting ring

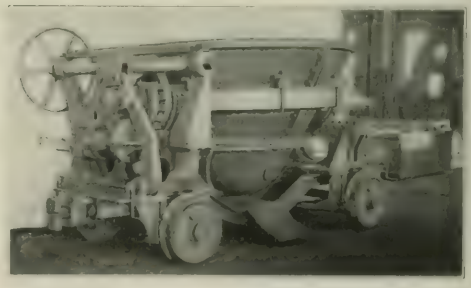


FIG. 1.—SLAG CAR.

by key-bolts. The supporting ring is securely riveted to the pinion-toothed trunnions which roll on a racked track.

The pinion and the rack are both made open between the teeth to prevent their being clogged by any slag which might be spilled into them. The trunnions are rolled by means of a sleeve, which is operated through a lever by a 12-inch compressed air cylinder.

This cylinder is designed to operate on 60 pounds air pressure, and to tilt and hold the pot in any desired position. The air valve for this cylinder is a simple four-way cock, conveniently operated from the side of the car.

All the levers and other operating parts are made of steel, to reduce the weight of the machine without sacrificing strength. A hand-wheel and screw is also provided so that the bowl can be dumped by hand, should the air supply fail from any cause. When tilted to the extreme position the side of the bowl makes an angle of 50° with the vertical, so as to readily allow the slag skull to slide out. The bowl in dumping is carried to the side of the car by the rolling of the trunnions, so as to dump outside of the rail. An apron in the middle of the truck prevents any drippings from falling on the track.

The entire body of the car is carried on eight powerful springs, placed under the front and rear supporting frames

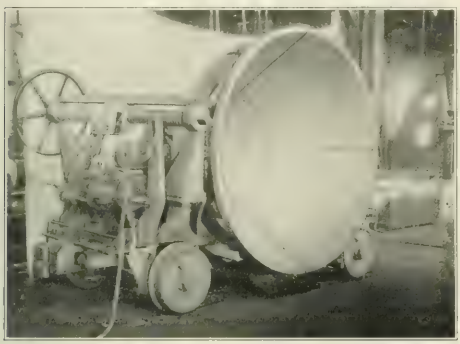


FIG. 2.—TILTING OF SLAG CAR.

and resting on the main bearing boxes. These bearing boxes are fitted with phosphor bronze bearings, which are kept lubricated by removable oil waste boxes. The two supporting frames are tied together by two 100-pound rails securely riveted to them and extended to carry the Tower automatic couplers. These are provided with the usual side-crank attachment for releasing the coupler. One of each pair of wheels is provided with a bronze bushing, so that it can revolve on the axle when going around a curve.

A large factor of safety was used in the design of all the parts of this machine, because of the rough usuage to which it is put. Special pains were also taken to make it as simple as possible to operate for the men usually employed for such labor.

Slag cars of this design are made in various sizes up to 15 tons in capacity, and with or without the air cylinder.

Enamelled Cast-Iron Apparatus for Chemical Industries.

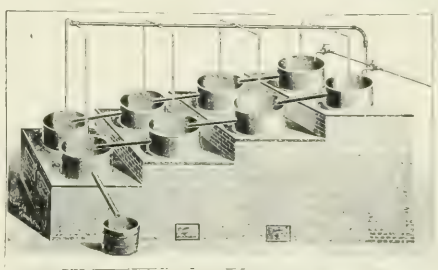
We have received from the Mannheimer Eisengiesserei und Maschinenbau A. G., through Messrs. Frederick Bertuch & Co., New York City, their catalogue 3, containing a large number of illustrations of the numerous cast-iron apparatus made by this company for use in the chemical industries.

According to the special purpose for which the apparatus is intended the composition of the iron is chosen and the company has found it convenient to establish three groups, according to whether the cast iron has to resist the effect of great heat or of acid or of alkaline solutions. Certain characteristic compositions of iron answer these three purposes fairly well.

Of still greater interest, however, are the enamelled cast-iron apparatus which are made by this company in sizes up to 2.3 meters (7½ feet) external diameter and 3.2 meters (10½ feet) depth; that is any sizes up to 10,000 liters (2,600 gallons). There are three grades of enamel used, two of white and the third one of bluish-gray color. The white enamel is used in such cases where such strong acids are to be handled which would gradually attack any bare unprotected iron, however careful its composition might be chosen. The white enamel resists satisfactorily the attack of acids, and at the same time adheres so firmly to the walls of the vessel that even considerable heat evolved by the processes is not detrimental. Care must be taken, however, to protect the enamel, which is like glass, against punching with sharp instruments. In stirring apparatus the stirrers are also enameled so as to resist the acid.

On the other hand, the blue enamel is not as well adapted to resist acids, but is extremely durable and refractory. The kind of enamel must, therefore, be chosen according to the requirements of the special case.

The adjoining illustration shows an evaporation plant for alkaline solutions made by this company. It consists of eight vessels, diameter 700 mm., depth 540 mm., content 165 liters. They are arranged as shown in the figure so that the solution flows over from the upper two vessels to the next two, and so on. The upper two receive the solution from the reservoir. The fire-hearth is placed below the two lowest vessels, so that the liquid in the same, which has already the highest concentration and, therefore, the highest boiling point, is subjected to the highest temperature. By the arrangement of tubes above the vessels, shown in the illustration, a hot-air current is produced by which hot air, which has passed a superheater, is



EVAPORATION PLANT.

blown upon the surface of the liquid in each vessel. Any fog is thereby removed, the evaporation is accelerated and foaming liquids are prevented from overflowing. The operation is continuous and very economical, since great attention is paid in the design to the thorough utilization of the fire gases.

Acetylene for the Laboratory.

By J. K. RUSH

In view of the fact that many of the best chemists in this country are not yet aware that acetylene gas is very valuable wherever an intense heat is desired, I take pleasure in throwing a little light on this subject. I was the first one in this country to install acetylene for laboratory purposes on a large or commercial scale, and I have nothing to regret for having done so, for although it was a radical departure from the uses for which acetylene was employed at that time, as well as at the present, it has given the very best of satisfaction. Some of the statements made by chemists who are using the systems which I installed, show on the face that they are made by enthusiasts, and that acetylene has multiplied all that I had hoped it would.

Acetylene is very rich in heat units, having 1,500 heat units per cubic foot.

The exact quantity of oxygen used to produce perfect combustion varies somewhat according to the construction of the burner and the pressure of the gas at the burner. Authori-

ties differ slightly as to the exact amount of oxygen required to produce the best results in a Bunsen flame.

By way of comparing acetylene with other gases, I will say that the best method is to ascertain how long it requires a given amount of gas to bring 1 gallon of water to the boiling point. Then compare the cost of the gases used. The following figures show the relative cost of acetylene for heating purposes as compared with coal gas and gasoline: Acetylene, 2¼ cubic feet per hour, boiled in 7 minutes. Coal gas, 15½ cubic feet per hour, boiled in 8½ minutes. Gasoline consuming 1 gallon per burner every 8 hours, 7 minutes. Thus it will be seen that acetylene compares favorably with city gas at $1.25 per thousand and gasoline at 16 cents per gallon, figuring 5 cubic feet of acetylene per pound of calcium carbide.

The heat produced by acetylene is very intense; there is a

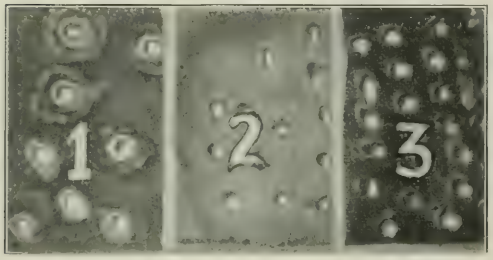
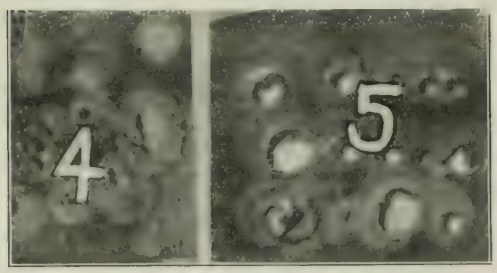


FIG. 1.—MICA. FIG. 2.—ASBESTOS. FIG. 3.—IRON SCREEN.

Holes in each burned with 1 second's exposure.

purplish hue with a cone of greenish-colored flame next to the orifice from which the gas is emitted. When applying acetylene with a blow-pipe with air pressure, it is possible to literally burn a hole through a piece of copper and perform many



FIGS. 4 AND 5.—OTHER EXAMPLES OF THE EFFECT OF THE ACETYLENE BLOW-PIPE.

other things which are absolutely impossible with any other gas. As a matter of fact, the intensity of the heat produced with acetylene can only be compared to that of the electric arc for intensity, but acetylene is much more convenient

Acetylene is now being successfully used for laboratory purposes by chemists for analytical work, in some instances on a large scale. In one of the largest steel plants in the United States acetylene is now used for these purposes after other gases had formerly been used. The results have been extremely satisfactory, since acetylene was found to be far superior to either natural gas or city gas, especially in view of the fact that this intense heat can be obtained free from smoke.

With such a rich gas as this to deal with it was no easy matter for manufacturers to design burners suitable for the acetylene Bunsen flame. But the obstacles which required constant and careful study to overcome have now been mastered to such a degree that it is possible to procure acetylene burners for practically every purpose conceivable. For the purpose of showing what an intense heat may be produced with acetylene under proper conditions and with the right kind of apparatus, I submit photos of specimens of work done with the blow-pipe, which speak for themselves. These are reproduced in Figs. 1 to 5.

One very important feature in connection with the use of acetylene for laboratory purposes, is the fact that according to the statements made by those who are using it extensively and on a large scale, "it does not corrode nor cause any deterioration of platinum ware."

FIG. 6.—ACETYLENE GENERATOR. DISSOLVING PROCESS.

Of course, it is most important to employ the proper apparatus with acetylene. It is now generally conceded that only when it is generated from carbide at a very low temperature is it possible to get it in its purest state and in satisfactory quantities. At Cornell University, Ithaca, N. Y., extensive tests have been made during the past year, and are still going on, whereby it has been shown that the patent process known as the "dissolving process," produces a coolness not obtained in any other system of generating acetylene, and that it surpases all others both in quality and quantity of gas produced.

This "dissolving process" is extremely simple and unique in its workings, and is operated upon the following lines: When carbide and water come in contact with each other a large amount of heat is set free, and it is necessary to get rid of this heat. In all other systems of dropping carbide into water, a tank, or dead body of water is used, wherein only the surface of the water gets heated, as the slacking of the carbide is very rapid. Consequently the surface of the water where the acetylene is generated assumes a temperature near the boiling point, while the remainder of the water in the tank remains relatively cool.

Now, the feature of the "dissolving process" is the provision of means for radiating and circulating the entire body of water during the process of generating gas. On this account the temperature of the water is kept at from 100 per cent to 150 per cent degrees lower than with other systems, and hence the improvement in quality of the gas produced, together with a gain of from 10 per cent to 35 per cent in quantity, which is a matter worthy of consideration when the installation of an acetylene plant is being considered.

CANANDAIGUA, N. Y.

Extensions of West Allis Works of Allis-Chalmers Company.

The new extensions to the West Allis-Milwaukee works of the Allis-Chalmers Co., which, when completed, will add 861,000 square feet to the plant's present floor area of 652,000 square feet, and make the entire plant capable of affording employment to 11,000 persons, will place the West Allis works among the large industrial plants of the country. The extended works, together with the capacities of the other works of the company in Milwaukee, Chicago, Cincinnati and Scranton, will be capable of affording employment to a total of approximately 18,000 persons.

The extensions built are as follows: Three machine shops, parallel to the existing units, Nos. 1, 2 and 3, each 575 feet long and 145 feet wide for units Nos. 1 and 2 and 168 feet wide for unit No. 3; an erecting shop, 1,136 feet x 113 feet, running north and south, as an extension of the existing erecting shop and adjoining the six machine shop units opening out into it; the extension of the existing foundry, north and south, which has a total length of 994 feet over all and a width of 222 feet, and the extension of the pattern and pattern storage building, which also has a length of 994 feet over all and a width of 119 feet.

The work of constructing these extensions involves an expenditure of over $3,000,000. Two buildings, the extensions to the erecting shop and the pattern shop, have been practically completed, while substantial progress has been made on the remaining buildings.

The West Allis site, situated in the town of West Allis, on the outskirts of Milwaukee, has a frontage of 1,575 feet, and runs back 2,690 feet, or more than half a mile, giving nearly four and a quarter million square feet of ground space, or about 100 acres, adjoining the tracks of the Chicago, Milwaukee & St. Paul Railway.

The general plan of the West Allis works provides for two sets of building units, one composed of those in which the work is common to every class of product, like the pattern shops, erecting shops and foundry; the other group, comprising the machine shop units, where the work is specialized for various machine operations. The two groups of buildings lie at right angles to each other; those of general utility lengthwise of the plant, north and south. The machine shops, six units in all, run crosswise of the plant, adjoining the erecting shop at the east end. Facing the ends of the machine shop units Nos. 1 to 6 stands the foundry with its extensions, extending north and south. The pattern storage shops stand farthest westward in the group and parallel to the foundry, with a span of 98 feet between them.

The extensions to the erecting shop, foundry and pattern shops are duplications of the existing units in practically every detail of construction. After the completion of the works we hope to give a full illustrated description

Instrument Factory.

The rapid growth of the Leeds & Northrup Co., of Philadelphia, is indicated by the accompanying illustrations, which show the new quarters recently occupied by this firm. The building was designed and built to meet the special require-

FIG. 1.—SHOP AND LABORATORY OF THE LEEDS & NORTHRUP CO.

ments of constructing and adjusting electrical measuring instruments. It has embodied in it many features of interest. The location, at 4901 Stenton Avenue, is away from the thickly settled part of the city, and has, therefore, all the advantages of ample light and freedom from dust. In selecting the site for location, a lot of sufficient size to allow for future expansion was acquired.

The main structure is 110 feet x 52 feet, three stories and

FIG. 2.—SHOP.

height, fire-proof in construction throughout, and has all provision for the comforts of the employee.

The laboratory occupies 85 feet of the third floor, and by means of skylights in addition to numerous side windows, an abundant supply of light, so essential for work of precision,

FIG. 3.—NORTH SIDE OF LABORATORY.

is furnished. The illustration shows the north side, which is practically a duplicate of the south side. Careful study has been given to the arrangement, distribution and insulation of the circuits for use in current, potential and general testing work. This current is furnished by a large battery of storage cells, the terminals of which are connected to a specially constructed mercury switchboard. This switchboard permits grouping the cells in any desired combination for any particular circuit. The battery room is shown at the end of the laboratory, on the right, with the charging panel outside. The mercury switchboard is not visible. All calibrating and standardizing apparatus is permanently set up in circuit, with galvanometers best adapted for the demands to be made upon them. It will be noted from the illustration that the Lepel galvanometer is extensively used.

In conjunction with the laboratory, but in a separate building, provision has been made for work in high temperature measurement.

The shop occupies 85 feet of the second floor, and is electrically driven. The equipment of tools and machines is specially adapted for the production of electrical measuring instruments. Co-operating with the shop and laboratories is a

inspection department, through which all apparatus must pass before shipment can be made. This inspection is most rigid, involving both mechanical and electrical tests, and thus insures the highest standard of workmanship, material and adjustment.

The basement is taken up by the heating and lighting plant, shipping room, locker room and wash room for the mechanics, fire-proof vault for records and a well-lighted room, 52 feet x 55 feet, for future manufacturing purposes.

The drafting and designing department is located on the second floor so as to be in close touch with the shop. In this department are filed accurate records and drawings of all apparatus manufactured. The sales office, stock room and show room are located on the third floor. In the show room are displayed, in easily accessible wall cases, a complete line of the more important instruments manufactured. In addition to this display a number of instruments are connected in circuit for demonstration under actual working conditions.

The occupancy of the new shop and laboratory has permitted

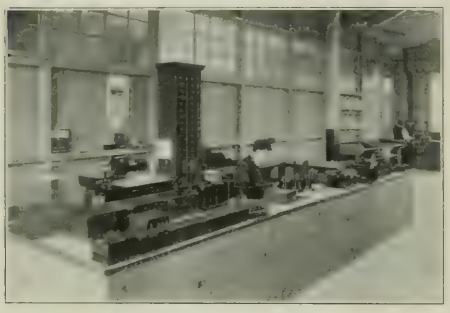


FIG. 4.—INSPECTION DEPARTMENT.

the installation of additional machinery and an increase in their mechanical and laboratory forces. As a result, a larger amount of manufactured stock and standard parts will be carried, which will insure prompt deliveries.

A cordial invitation is extended by the company to all interested persons to visit and inspect the new plant of the Leeds & Northrup Co.

Pyrometers.

A direct-reading thermoelectric pyrometer has recently been placed on the market in England by Messrs. Crompton & Co. This is another indication of the great interest which is now being paid to the importance of measuring temperatures exactly in industrial work.

There is nothing essentially new in the principle of the Crompton pyrometer. Like all thermoelectric pyrometers it consists of a thermoelectric couple, the indicating instrument

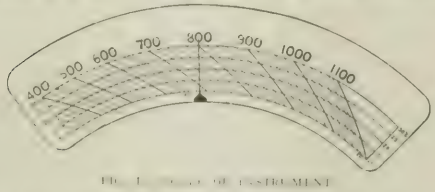
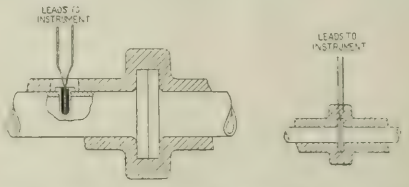


FIG. 1.—FACE OF INSTRUMENT.

and the leads connecting the couple and the instrument. For temperatures up to 1,000° or 1,100° C. the couple is constructed of nickel and steel. A nickel rod runs down the center of a steel tube without touching it, and at one end the two materials are welded together. At the other end terminal clamps for the connecting leads are brazed to the rod and tube. For measuring temperatures of boiler flues, superheated steam, and in other cases where the temperature does not exceed 500° C. a constantan-copper couple is used. For measuring temperatures at various parts of a steam system, such as valve boxes,



FIGS. 2 AND 3.—CONNECTION OF LEADS TO INSTRUMENT.

steam chests and different points of the pipe line, small wells or pockets are fixed at the points where the temperature is to be measured, as indicated in Figs. 1 and 2.

In cases where the steam pipes are too small to admit of a pocket, and where there is no convenient flange joint, the couple should be fastened to the pipe and heavily lagged. The lower the temperature to which the couple is continuously subjected, the longer it will last, and copper constantan couples, when measuring temperatures up to 500° C., are said to have a very long life. The life of the couple is considerably increased by protecting it with a renewable steel sheath or a porcelain cover, or by embedding it in a shallow trough containing silica. Platinum wire pyrometers resist the action of the furnace gases permanently if protected by a porcelain tube, but couples of this description are costly and easily broken.

The scale of the instrument is constructed to take into account variations in the temperature of the indicator, these variations being measured by a small thermometer attached to the face of the instrument. The several arcs are calibrated for different temperatures of the indicating instrument, and the reading should be taken on that arc which corresponds with the thermometer reading. For this purpose the end of the pointer is finished to a knife edge, which crosses radially the whole depth of the scale. Since close proximity of iron may alter the readings to some extent, the instrument should not be fixed in close contact with an iron mass, such as a pillar, but where possible it should be fixed to a wooden support or brick wall.

Oxy-Hydrogen Apparatus for Welding.

By Frank C. Perkins.

The accompanying diagram, Fig. 1, shows the arrangement of a new German electrolytic equipment for the production of

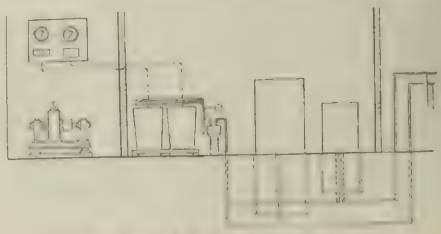


FIG. 1.—PLAN OF ELECTROLYTIC PLANT.

oxygen and hydrogen gases, while Fig. 4 illustrates the method of using the oxyhydrogen flame for welding purposes. Fig. 3 shows a large plant for producing more than 40,000 cubic feet of hydrogen, and more than 20,000 cubic feet of oxygen in a day of 24 hours, and Fig. 5 a smaller cell for the electrolytic

production of oxygen and hydrogen, as constructed by the Schuckert Co., of Nürnberg, Germany.

Up to the present time the process of welding by the use of oxyhydrogen gases has been very satisfactory, but has been too expensive for very extended use, since the hydrogen and oxygen have been produced largely by chemical means. The electrolytic production of oxygen and hydrogen has been well known ever since the beginning of electrochemistry, but commercial apparatus have been placed on the market only during the last years. At the present time, in addition to the Schuckert electrolytic apparatus, an electrolyzer has been designed by the Oerlikon Co., which was described and illustrated in this journal, Vol. II., p. 293.

The Schuckert apparatus consists of a cast-iron tank, containing a number of cast-iron electrodes in various chambers separated by diaphragms, extending from the top downward about three-fourths the depth of the cell, the gases being conveyed through a pipe system to separators, whence the wash water is returned to the electrolytic cells.

The electrolyte is a 20 per cent solution of caustic potash. The cells are embedded in a sand layer about 2 or 3 inches in thickness, arranged to protect the apparatus from heat radiation, the temperature of the electrolyte being maintained at about 75° C. This is said to be the most satisfactory temperature, as the lowest voltage is required at this temperature for decomposing the electrolyte. The pressure is from 2 to 3 volts, and the various cells are connected in series very much the same as a battery of accumulators. The hydrogen and the oxygen gases when generated at the electrodes are conducted through pipes to separate gasometers or tanks for storage.

The use of hydrogen and oxygen is of great importance, particularly for lead burning, and is employed to advantage for welding purposes, the electrolyzer illustrated being designed particularly for that purpose. A more detailed discussion of autogenous lead welding was given by M. U. Schoop in this journal, Vol. III., p. 260.

When current is obtained at low rates from hydroelectric power plants, or from large central power stations and railway plants, the cost of production is low. It is stated that the time for welding 1 meter of sheet iron 3 mm. in thickness is about 15 minutes, while for welding 1 meter of sheet metal .5 mm. in thickness, it is from 4 to 6 minutes.

FIG. 2.—GAS TANKS CONNECTED TO BURNER.

The Schuckert electrolyzers produce the oxygen and hydrogen gases of 97 to 99 per cent purity. The diagram of the electrolyzer plant noted in Fig. 1 shows the direct-current generator at the left, the switchboard above it, connected with the electrolyzer in the center, the piping system conveying the gases to the gasometers at the right, the larger tank being provided for the hydrogen and the smaller for the oxygen gases.

The gases when generated by the electrolyzer are often pumped into oxygen and hydrogen tanks for storage under pressure, and these tanks are transported to any manufactur

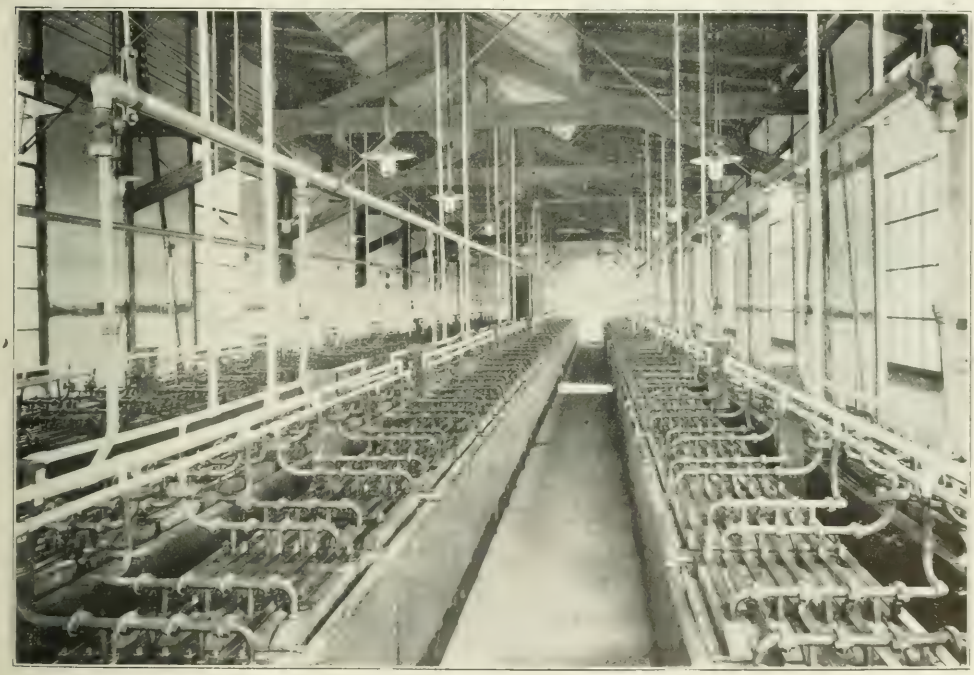


FIG. 3.—ELECTROLYTIC PLANT FOR A DAILY PRODUCTION OF MORE THAN [illegible] CU. FT. H AND 50,000 [illegible] O.

ing plant where welding or lead burning is to be done and connected to the oxyhydrogen burner, as noted in the accompanying diagram, Fig. 5.

Mining and Scientific Press.

In the disaster which has befallen San Francisco, our esteemed contemporary, the *Mining and Scientific Press*—the leading mining journal of the West—has been a heavy loser. All the records, the library, the note-books of the editors, the manuscript for the printer and the whole of the plant in the

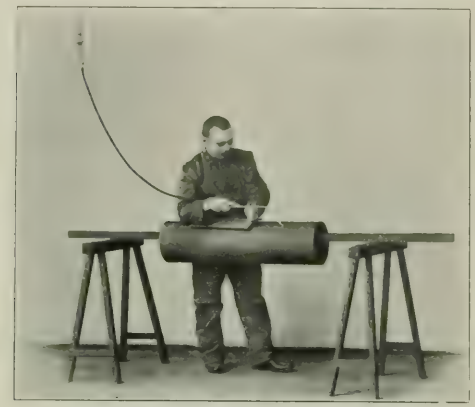


FIG. 4.—WELDING WITH OXY-HYDROGEN FLAME.

FIG. 5.—CELL FOR PRODUCTION OF HYDROGEN AND OXYGEN.

composing room lie buried under the ruins. Fortunately, a duplicate of the subscription list had been kept in Berkeley, and was thus preserved. But it is a splendid sign of the spirit which reigns to-day in San Francisco, that while the fire was still raging new offices and a new printing shop were secured in Berkeley, the new address of the journal being at the First National Bank Building at Berkeley.

In an announcement to the readers, Mr. T. A. Richard, the distinguished editor of the *Mining and Scientific Press*, writes as follows: "Our plant has been demolished, but this journal is built on nothing so ephemeral as paper and on nothing so cheap as machinery. It is based upon the support of many thousand readers. . . . The good-will of the *Mining and Scientific Press* is locked up in no safe, confined to no printing room, it cannot be shaken by an earthquake or consumed by fire. And there is another something that is not destructible by physical misfortune or financial advertising, and that is the spirit that gives life to the printed word."

Notes.

American Eelectrochemical Society.—At the meeting of the board of directors, held on April 7, the following gentlemen were elected members: E. W. Bacon, Philadelphia, Pa.; J. Lainson Wills, Brooklyn, N. Y.; Edward E. Free, Ithaca, N. Y.; B. G. Klugh, Marquette, Mich.; Philip C. Walsh, Jr., Newark, N. J. At the next meeting of the board of directors, which will be held in Ithaca on May 1, the names of the following gentlemen will come up for election: E. A. Ashcroft, New York City; Arthur Hough, Dover, N. J.; H. H. Fries, New York City; A. A. Breneman, New York City; Fanny R. M. Hitchcock, Philadelphia, Pa.; A. L. Queneau, South Bethlehem, Pa.; C. M. Joyce, Arlington, N. J.; J. E. Teeple, New York City; David Weeson, Montclair, N. J.; George Archbold, New York City; Chas. W. Moulton, Poughkeepsie, N. Y.; Irvin Kline Giles, Ithaca, N. Y.; S. H. Blake, Pittsfield, Mass., and Julius Kahn, New York City.

Metallurgy and Electrometallurgy at Lehigh.—The following is a list of the titles of seniors' theses on metallurgy and electrometallurgy at Lehigh University: W. H. Hendricks, a calorimetric investigation of copper slag and matte; H. M. Burkey and A. W. Moore, on the electric conductivity of melted metals; S. J. Cort and H. R. Lee, design and construction of an induction furnace; Y. H. Hardcastle and F. R. Pyne, an investigation of the properties of cryolite-alumina electrolytes; P. H. Herman, on the electric conductivity of some refractory materials.

Protective Coatings.—The American Society for Testing Materials, at its annual meetings, provides for a special session for the consideration of protective coatings. The executive committee has approved of the suggestion that this subject of protective coatings be amplified and made to include all paints, varnishes, etc., having protective purposes, although not designed exclusively for the protection of iron and steel. An advisory committee, consisting of G. W. Thompson, chairman, Robert Job and S. S. Voorhees, has been appointed to suggest a programme for the next meeting.

A New Departure in Electroplating.—The New York General Metal Works, with offices at 9 East Twenty-second Street, New York City, is the consolidation of the Nonick Co., controlling patents of Mr. G. F. Marsh, and of the Architects' Bronze Works, owners of Dr. John F. Daly's patents on architects' bronze and metal lace inventions. While the chief product of the company is at present architects' bronze they also make metal grills, balustrades, newel posts, metal window frames, chandeliers and other specialties of wood or plaster, etc., and covered with an electrodeposit of bronze, copper, brass, silver or gold. The core material may be left in or taken out; in the latter case the shell of the electrodeposited metal only remains. The company also makes what they call Nonick china, a seamless and indestructible metal rim being deposited around the edge of china or glass or on any part desired. Electrodeposits are also made on lace curtains, lamp shades, etc., which process not only contributes much to the beauty of the articles but also makes them fireproof.

Division of Willson Aluminium Co.—We have received the following official announcement: "The Virginia Laboratory Co., 99 Cedar Street, New York City (Geo. O. Seward, president), has been formed for the purpose of owning and exploiting the processes and patents of F. von Kugelen and Geo. O. Seward. The Virginia Electrolytic Co., 99 Cedar Street, New York City (Geo. F. Seward, president, Geo. O. Seward, vice-president and general manager), has leased the hydroelectric plant at Holcomb's Rock, Va., from the Willson Aluminium Co., and will operate under licenses from the Virginia Laboratory Co. The Willson Aluminium Co. will continue its ferroalloy business at its plant at Kanawha Falls, W. Va." The chief product of the Willson Aluminium Co. is

ferrochrome. The extensive experimental work of Messrs. H. von Kügelen and Geo. O. Seward during the past years in Holcomb Rock had to do especially with the production of metals and alloys free from carbon and with the refining of metals and alloys in general; they have also developed an electrolytic process for winning tin from scrap. The patents relating to these inventions have been abstracted in our columns. It was reported some time ago that a detinning plant is to be erected, using the Kügelen-Seward process, but no official confirmation of this report could be obtained.

Samples of Standardized Iron and Steel.—For several years the American Foundrymen's Association has prepared and distributed samples of standardized irons. By an agreement with this Association, the samples which have been prepared under its direction have been transferred to the Bureau of Standards, and this Bureau will in the future have charge of the preparation and distribution of such samples, and will also, as soon as practicable, undertake the preparation and distribution of samples of standardized steels. At present only a single copy of steel has been prepared. Further information may be found in Circular 11 of the Bureau of Standards, Washington, D. C.

Morgan Gas Producer in Europe.—The following report from Europe is interesting, with respect to the progress made abroad by the Morgan continuous gas producer, which is so well known in this country. Messrs. Ehrhardt & Seimer, who are the European manufacturers, state that after a sharp competitive test against the Poetter producer, they have received an order for twelve 10-foot Morgan producers from a steel works in Bochum. They have also an order from the Société St. Gobain, in Pisa, for six feeding mechanisms, also an order from the Milowice Iron Works, in Russian Poland, for three producers. The latter, they think, is an entering wedge to a considerable producer business in Russia.

Gas Engines.—The National Carbon Co., of Cleveland, Ohio, have for some time operated gas engines of different makes up to 200 hp. in their various factories. They are now installing in their works at Clarksburg, W. Va., a 600-brake horse-power Koerting gas engine, supplied by the De La Vergne Machine Co., of New York. The engine will be used for general power purposes, and will operate on natural gas.

"Metals."—Mr. Sherard Cowper-Coles, the well-known British electrochemist, and formerly proprietor and editor of the *Electrochemist and Metallurgist*, is now editing a fortnightly metallurgical section of the London *Engineering Times*. The first instalment appeared in the issue of March 22.

Steam Turbines.—The increasing demand for Westinghouse Parsons steam turbines is indicated by the fact that the Westinghouse Machine Co. received during the months of February and March orders for thirty-five steam turbines, aggregating approximately 50,000-brake horse-power capacity. The largest is of 7,500-kw. capacity, or 11,000-brake horse-power, and will be installed by the Transit Development Co., of Brooklyn. It is of the well-known multiple-expansion parallel flow type, and will drive a direct-connected alternator running at 750 r. p. m., and developing 10,000 electrical horse-power, at full load. By means of a secondary governor valve, with which all Westinghouse Parsons turbines are provided, 12,000 kw. may be developed when running condensing, or full load when operating without condenser. This corresponds to a maximum capacity of at least 16,000 hp. at the shaft.

Things Chemical.—Under the title "Things Chemical," a monthly bulletin is published by the Charles E. Sholes Co., of 164 Front Street, New York, in the interests of the J. T. Baker Chemical Co., the Georgetown Chemical Works, the Naugatuck Chemical Co., the Industrial Laboratories, the Triolet Co., the Metzger Chemical Co. and the Joseph Binns Chemical Works. The first number of the bulletin appeared in April, and [illegible] with the products of the above named companies. A brief price list is also added.

Bacteriology.—The [illegible] of the Bausch & Lomb Optical Co., of Rochester, N. Y., [illegible] an illustrated review of the various types of [illegible] apparatus made by this company, which meet all requirements of bacteriological laboratories.

Electric Hoists.—A recent neatly illustrated pamphlet of the Wellman-Seaver-Morgan Co., of Cleveland, gives descriptions of the various types of electric hoists made by this company, both for operation by direct and alternating-current motors.

Blowers and Fans.—We have received from the American Blower Co., of Detroit, Mich., a bound set of their chief catalogues, which contain a very large amount of useful practical information, with numerous illustrations. This set of catalogues is, in fact, a reference work on blowers, fans, mechanical draft, etc. The catalogues deal in detail with the following subjects: "A B C" blowers for cupolas, forges, melting and heating furnaces, forced draft, pneumatic tube systems and all other appliances where a strong, steady air pressure is applied; A B C exhaust fans, disc ventilating fans, steel plate fans, heating apparatus, fan system of heating and ventilation, mechanical draft, forced and induced by blowers and exhaust fans, two interesting papers by F. R. Still and S. S. Howell on mechanical draft; waste heat drying system, dry kilns for timber purposes, steam pumps, vertical engines.

Smelting.—Bulletin 50 of the Traylor Engineering Co., of New York City, deals with furnaces and smelting accessories, and contains many illustrations of modern types of copper matting furnaces, silver-lead furnaces, fore-hearths, copper converters, cupelling furnaces, blowers, etc.

Personal.

Dr. G. G. Revay, of 45 Cedar Street, New York, the well-known fuel specialist and an authority on modern by-product coke ovens, left on April 18 for Europe, per steamship "Kaiser Wilhelm der Grosse." Several members of the staff of the United States Steel Corporation sailed on the same steamer, for the purpose of inspecting different plants in connection with the big projects now under consideration at the various works. We learn that Dr. Revay is also interested in several chemical manufactures, and intends to investigate some new electrochemical processes in Europe, which, it is said, will be established in America under most advantageous conditions, and for which purpose negotiations for large amounts of cheap water-power are carried on by some influential parties. Dr. Revay, as will be remembered, crossed to Europe last Fall in company with Mr. Charles M. Schwab and some of his engineers, to visit several important European works, and on his return worked out extensive plans for the Bethlehem Steel Works.

Edwin Reynolds and the Design of the Horizontal Vertical Type of Steam Engine.—The award of a patent relating to the horizontal vertical type of steam engine by the United States Patent Office, Jan. 30, 1906, to Mr. Edwin Reynolds, dean of Corliss engine designers, and for many years identified with the Allis-Chalmers Co., together with the recent remarkable showing in the tests of combined vertical and horizontal engines in the New York subway power house, suggest an incident characteristic of Mr. Reynolds related by those who were close to him at the time when this type of engine was first evolved, nearly seven years ago, when the application was filed for the patents only recently granted. The incident referred to occurred at the time when the Allis-

Chalmers Co., as one of the largest builders of steam-driven units in the world, having built eleven 3,500-kw. standard cross-compound, vertical direct-connected engines for the Metropolitan Street Railway Co., was called upon for advice as to the type of engines to be used in the immense new power house then being planned by the Manhattan Railway Co., New York City. It was the intention of their engineers to install units of 5,000-kw. capacity. The first type of engine considered was the cross-compound, vertical machine, similar to those furnished for the Metropolitan Street Railway, and some correspondence passed between Mr. Reynolds and the engineers of the Manhattan Railway Company on the subject, so that finally Mr. Reynolds was invited to come to New York and discuss the matter in detail. Mr. Reynolds left Milwaukee with the question of the type of engine still unsettled, but with the understanding that the straight, cross-compound, vertical unit would be used if it were found practicable. On the train, en route for New York, however, Mr. Reynolds evolved in his mind the possibility of using a combined vertical and horizontal engine, allowing four cylinders to be connected to the one shaft, and before reaching New York he was prepared to offer to the engineers in charge a definite proposition, covering a combined vertical and horizontal four-cylinder engine, now generally known as the "Manhattan" type; and, furthermore, he submitted to them the size of cylinders which he would recommend, the size of crank-pin and cross-head pin, and the size of main journals, with rough sketches of the proposed unit as he had planned it in his mind's eye. As a result of that conversation the Allis Co. received orders for eight pairs of such machines. On Mr. Reynolds' return to Milwaukee work was at once begun on the design of the new engines in accordance with the data used by him in his New York proposition. When the complete designs were ready, it was found by comparison that in the rough notes made by Mr. Reynolds while on his way East, he had arrived at identically the same dimensions of various parts as those obtained from the carefully worked out designs. It is said that this incident is only one of a great many which have occurred during Mr. Reynolds' long service with the old E. P. Allis Co., and later with the Allis-Chalmers Co., which display his wonderful engineering ability and his promptness in making decisions. The horizontal vertical type of engine was later adopted by the Rapid Transit Subway Construction Co., showing the confidence of its engineers in these units. The patent, No. 811,520, which has now been issued to Mr. Reynolds, has been assigned by him to the Allis-Chalmers Co., who have the sole right of manufacture of this engine.

Digest of U. S. Patents.

Compiled by Barnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

PRODUCTION OF OZONE AND MISCELLANEOUS GAS REACTIONS (Continued.)

No. 272,187, Feb. 13, 1883, Charles E. Ball, Philadelphia, Pa.

Produces illuminating or heating gas from hydrocarbon liquids, by injecting the latter into, upon or through an electric arc between carbon electrodes. It is stated that the hydrocarbon is substantially completely converted into a fixed gas, some carbon being deposited, however, upon the electrodes and serving to repair the waste of the same.

No. 282,070, July 31, 1883, Hugo D. Hall, New York, N. Y.

For the production of ozone, utilizes the discharge from a Holtz or other induction generator, the discharge being diffused through a space through which a current of air or gas is forced by a blower. The discharge surfaces may comprise wooden plates or partial partitions so arranged that the air is caused to pass between them, the partitions being provided with numerous contacting points, or metal partitions suitably roughened or serrated may be used. In order to prevent sparking it is preferable to arrange an air gap at the rear of one or both discharge surfaces, in such manner that these surfaces are not in direct electrical connection with the terminals of the static machine.

No. 291,463, Jan. 1, 1884, Charles E. Ball, Philadelphia, Pa., and Charles S. Bradford, Jr., West Chester, Pa.

An improvement on the process of No. 272,187 above. The gas produced by vaporizing hydrocarbon liquids in arc is stated to be too rich in carbon for certain purposes, and the present invention consists in diluting the gas by inert gases, such as hydrogen. In the apparatus shown, two arcs are employed, the hydrocarbon being injected into one and water or steam into the other. The gaseous products of the two arcs are mixed and collected. The gas may be utilized for operating a gas engine, which in turn operates the dynamos for maintaining the arcs.

No. 342,548, May 25, 1886, Alfred O. Walker, Chester, England.

Utilizes high-tension discharge for effecting the deposition of fumes from smelting furnaces and for analogous purposes. In the apparatus shown, the furnace gases conveying the fumes are conducted into any suitable flue or chamber, and subjected to a high-potential discharge from metal points or equivalent projections. As illustrated, the discharge passes from a metal framework provided with points to the walls of the receptacle, one terminal of the generator being grounded. A number of similar devices may be applied at short distances along the flue. The deposit may be flushed out by water or otherwise removed in any convenient manner.

No. 361,923, April 26, 1887, Arthur Brin, 59 Brompton Crescent, County of Middlesex, England, and Leon Q. Brin, Paris, France.

An ozonizer constructed of three concentric tubes of varying sizes, disposed to leave an air gap between the inner and intermediate tubes. The inner tube and the annular space between the outer and intermediate tubes are filled with finely-divided conductive matter, and constitute opposite discharge terminals. The divided matter may comprise plumbago, filings of iron, zinc or copper, or dust shot. The discharge passes in the form of a large number of sparks. In the apparatus shown several ozonizers constructed as above are connected electrically in multiple, the air or oxygen to be ozonized passing through several devices in series.

No. 387,281, Aug. 7, 1888, Hartley Fewson, Buckingham, England.

Apparatus intended particularly for treating noxious gases in order to render them harmless. The device comprises a glass lined box, provided with glass partitions so arranged as to provide a zizzag path for the gases. Each partition is faced with tin-foil on the side nearest the gas inlet, and alternate partitions are connected with opposite poles of the high tension generator. The lead wires are of platinum, and are arranged close to but not in actual contact with the tin-foil of the respective electrodes.

No. 397,402, Feb. 10, 1889, Gottfried H. Merkel, Boston, Mass.

The ozonizing chamber, a wide, horizontal glass tube, contains a narrow glass tube which is bent back and forth into several loops. A high-resistance wire of platinum, selenium or aluminium extends through the loops and is connected at its opposite ends to an induction coil.

No. 409,903, Aug. 27, 1889, Hilarion A. B. H. de Vars, Paris.

The electrodes are a concentric inner glass cylinder lined with tin-foil, and an outer glass bell-jar coated with tin-foil, both enclosed in another bell-jar. The lower ends of the cylinder and jars fit in grooves in an ebonite stand. The air passes up through the stand and cylinder and downward outside the cylinder through slots in the stand, the ozone being delivered into a nickel-plated receiver.

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Cornell Meeting of the American Electrochemical Society.

The Ithaca meeting of the American Electrochemical Society—a full report of which will be found in this issue—was a most decided success. In the atmosphere of Cornell the members and guests of the Society all felt at home, and the beauty of the town revealed itself loftily in the glory of Spring. Indeed, the view from the Chemical Laboratory down to the lake and the valley in the tender green of the young May, was an inspiration in itself. For the hospitality extended to the Society, not enough credit can be given to the local committee in general and to the distinguished retiring president of the Society, Prof. Wilder D. Brancroft, in particular. All visits, excursions and social functions were extremely enjoyable, while the programme of the three sessions, devoted to the reading of papers, was exceptionally well balanced. There were presented thirty papers in all, dealing in about equal parts with theoretical and practical subjects. Ten papers were contributed from engineers and chemists in professional practice, twenty papers from universities, among which Cornell figured with seven papers, Wisconsin with six, Massachusetts Institute with three, Lehigh, Michigan, Princeton and Karlsruhe (Germany) each with one paper. The only drawback was the comparatively small attendance, and it is to be feared that no radical change can be expected with regard to this point if the policy of holding two meetings a year is to be kept up as heretofore. Perhaps the compromise plan, mentioned on another page of this issue, and recommending one annual principal meeting in Spring and a second and secondary meeting in Autumn in New York City, may prove the best solution of the problem. Under such an arrangement every member would make it a point to try to attend the meeting in Spring, which would be held in a different city each year. The second meeting in Autumn should logically be held in the city which is the official seat of the Society; but this would, in the present case, defeat the purpose of the arrangement. New York City seems the most suitable place on account of its large resident membership, and for the other reason that many professional men have to visit New York in the Autumn, and could combine this visit with the attendance of the meeting.

The Chemistry of Electric Lamps.

Few people realize the fundamental importance of the proper solution of the chemical problems which underlie almost any engineering proposition. The user of electric lamps considers them as a product of the electrical industries, which is, of course, true. He also understands that electrical engineering knowledge is required for the proper installation of the lamps. But very few realize that, for instance, the production of the filament of an incandescent lamp is essentially a problem of

chemical engineering. Mr. Edison's momentous pioneer work of twenty six years ago was the foundation of the modern electric lighting industry. But in giving to the world the first commercial electric incandescent lamp, Mr. Edison was essentially a chemical engineer. Any real progress in the invention of new lamps must be ultimately based on chemical research work. The truth of this statement will be evident when we look over the progress that has recently been made in this field. It is a timely subject. For we are now right in the midst of a development the end of which nobody can yet prophesy. It is unnecessary to dwell on the rapid development of electric lighting by incandescent as well as arc lamps during the last twenty years of the nineteenth century. It seemed that the gas lighting industry was doomed, that the electric lamp would practically replace the gas lamp just as the latter had replaced the oil lamp. This would probably have occurred if the chemical engineering work of Dr. Auer Freiherr von Welsbach had not saved the situation for the gas industry by the invention of the gas incandescent lamp. Then came the natural reaction. The gas people pushed their new lamp to the utmost, and it is no exaggeration to say that electric central station engineers in this country as well as in Europe got scared. It became evident that electric lighting, to be able to compete successfully, must be cheapened. New and more efficient electric lamps had to be invented. This is the other reaction which has set in during the last years and which is now proceeding with a surprisingly high reaction velocity.

* * *

It is again the chemical engineer who does the work. Dr. Auer von Welsbach has again come to the front with his osmium lamp; Dr. von Bolton and his chemical coworkers of the Siemens & Halske Co., with the tantalum lamp; Dr. Whitney, of the research laboratory of the General Electric Co., with the graphitized filament lamp, and still more recently Dr. Kuzel and others with the tungsten lamp. Then we hear of a zirconium lamp, and surely there will be others. A peculiar incident of this development is the reaction of the chemical research work in rare elements on both the steel industry and the electric lighting industry. In the steel industry a revolution has been brought about by the introduction of high-speed tool-steels, and this was made possible by the introduction of ferro-alloys. It is important to have the ferro-alloys as pure as possible, with as little carbon as possible, in order not to introduce any undesirable elements into the steel. For many purposes it is preferable to introduce the element with which the iron is to be alloyed not in its elemental form, but as a ferro, on account of the decreased melting point. For a filament of an electric incandescent lamp it is even more important to have the greatest possible purity; here the melting point should be as high as possible, since the higher the temperature at which the lamp is operated the higher its efficiency. As a general proposition, it will be important that the filament shall be in the purest elemental form, since any small addition of an impurity considerably decreases the melting point. The enormous amount of research work on rare elements which has recently been done to further the above mentioned commercial ends, has made it clear how little we have known before of the physics and chemistry of these elements. Thus von Bolton claims that nobody before him ever produced pure tantalum; he seems to be backed up in this by the United States Patent Office, if we consider the extremely broad claims which have been allowed to him, as mentioned in the Analysis of Current Electrochemical Patents in our present issue.

* * *

In connection with the latest comer, the Kuzel lamp, for which an exceedingly high efficiency (1 candle-power per watt, and even better) is claimed, one special point is of particular interest. This is that Dr. Kuzel brings the metal first into the colloidal state, then puts it into filament form and dries it. The filament is then a conductor of the second class (an electrolytic conductor). By electrically heating it, the nature of the filament changes; it becomes a conductor of the first class (a metallic conductor). The filament is then ready for use. Thus the study of colloids—which has been considered by many a very interesting, but exceedingly theoretical subject, barren of practical applications—has led in this case to a very promising and commercially important result. A few words may be added on arc lamps. In this field, too, it is chemical research work which has given us a new lamp—the "flaming arc lamp." Its feature is the impregnation of the carbons with certain chemicals. The evaporation of the latter results in the production of an artificial, highly ionized atmosphere. Light is given off not only from the terminals of the carbons, but also from the arc itself, which can be drawn out much longer than in ordinary arc lamps. This results in a higher efficiency and in the possibility of producing any desired color effects. Taken as a whole, the progress already made in the invention of new electric lamps is very gratifying, but even greater progress may be confidently expected in the near future.

New Developments at Broken Hill.

The great mines of the Barrier Reef at Broken Hill, Australia, form one of the greatest and richest bodies of ore known to the mining world. The deposit is comparable in size and general uniformity to the celebrated Rio Tinto copper mine of Spain. The typical ore assays about 16 per cent lead, 11 oz. silver, and 15 per cent zinc, and is of the character described as a "complex sulphide." For years this has been worked only for the recovery of the argentiferous galena by careful wet concentration, and the zinc values have accumulated in vast heaps of tailing to an aggregate tonnage of 6,000,000 tons. Such an amount of potential mineral value has been the means of stimulation to much metallurgical experimentation in the past twenty years, but no real commercial process was developed until recently. Magnetic separation has been only moderately successful, and it is to the "acid flotation" process that can be ascribed the final unlocking of this stored mineral wealth.

* * *

The theory of this process has been discussed in our columns recently, but these theories are but inadequate explanations of the facts. Both the "Potter" process and the "Delprat" process are based simply on actual experimental groundwork. These processes both produce a zinc concentrate of fair grade (40 to 45 per cent zinc), with small recoverable values of lead and silver. The several companies preparing to produce the zinc concentrate have contracted with large companies to absorb this tonnage at their plants on a sliding scale basis, which we described in a late editorial. Such an ar-

rangement is eminently fair and satisfactory to each contracting party. The losses in the wet concentration and the following "flotation" concentration are high, but as this is compensated for by the low cost of mining and of the combined treatment, the result is a very nice margin of profit on the zinc value of the original ore.

* * *

The total reserves, both in ore and tailings, foot up to the enormous gross value of, in round numbers, seven hundred million dollars, distributed as follows: Lead, $200,000,000; silver, $100,000,000; zinc, $400,000,000. Of course, these figures are subject to deductions for mining and treatment costs and losses and market fluctuations, so that the probable net profit is only a fraction of these astounding figures, but still large enough. Market fluctuations will be controlled to some extent by a policy of regulating supply of ore to suit increasing demands for metal. Such a course is wise and conservative business. We can look to these great mines in a way as a balance wheel in the markets for these three metals in Europe to absorb irregularities in the trade. In conclusion, it should be stated that the innovation in the metallurgical treatment which has so expanded the future of Broken Hill as a zinc producer was due originally to an observation of the floating of the sulphide particles in a beaker during a chemical analysis. So much for the fecundity of an idea.

Industrial Development Increasing the Productivity of Capital.

The definitions of political economists often vary with the individual thinker. With regard to their great element, "capital," there is, however, little disagreement. Capital is stored up wealth, available for man's economic needs; and interest is in a measure the average and normal rate of the growth of capital. There is a natural correction for insurance or the cost of protecting capital against economic decrease. And the net interest return sometimes is negative, as in the case of silverware stored in a vault, where productivity is nil and cost of protection considerable. It can thus be laid down as an axiom that any general tendency which increases the productivity of capital increases the rate of interest. Naturally, in a new country as was the West of the United States forty years ago, the interest rate was high, because the fertile prairies were being turned into farm lands by the influx of settlers. In Central America to-day the productive tropical sun does the work while man is so indolent that the natural resources are only touched. Therefore, 20 per cent per year is considered not at all usurious in Guatemala.

* * *

At the present boom period we perceive a gradual increase in rate of interest and a great increase in market price of that class of securities which have the final marginal share in the equity, as the common stocks of railroads and "industrials." The simplest and boldest explanation of this is to ascribe it to the great wave of prosperity that is now sweeping over North America. Just as England in the past century slowly and surely forged ahead to pre-eminence in commerce, so does America seem now to be undergoing lasting industrial growth. But a deep and critical search into the basic cause of this action would lead us to believe that it is the peculiar combination of native adroitness in changing [illegible] of the American type with the good natured [illegible] to the progress of applied science. The [illegible] of the telephone and telegraph and [illegible] ties all reduce "industrial friction" to a decreasing minimum. The widespread growth of technical engineering [illegible] the embryonic engineer and engineering societies for the older men, diffuse accurate knowledge of [illegible]. And likewise the wonderful rise of the technical and [illegible] press is having a profound effect, greater [illegible] of us imagine.

* * *

New developments in engineering create unlooked for values of the natural resources. For instance, the largest item of the fundamental wealth of the United States Steel Corporation is its holdings of Mesabi ore lands. Due to the physical fineness of these ores it was impossible in the early "nineties" to sell these ores to any blast furnace, and until as late as 1897 these lands possessed but little cash value in spite of high iron content and other valuable chemical properties. However, due to a gradual increase in the practical knowledge how to treat these ores, to which hundreds of individuals each has contributed his increment, these ores have assumed an enormous value, about the "psychological moment" when the great iron and steel merger took place. In the case of non-Bessemer or phosphatic ores the change was a much more gradual one, covering a period of nearly two decades in this country. In the copper industry the change has been very marked. One large Western mine some twenty years ago was throwing away a slag analysing 12 per cent copper. Now its average of ore mined is less than one-quarter of this. The immense value of phosphatic rocks in the South is all due to the application of a simple chemical reaction—of sulphuric acid rendering native phosphate available to plant growth. The electrolytic refining of copper, the other various electrochemical industries of aluminium, calcium carbide, carborundum, etc., have all created new national assets. The cyaniding of gold ores, the improvements in lead smelting, in the zinc industry and in the metallurgy of minor metals have had a like effect. By-product coke ovens, electrical transmission of power, high-speed tool-steel, oil refining, the packing of meats, agriculture, simply continue the list of industries in which the widespread use of modern methods and application of scientific facts have created values by the adherence to the ideal.

* * *

Though there are countless failures to chronicle, yet the sum of this all is progress. And best of all, due to modern ways of diffusing correct principles, this progress is made more efficiently now than before. In the future, we can expect greater advance, some few startling, but oftener a simple and sure evolution resulting in operations done on a large and diverse scale, but with greater usefulness to the [illegible]. As a result of this tendency, we can expect the rate of interest to measure this productivity of capital more slowly increasing normal rate, with only slight waves of recession until some great counter current nullifies this current. But the all-pervading advance of scientific engineering and [illegible] commercialism does but render the future bright. And in the United States the consistent [illegible] run.

Faraday Society.

The twentieth ordinary meeting of the Faraday Society was held on Tuesday, May 15, 1906, Dr. F. Mollwo Perkin, treasurer, being in the chair.

Platinized Platinum Electrodes.

Mr. H. D. Law read a paper entitled "Behavior of Platinized Electrodes." The author desired to find an electrode on which the reduction of the aromatic-aldehydes and similar easily reducible compounds could not be effected. Platinized platinum, as being the metal from which hydrogen is liberated at the lowest potential, was tried as the cathode in an acidified alcoholic solution of benzaldehyde. At first, energetic reduction took place; the activity of this, however, diminished in successive experiments, and was extremely small after 12 hours' polarization. The cause of the reaction is obscure; it is probably catalytic in its origin.

Dr. N. T. M. Wilsmore drew attention to Tafel's work, which the author's results appeared to contradict.

Dr. F. Mollwo Perkin remarked on the differences between lead and platinum electrodes, and discussed the general bearing of the author's experiments.

Electrolysis of Fused Zinc Chloride.

Mr. Julius L. F. Vogel read a paper on "The Electrolysis of Fused Zinc Chloride in Cells Heated Externally." The author begins by explaining that the work which formed the subject of the paper was carried out for the most part between 1898 and 1901, and could be conveniently divided into three periods. He then explains how the investigation came to be undertaken; the original inception of the process was due to the late F. Maxwell Lyte, who took out patents for a complex ore process involving the electrolysis of fused zinc chloride and for the dehydration of zinc chloride. The process as such was never investigated practically. The second period deals with the investigation of the dehydration of zinc chloride by evaporating under reduced pressure, and the electrolysis of the salt in a fused state in externally heated cells by Dr. O. J. Steinhart and the author jointly on behalf of the Smelting Corporation, Ltd. The various experiments and the apparatus employed for these experiments are described. The third period deals with the further investigations made after the United Alkali Co. had joined the Smelting Corporation in testing the process, and details are given of the work as carried out under the joint supervision of the author's firm and the chemical staff of the United Alkali Co. The author describes how the process was carried successfully to a stage when continuous electrolysis was carried on for eleven days and nights, and three cwt. of pure zinc was produced.

He then explains how, on the failure of the Smelting Corporation, the work was suspended, and finally abandoned, although further elaborate investigations were undertaken by the United Alkali Co. utilizing cells heated internally by the current.

The author finally reviews the work done in cells heated externally, and expresses his opinion that the process was "non-proven," and that it still offers possibilities of being carried to commercial success.

The paper is illustrated with sectional drawings of apparatus and lantern slides, prepared from photographs of the actual plant, were exhibited.

Dr. O. J. Steinhart referred to the production of zinc from zinc oxide by feeding the latter into a molten bath of chloride, after the fashion of aluminium electrolysis.

Mr. W. Murray Morrison mentioned some of the advantages of internal heating with regard to local inequalities of temperature.

Dr. H. Borns referred to the formation of zinc clouds diffused in the electrolyte under certain conditions, and consequent reduction of efficiency observed by Lorenz. He asked whether the addition of impurities, such as ammonium chloride, had been tried. Evaporating with HCl was a method of dehydrating the chloride.

Mr. L. Gaster asked for information regarding comparative energy losses in externally and internally heated cells, and also regarding the consumption of carbon.

Dr. G. Rudorf (communicated), after reviewing the various methods proposed for dehydrating the chloride, affirmed that the author's was the only practicable one. He asked for information regarding the effect on the enamel of continuous heating with zinc chloride; his experience was that a perfect enamel was not obtainable.

Mr. J. L. F. Vogel, in reply, said that the use of a fused metallic cathode and proper temperature and other conditions prevented the formation of zinc cloud. Carbon losses depended greatly on the dehydration; they had not had opportunities of trying Acheson graphite. It was undesirable to add foreign substances, as these would accumulate in time; it was likewise impracticable to dehydrate in the fashion referred to by Dr. Borns. The difficulty in electrolyzing a solution of ZnO in fused chloride was that the solubility was slight, and at the high-current densities demanded by practice not sufficient oxide would pass into solution.

American Institute of Electrical Engineers.

The twenty-third annual convention of the American Institute of Electrical Engineers was held in Milwaukee, Wis., from May 28 to 31, and was a full success with respect to attendance (which was about 230), and the number and quality of papers presented, and was highly enjoyable in its social functions.

The president, Dr. S. S. Wheeler, who will retire in autumn, took occasion to introduce to the convention the new president-elect, Dr. Samuel Sheldon, of Brooklyn, N. Y.

Among the numerous papers presented there was one by President Wheeler, who discussed the topic, "Engineering Ethics." Electrical engineers are professional men, and as such they are under moral obligations to the public, to their clients and to the engineering society. This subject was discussed along broad lines, and later on a motion of Dr. C. P. Steinmetz was adopted that the Institute, through its president and board of directors, shall formulate a code of ethics to the membership for approval.

A paper by Prof. D. C. Jackson dealt with economies derivable from the use of relatively small water powers of low head in the Middle West.

A very interesting paper by Mr. E. F. Northrup discussed the measurement of temperatures by electrical means; we will give a longer abstract of this paper in our next issue.

Magnetic Properties of Electrolytic Iron.

A paper which will certainly appeal strongly to our readers dealt with the magnetic properties of electrolytic iron, and had as authors Prof. C. F. Burgess, of the University of Wisconsin, and Mr. A. H. Taylor.

In the introduction the authors remarked that they have made marked improvement recently in the purity of electrolytic iron. Two analyses were given. One sample contained no S, 0.013 Si, 0.004 P, no Mn, 0.012 C, 0.072 H; the other sample 0.001 S, 0.003 Si, 0.020 P, no Mn, 0.033 C, 0.083 H.

The electrolytic iron was made, of course, by the method of Burgess, which was described in our Vol. II., p. 183. The chief impurity in iron is hydrogen, and it has often been thought that this content of hydrogen is responsible for some of the peculiar properties of the iron, but the authors are not sure that there really exists a relationship between the hardness of the iron and the hydrogen content.

The authors are now making an extended investigation under a grant from the Carnegie Institution, of Washington, concerning the magnetic properties of electrolytic iron as

altered by heat teatment. The purpose of the investigation is to establish the relationship between coercive force, permeability, hysteresis constants and the temperature at which the iron is heated. The paper presented at Milwaukee is in form of a preliminary report.

Some curves were shown giving the results of the measurements. The authors first tested electrolytic iron which had not been subjected to heat treatment. The iron is evidently very hard; the coercive force is 18 dynes and the retentivity 10,000. It seems that a field of 210 dynes does not saturate the iron, although it carries the induction to the high value of 21,250. At the same time there is relatively little area added to the hysteresis loop by running the field so high.

The test ring was then heated for several hours at a temperature of 200° C., and magnetic tests were again made. They gave a curve almost identical with the first.

The ring was then unwound, embedded in magnesium oxide and heated for 8 hours at about 1,200° C. The ring was found to be much softer than at first, and on being tested with the step-by-step method, gave values for the magnetic induction 17 per cent lower than those obtained by the method of reversals. This difference was traced to magnetic viscosity, and a modified ballistic method was devised to eliminate the error.

The new curve obtained shows that a tremendous change has taken place in the iron with heating at 1,200° C.

The iron is now in the condition of a rather soft steel, with a coercive force of about 2.5 dynes, a retentivity of 12,500, and a large amount of magnetic viscosity in the steep parts of the curve. A second heating of the ring to over 1,200° C. produced no appreciable change in the magnetic properties.

The paper elicited a lively discussion, in the course of which Dr. C. P. Steinmetz stated that thirteen years ago, during some investigations concerning the permeability of pure iron, he had found a mixture of iron deposited by means of a mercury electrode, the mercury subsequently being boiled out, had a permeability of about 2. He stated that during recent years a considerable change had developed with reference to the conceptions of magnetization. Many of the materials which had previously been considered non-magnetic were now known to be magnetic to a small degree, and many materials which are non-magnetic in themselves are found to be magnetic when combined with other so-called non-magnetic materials. Thus a mixture of 60 per cent of copper, 20 per cent of aluminium and 20 per cent of manganese possesses high permeability although it contains no magnetic materials. The permeability of these materials is effected by heat in a manner essentially different from that which is true with reference to iron. The alloy above referred to when heated and cooled quickly is found to be very magnetic, having a permeability about equal to that of nickel. A powdered mixture of antimony and manganese in portions of 3 to 1 will show the lines of magnetic force in the neighborhood of a magnet. Alloys of manganese and zinc have been found to be quite magnetic. Experiments with the thermit process have shown that manganese is magnetic when combined with certain other materials. He expresses the opinion that it is not impossible to find certain alloys possessing permeabilities greater than that of iron, the importance of which to the electrical industries is extremely great.

Recent Practice in the Treatment of Tin-Tungsten-Copper Ores.

(*Concluded from page 168.*)

Each of the five distributing boxes in front of the Lührig vanners has two compartments, each of which supplies one-half of a double vanner. Four of these boxes take the original pulp from the large spitzkasten, whilst the fifth takes the middlings from the other four vanners, so as to allow of a separate retreatment of the middlings. The middlings from the vanners are collected and pumped by a [illegible] small condensing spitzkasten, [illegible] and the bottom discharge of this spitzkasten passes to a distributing box from which each compartment feeds one half of a double vanner.

The heads from the vanners are a high-grade concentrate, and go to the calciner, where they are roasted [illegible] the coarser concentrates of the Buss tables. This arrangement has been found to save some reclassifying of the fine concentrates, and secures a better market for the coarse products. The concentrates from the Buss tables are [illegible] free from any silicious waste, and contain only the sulphides of iron, copper and arsenic, the oxides of tin and tungsten. They are of a very much higher grade of concentration than the average Cornish concentrates called "witts," and are separately roasted and then electromagnetically treated. The vanner concentrates, which contain much of the tin oxide originally in the ore, are of lower grade as compared with those from the Buss tables, and are, therefore, prior to their electromagnetic treatment, reclassified and enriched in ordinary convex Cornish buddles, in which the bulk of the tin oxides are removed and go to waste.

This enrichment by buddling would be avoided if a second magnetic apparatus were at the disposal of the company; but as the capacity of the magnetic separator is only about [illegible] per 24 hours, the enrichment of a portion of the concentrates becomes necessary in order to enable the magnetic separator to deal with the whole of the concentrates. The vanner concentrates are, after enrichment, dried in a furnace of the reverberatory type. This drying is necessary, as the Wetherill magnetic separator requires the material to be in a very dry form. Formerly a drying furnace was used, having a bed of cast-iron plates on which the concentrates were spread, the plates being heated by a fire underneath, but the capacity of this open furnace is very much smaller than that of the closed reverberatory type. A revolving tube furnace of the Bruckner type, with provisions for catching the dust, would no doubt be the best furnace for drying purposes

The milling and dressing costs during June last for 2,540 tons of ore crushed amounted to 69.2 cents per ton, a low figure, due to the extensive use of labor-saving machinery, but which the author expects to see brought to a lower figure through the use of a magnetic separator. In his concluding pages the author attacks Cornish metallurgical methods in general. He points out that "the success of the mill as regards high percentage (the loss being only 11.4 per cent) recovery at low cost upon a comparatively low-grade ore, is principally due to the introduction of a thorough system of classification and to the use of side discharge tables, which produce middle products, and as regards the separation of the denser minerals the success is especially due to the inclusion of electromagnetic separation in the scheme. Notwithstanding the work done in a few mines during recent years as a general rule, classification has not received the attention it deserves in Cornwall; indeed, many of the Cornish tin dressers and mine managers regard it as an altogether useless incumbrance, and at only a few tin mines in the county has any attempt at classification even been made.

"In addition to classification, another important point was overlooked at the time when Frue vanners were introduced at the Cornish mines, *i. e.*, graduated crushing. Tin usually occurs in a very fine state of division in the Cornish ores, and therefore requires fairly fine crushing; but the reason why the whole of the ore should be pounded into slimes to start with is difficult to understand, though upon most Cornish dressing floors this is done.

"Slimes are the bugbear of the dresser and [illegible] is always to be deprecated, yet little care is taken in Cornwall to reduce the proportion of slimes in the [illegible] cases everything seems to have been done to increase the proportion. Punched stamped grates are still strenuously de-

fended by the Cornish dresser, although owing to the unduly large extent of the blind places as compared with the open spaces in the grate there is an area entirely out of all proportion, and one that gives an abnormally small discharge. Cornish men are conservative in the extreme, and love to adhere to old methods. When abroad they are more open to modern methods and machinery, but at home they are slow in introducing them. The mines have often low-grade ores, and, therefore, especially require appliances for saving time and labor. The author firmly believes that with modern methods and machinery many Cornish mines could be profitably worked, and when this has once been realized the influx of capital to the county will be stimulated, and Cornwall, with its undoubtedly great mineral wealth, will once more regain its old prosperity."

The paper being a lengthy and contentious one, elicited considerable discussion. Indignant repudiation of incompetence was the keynote of many of the speeches. For instance, Mr. Arthur Thomas, of the Dolcoath mine, sent a caustic and lengthy communication, which called into question the accuracy of the vanning-shovel method of computation. It was further pointed out "that practice has disproved the theory that classification is essential for concentration on the vanner, and the custom obtaining is thoroughly justified by the excellent concentration resulting. Evidently the author is satisfied with the treatment peculiarly adapted such a singular ore as his own, but it would be unwise to generalize from this one instance. The balance of the evidence of practice, certainly against the theory of classification in every instance previous to concentration on vanners, and those who hold that classification is essential to close concentration, can certainly not have had an opportunity of studying all the facts under varying conditions of working. The author could not be taken seriously in respect to the unwarrantable assertion that 'why the whole of the ore is pounded into slimes to start with, is difficult to understand, though upon most Cornish dressing floors this is done.' This is not in accordance with fact, and a visit to most of the Cornish dressing floors will convince him that no such lamentable ignorance exists.

"Such strictures in this and other respects on the practice obtaining generally in the mines of Cornwall, are unjustified, and exhibit a lack of knowledge and appreciation of the circumstances."

The writer said he did not hold a brief for Cornishmen, but he suggested that perhaps the conservatism alleged might indicate more knowledge and proper appreciation of old methods and their present-day usefulness in justified circumstances than is generally supposed; but Cornishmen generally would no doubt be relieved to hear that there is some chance for them when they go abroad.

Mr. Walter MacDermott followed in very much the same vein. He thought that Mr. Dietzsch was speaking without much experience of what had been done beforehand. He had great sympathy with the author's conclusions, because he himself made the very same mistake more than twenty years ago, and not only he thought the same, but he wrote the same. But facts were things which must take precedence of any theory, and the almost universal experience of men in the mining world in connection with material that had been stamped to a grade sufficiently fine for a Frue vanner to begin with was that classification had no advantage. Now, that was a serious difference to be raised between two authorities, or two writers, one stating that it was the practice of a large number of men for a great many years, on various ores all over the world. The theory at the need for classification had frequently cropped up. It was the natural outcome of some of the older books on concentration. He remembered himself, in writing his first description of the Frue vanner and its work, with the line of Burgess in his heart, that he made the statement that the better the classification, the better would be the work of the Frue vanner. But it was not so, and there must be some failure in a theory which did not agree with the results of practice. He had long ago determined to his own satisfaction that there were elements in the working of the concentrator which made it very difficult to theories with regard to the best results that could be obtained. The elements of inertia, of momentum, of flowing water, of adhesion to a surface, were all elements which might be individually stated; but the combined effect of which depended on the motion and the construction of the machine.

Mr. E. H. Davies considered that there was evidence of considerable loss, and that he believed the costs of dressing to be greatly in excess of that recorded by the author. For his part he agreed with Mr. Dietzsch, that with modern methods and machinery many Cornish mines might regain their prosperity. Mr. J. G. Green stated that if he were now going to erect a mill he would install the closest possible classification or sizing before attempting to get out the concentrates. Other speakers, such as Dr. Rose, Prof. Bauerman and Prof. Gowland, discussed the details of the roasting of the ore. Mr. D. A. Louis regarded classification as not always indispensable—at Dolcoath it was unnecessary, but with complex ores there was little doubt that close classification was of importance.

After further discussion Mr. H. C. Jenkins replied briefly, on behalf of the author, to several of the points raised, leaving Mr. Dietzsch, who was then in Burmah, to reply in writing to the discussion as a whole.

CORRESPONDENCE.

Kryptol.

To the Editor of Electrochemical and Metallurgical Industry:

SIR:—In the ELECTROCHEMICAL AND METALLURGICAL INDUSTRY of April, 1906, page 148, you give a synopsis of an article by J. Bronn on "Kryptol," and it is said that this "is not simply granulated carbon, but a mixture of carbon, graphite, sand, clay and other silicate, carborundum, etc."

In 1904 we determined the ash in a sample of kryptol and found that it contained 0.24 per cent (ELECTROCHEMICAL AND METALLURGICAL INDUSTRY, January, 1905, p. 5). Recently another sample weighing 10 grams was used for an ash determination and found to contain 0.64 per cent ash. The ash was then treated with hydrofluoric and sulphuric acid to remove silica and get iron, aluminium, etc., into solution. The insoluble matter that remained was then examined carefully under the microscope, and apparently consisted of a little unburnt graphite and some silica or silicates that had escaped the attack of the hydrofluoric acid. No carborundum was positively detected, although when magnified 450 diameters one crystal that looked something like carborundum was observed. However, there can be no doubt that any mere trace of carborundum which might have been present could have no effect on the kryptol as a resistor material. Any specimens we have had at our laboratory may be considered simply as a mixture of amorphous and graphitic carbon. Apparently therefore, the kryptol described by Mr. Bronn differs from that which is offered for sale in this country under that trade name. It would be interesting to learn where kryptol of the composition mentioned by Mr. Bronn may be purchased.

The use of a mixture of carbon and carborundum would be interesting, but the very large thermal coefficient which characterizes the resistivity of carborundum might prove troublesome.

As to the admixture of silica and silicates in a granular carbon resistor, these would prove very objectionable after a certain temperature was reached, this temperature being that at which reactions occur between carbon and these compounds, thus introducing a permanent change in the resistivity of the material.

FRANCIS X. J. FITZGERALD

Fitzgerald & Bennie Laboratories,

Niagara Falls, N. Y., May 4.

CORNELL MEETING OF THE AMERICAN ELECTROCHEMICAL SOCIETY.

The ninth general meeting of the American Electrochemical Society was held at Ithaca, N. Y., on the first three days of May. Since we comment on the general features of this highly pleasant and successful meeting elsewhere in this issue, we will go in this report at once *in medias res*.

TUESDAY SESSION.

The first session was called to order on the morning of May 1 in the lecture room of the Chemical Laboratory (Morse Hall) by President Bancroft, who introduced the distinguished president of Cornell University, Dr. J. G. Schurmann.

In his graceful speech of welcome Dr. Schurmann pointed out that the founder of Cornell University had always insisted on the double purpose of the University: to discover and teach the truth with regard to Nature, and to apply this truth to the improvement of man's conditions. Since the American Electrochemical Society endeavors to fulfill the same double purpose, Dr. Schurmann heartily welcomed its members, and said they would surely feel at home in the atmosphere of Cornell.

President Bancroft expressed to President Schurmann the thanks of the Society, and a short business meeting followed. The reports of the secretary and treasurer will be printed as circulars.

Secretary Sadtler read the report of the tellers on the results of the recent election, which are as follows:

President (for one year), Mr. Carl Hering, Philadelphia, Pa.

Vice-Presidents (for two years), Col. Samuel Reber, Washington, D. C.; Dr. J. W. Richards, Bethlehem, Pa.; Dr. S. P. Sadtler, Philadelphia, Pa.

Managers (for three years), Mr. E. G. Acheson, Niagara Falls, N. Y.; Prof. C. F. Burgess, Madison, Wis.; Mr. C. J. Reed, Philadelphia, Pa.

Secretary (for one year), Mr. S. S. Sadtler, 39 South Tenth Street, Philadelphia, Pa.

Treasurer (for one year), Mr. P. G. Salom, Philadelphia, Pa.

Some other business announcements were made during the following sessions. The Frenzel prize for a paper on the production and uses of rare metals has been awarded to Mr. G. Gin, of Paris, France. Mr. Gin's voluminous memoir (in French) will be kept in the archives of the Society.

The first grant of $100 from the Society's funds for special research work has been awarded to Dr. A. T. Lincoln, of the University of Illinois, for an investigation of the electrolytic corrosion of brass and special bronzes. Dr. Lincoln will report on the results of his research at the next annual meeting in 1907.

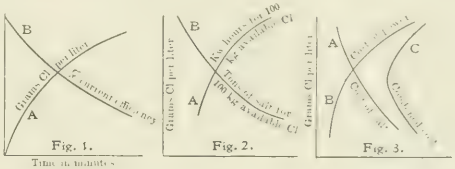
The question whether one or two meetings shall be held each year has not yet been definitely settled. It seems that the majority of the members favor a compromise by which the main annual meeting is to be held in Spring, at a place to be determined upon each year, while a second and secondary meeting shall be held in autumn in New York City. The idea is that many members, not residing in New York, visit New York regularly in autumn for professional reasons. These, together with the large resident membership of New York, would insure a full success, while, on the other hand, such a meeting would not constitute too much of a burden on the time and the pocketbook of the members. A clause in a motion to that effect stipulates that no special entertaining shall be done by the New York members in connection with this meeting. If this arrangement is finally adopted, two volumes of Transactions will be issued as heretofore.

AN INSTRUCTIVE LABORATORY EXPERIMENT IN APPLIED ELECTROCHEMISTRY.

The first paper was presented by Prof. WM. H. WALKER, of the Massachusetts Institute of Technology, who pointed out that it is one of the most important functions of a laboratory experiment in applied chemistry, as distinguished from an experiment in general or pure chemistry, to enable the students to acquire an economic point of view. It is only by connecting the data obtained by experiments with the important economic considerations involved in the problem, that the experiment becomes an industrial one.

The author illustrated these views in detail in discussing a special problem which meets all requirements more or less perfectly. This is the preparation of bleaching liquor by the electrolysis of brine without a diaphragm. (Concerning commercial apparatus in this line of work, see the article by Dr. Walker in our Vol. I., p. 439.) The raw material is ordinary salt of known purity, and while the apparatus may take many forms, the following has been found convenient:

An Acheson graphite crucible, holding two liters, is saturated with paraffine, and made to serve as the container and also the anode. This is set in a cooling bath supplied with running water. The cathode is an inverted U (∩) shaped piece of hard rubber, upon the outside edge of which is placed



FIGS. 1, 2, 3.—ELECTROLYTIC PRODUCTION OF HYPOCHLORITE.

a piece of copper wire, forming the cathode. The wires from the two limbs of the ∩ are united at the top, and by these the rubber frame is suspended from a bearing, such as a bicycle hub. By means of this bearing the cathode may be rotated, and the solution continually agitated. While the current density at the anode may be changed only by varying the current or the volume of solution in the crucible, the density at the cathode may be changed at will by substituting a different wire or by soldering on a metal plate. A current of 20 to 25 amps. at 6 volts is led into the cell through a mercury cup in the end of the bearing supporting the cathode, and back to the generator through an ammeter. Across the cell is connected a small voltmeter. At intervals of 10 minutes for the first half-hour, and after that each 20 minutes, a sample of the electrolyte is drawn from the cell by means of a 10 c.c. pipette, and analyzed for available chlorine by titration with tenth-normal sodium arsenite solution. At the same time the ammeter and the voltmeter are read and the temperature of the brine observed.

When the content of the chlorine ceases to appreciably increase, the experiment is interrupted and experimental data plotted, as shown by curves *A* and *B* in Fig. 1. As abscissæ are shown, the intervals of time elapsed from the beginning of the electrolysis, as ordinates in curve *A*, the grams available chlorine per liter, and in curve *B* the average current efficiency which has obtained from the start of the experiment up to the corresponding time. The calculation may be simplified and re-

considerable error introduced if the average of the run be taken with respect to the volume, the current strength and the fall of potential across the cell.

In practical cotton bleaching, this liquor, however strong in available chlorine it may have originally been made, is diluted to about 3 grams per liter. When the available chlorine is exhausted by the cotton the residual liquor containing the salt is discarded. Salt will be economized by continuing the electrolysis as long as any hypochlorite is being formed. On the other hand, the hypochlorite thus formed may, on account of the low efficiency obtaining toward the end of the experiment, be costing more in power than is being saved in salt. The question then arises: for a given strength of brine, assume 200 grams NaCl per liter, and for a given quantity of available chlorine assume 100 kg., at what concentration of hypochlorite is it most economical to stop? At what point does the cost of electricity consumed overbalance the saving made in salt consumed?

To determine the relation of the decrease in the cost of salt as the content of hypochlorite is increased, to the increase in the cost of power under the same conditions, for a given weight of hypochlorite, the curves of Fig. 2 are drawn. From the electrochemical equivalent and the average voltage across the cell, the energy in kilowatt-hours is calculated, assuming that the process is carried out at 100 per cent efficiency. From this figure the energy necessary to produce 100 kg. available chlorine, at the efficiency obtaining when the solution contained from 1 to 10 grams available chlorine per liter is obtained, and is represented in curve *A*. The volume of brine, and hence the weight of salt necessary to make 100 kg. available chlorine at concentrations from 1 up to 10 grams per liter, may be easily calculated. This is given in metric tons in curve *B*.

We are now in position to determine the most economical concentration of available chlorine at any particular place, if we know the cost of salt and power prevailing there. For example, assume that salt can be bought for $5 per ton, and power for 1¼ cents per kilowatt-hour. The cost of salt necessary to make 100 kg. available chlorine in the form of bleaching liquor at several different concentrations is given by curve *A*, Fig. 3. The cost of power is represented by curve *B*. By adding the abscissæ for any one concentration we obtain the combined cost for the two factors, which is shown by curve *C*, with a minimum at $36.80.

In discussing the problem with the class before beginning the experiment, a number of questions will arise. For example, assuming a constant cost for salt and power, what is the most economical concentration of brine to use? From theoretical considerations we know that the resistance, and hence the voltage necessary, will fall as the solution becomes more concentrated; this will decrease the cost of power but increase that of salt. The reduction at the cathode will be proportionately less as the relative number of Na and Cl ions increases; this will tend to increase the current efficiency, thus decreasing the cost of power, while the consumption of salt is made greater. Just what this relationship is, can be determined by having a certain number of students work at each concentration of salt, from 50 grams per liter, in steps of from 50 to 350 grams.

By assembling the resultant curves in one plot on the blackboard, the results are available for the entire class.

In the same way may be discussed and made the subject of investigation by members of the class for the benefit of all, the question of the relationship of current density to the cost of the process. It can be shown theoretically that to increase the size of the cathode, for example, decreases the resistance, and this lowers the energy consumed. But, on the other hand, it increases the amount of reduction of hypochlorite at the cathode and hence decreases the current efficiency of the process, which increases the power necessary.

In the concluding part of the paper, Dr. Walker gave some analytical considerations of Mr. Warren K. Lewis, who has reduced the curves of the above three figures to mathematical equations so that the concentration of available chlorine, which corresponds to the minimum cost of power and salt, may be calculated from a simple formula.

In the discussion which followed, Dr. J. W. Richards agreed with Dr. Walker that it was an excellent scheme to divide a class into groups working on the same problem under different conditions. Mr. Carl Hering suggested to reverse the abscissas and ordinates in Fig. 3, since the minimum total cost in Fig. 3 would then correspond to the lowest value of the ordinate of the curve. Dr. Richards said that the curve would then directly illustrate the expressions, high cost and low cost.

Some Principles of Resistance Furnace Design.

A very long and very useful paper on this subject was presented by Mr. Clarence L. Collens, 2d, of Niagara Falls.

The discussion is restricted to intermittent resistor furnaces in which the net chemical reaction, if there is any, is endothermic, and in which the heat-generating body or resistor is separate and distinct from the materials to be acted upon. The flow of electricity is assumed to be limited to the resistor itself. The heat is assumed to be diffused principally by conduction.

If one constructs graphically the system of isothermal surfaces and that of the lines of flow of heat (perpendicular to the isothermals), one gets a concise picture of the heat diffusion within the furnace. The isothermal of highest temperature is somewhere within the resistor and is represented by some neutral point, line or surface, from which emanate the lines along which all the heat generated flows outward through the furnace.

The desired reaction or effect must usually be carried on between definite limits of temperature; the minimum temperature may be designated t_1, the maximum temperature t_2. (The minimum temperature t_1 is evidently the temperature at which the desired chemical reaction starts. The maximum temperature t_2 is somewhat arbitrary.)

Those particles of the material under treatment which are next to the resistor are at the highest temperature. The surface separating the resistor and the material under treatment is called the "surface of productive heat-diffusion." Going further outwards, away from the resistor, the temperature gradually decreases until we come to an isothermal surface at which the temperature is just the minimum temperature t_1 required for the reaction. This surface is called "the surface of non-productive heat diffusion," because any heat passing through this surface to the outside is wasted, since the temperature beyond this zone is no longer sufficient to produce the desired reaction. The change of the raw material M into the desired product P, therefore, goes on within the zone between the surface of productive heat diffusion and the surface of non-productive heat diffusion.

The second part of Mr. Collens' paper deals with an analysis of energy consumption within the furnace. Energy is first consumed to heat (to t_1) and superheat the resistor. The total balance of energy consumed passes through the surface of productive heat diffusion into the charge material M, and is consumed there in four different ways. (1) To heat the material M to the temperature t_1 at which the reaction starts; (2) to produce the chemical conversion of the raw material M into the product P at the temperature t_1; (3) to superheat the product P next to the resistor; (4) to heat the material M beyond the area of non-productive heat diffusion. This fourth part of energy is wasted in so far as it does not serve to change the material M into the product P. In Mr. Collens' paper the analytical expressions are given for the different amounts of energy as above defined. The formulas look quite complicated, since it is necessary—even for the most fundamental expressions—to make free use of the calculus

The third part of Mr. Collens' paper gives a mathematical discussion of the rate of temperature change. In intermittent resistor furnaces we have not a steady stationary flow of heat, but the temperature of each part is continually more or less changing. The temperature changes with the distance (x) away from the surface of productive heat diffusion, and, if the fixed stationary state is not reached during the entire run, it also varies with the time or length of run; hence the temperature t at any point is a function of both the distance x and the time. The author's analysis shows that the rate of temperature change with respect to the time is greater the higher the specific thermal conductivity of the materials, the lower their density, the lower their capacity for heat, the greater the rate of change of the rate of change of the temperature between successive points, and the less divergent the lines of flow of heat.

In actual practice the general shape of the isothermal surfaces after a given interval of time can usually be determined very accurately by certain definite indications within the materials under treatment, such as the point where a reaction commences or ceases, the formation or decomposition of a carbide, the point where a deposit occurs of some material vaporized in the hotter regions of the furnace, certain gradations in the color of the materials formed, etc. A white deposit may even be formed in some cases, about ¼ inch in thickness, which, in cutting away the materials after a run and exposing a cross-section of the furnace, gives an indication of the shape of that particular isothermal just as distinctly as if a chalk line had been drawn around the resistor.

Invariably with a rectangular granular resistor the indications of the furnace show that as time elapses the isothermals gradually become rounded surfaces of circular cross-section. This is a striking example of one result of the author's analysis. For when starting the furnace, the isothermal surfaces next the resistor must have similar or rectangular cross-section. That means, at the corners, the lines of flow of heat are

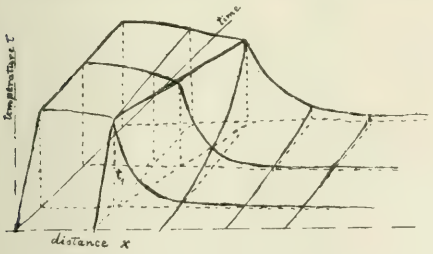


FIG. 4.—GRAPHICAL REPRESENTATION OF THE FLOW OF HEAT IN A RESISTANCE FURNACE.

more diverging than at the sides. Hence, according to the above theory, the rate of temperature change with the time is slower at the corners than at the sides. This necessarily rounds up the corners of the isothermals.

The conditions of a single-resistor furnace are shown in the diagram of Fig. 4. Here we have three rectangular co-ordinate axes, one representing the distance x measured along a line of flow of heat, the second the temperature t, and the third the time. The figure shows a uniform advance of the isothermal surface t_1 with the time, although this condition does not necessarily exist. A plane drawn parallel to the temperature and time axes at any value of x cuts from the temperature surface a curve representing the increase in temperature of the point x with the time. Similarly a plane drawn parallel to the o t and o x axes, cuts from the surface a curve showing the temperature drop along the line of flow of heat after a certain time has elapsed.

In the fourth part of his paper, Mr. Collens discusses the rate of energy supply per unit of surface of productive heat diffusion. He shows that a high value of this rate results in a high thermal efficiency, as far as the heating of the materials is concerned. A high value of the rate of energy supply, and consequently a rapid advance for the isothermal surface t_1 tends to maintain a steep slope for the temperature curve just outside the isothermal surface t_1 with a flatter curve beyond. This condition gives very efficient heating in the zone between

But this thermal efficiency should not be confused with the commercial efficiency of furnaces of this type, for in actual practice a maximum economical limit is often placed upon the rate of energy supply by other items effecting the total costs. Thus, if there is only a very small margin between the maximum and minimum temperature limits of a given operation, a high rate of energy supply means a very small thickness of product formed for the same cost of loading the furnace as if the rate of energy supply had been smaller and the thickness formed greater. Or, in certain cases, a time element may be necessary for the reaction in question, precluding too rapid rates of energy supply. But in spite of these limitations the importance of the rate of energy supply must not be minimized. In some cases the commercial efficiency may be increased 25 per cent by doubling the rate of energy supply.

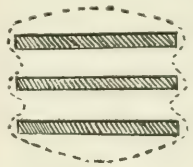


FIG. 5.—ISOTHERMAL WITH THREE-RESISTOR FURNACE.

The author sums up his general considerations in the following four principles:

1. Increase as far as possible the ratio that the surface of productive heat diffusion bears to the surface of non-productive heat diffusion.

2. Increase up to the economical commercial limit the rate of energy supply per unit of surface of productive heat diffusion.

3. Avoid as far as possible in the furnace design, shapes or conditions which give lines of flow of heat which are widely diverging.

4. Surround the productive zone of the furnace with materials which are as refractory as possible, both as regards thermal conductivity and thermal capacity, and which will still withstand the conditions to which they are exposed.

In actual practice these principles are already being utilized to a large extent, especially the first. Take, for instance, the multiple resistor furnace (Acheson, siloxicon, our Vol. I., p. 287) and furnace types in which the resistor is zigzagged through the materials from one furnace terminal to the other, such as Tone's silicon furnace (our Vol. II., p. 111) and a recent compartment furnace of FitzGerald and Bennie (our Vol. III., p. 346). In all these cases the saving effected is due to an increase in the surface of productive heat diffusion within a given surface of non-productive heat diffusion. As an example of the saving effected by multiple resistors a cross-section of a three-resistor furnace is shown in Fig. 5. On a certain class of work with 10 inches of product formed between resistors only three or four would be found on the top and bottom, the isothermal t_1 assuming the shape shown by the dotted line.

The above four general requirements may be summed up in one fundamental principle of intermittent resistor-furnace design: "inside the advancing isothermal surface [illegible] work, approach as near as possible, and outside avoid as much as possible the state of steady flow of heat and excessive temperatures."

The concluding part of Mr. Collens' paper deals with applications of the above principles to [illegible] commercial resistor furnaces. The carborundum furnace is discussed at great length, and it is pointed out that the principal

factor which enters into the determination of the most efficient size of carborundum furnace is the rate of energy supplied per unit of surface of productive heat diffusion.

If the radius of the resistor is increased its volume is increased in greater proportion than the surface of productive heat diffusion, which means that the energy required to heat the resistor itself will be greater per unit of product manufactured. With the radius of the resistor increased, the lines of flow of heat fom the surface of productive heat diffusion become less diverging, which is a slight advantage, but this is more than offset by the smaller rate of energy supply, and consequently lower thermal efficiency. If the radius of the resistor is decreased, the rate of energy supply per unit of surface of productive heat diffusion is increased, resulting in greater thermal efficiency. But at some point a maximum economical limit is reached. Thus if the surface of productive heat diffusion is halved, the rate of energy supply is doubled, which means that with definite maximum and minimum temperature limits the product can be formed to only one-half the thickness.

Hence we have only one-fourth the previous amount of product formed in practically one-fourth the time, and although the smaller amount is manufactured at a higher thermal efficiency, or power cost, the labor cost of loading and unloading the furnace and other fixed charges may not have decreased in the same proportion as the output of the furnace. Furthermore, as the size of the furnace is decreased the loss in the furnace heads becomes a larger factor in the total costs. Thus we have the two extremes, and by proper elimination the economical intermediate point is determined. Even for each size of resistor, however, there is a definite length of run giving the maximum efficiency, so the problem is not quite as simple as it appears at first sight.

The formulas given in the former parts of the paper are then applied by Mr. Collens to the carborundum furnace, and the heat diffusion as well as the different factors of energy consumption are discussed both mathematically and graphically.

In the Acheson siloxicon furnace a very low rate of energy must be used on account of the narrow temperature limits of the reaction. To counterbalance the low thermal efficiency which would follow from the low rate of energy supply a multiple-resistor design is used. But it is very probable that the low rate of energy supply more than offsets the high ratio that the surface of productive heat diffusion bears to the surface of non-productive heat diffusion, and that consequently the manufacture of siloxicon is carried on with less thermal efficiency than the manufacture of carborundum, on account of the characteristics of the two materials.

In the Acheson graphite furnace the surface of productive heat diffusion is infinitely large as compared with the surface of non-productive heat diffusion. As the only maximum temperature limit, if any, is that at which carbon vaporizes, a very high rate of energy supply may be used. Furthermore, the surface of non-productive heat diffusion may be surrounded by the most refractory materials which it is possible to obtain. Hence very much higher thermal efficiencies are possible in the manufacture of graphite than in the manufacture of carborundum or siloxicon. In the Acheson furnace for graphitizing electrodes the conditions are similar, and very high thermal efficiencies are possible.

In the Castner process for graphitizing electrodes, the current is passed lengthwise through a single electrode connected to the two furnace terminals. Theoretically such a method is more efficient thermally than the Acheson process. The surface of productive heat diffusion is infinite, being the surface of the molecules, and the rate of energy supply can be made exceedingly high. But commercially the Castner process cannot compete with the Acheson process, as the labor costs are much higher. Furthermore, the loss of heat through the furnace terminals in the Castner process is excessive. These must inherently be very much larger in cross-section than the carbon under treatment, and are made of material which is a good thermal conductor.

The Acheson process for polymerizing carbons is thermally the same in principle and design as the graphitizing process, and is as efficient. In Hall's process for baking carbon electrodes the articles are placed at one side of a granular carbon resistor, and are separated therefrom by a definite thickness of a material which is a poor thermal conductor. The articles are also embedded in similar materials. Mr. Collens thinks that, obviously, such a process cannot be as efficient thermally as the Acheson process.

The paper of Mr. Collens was discussed by Messrs. Bancroft, Richards, Burgess and Hering. Prof. Burgess spoke of the paper in very complimentary terms, while Mr. Hering asked for commercial figures and dimensions, in general for such data as are needed by designers. He also referred to the design of the terminals. Mr. Collens replied that the effect of terminals cannot be easily given; it is the more important the shorter the furnace. If the current passes not only through the resistor, but also through the material under treatment, this is no disadvantage for the thermal efficiency of the process, since heat is then also produced directly at those places where the chemical reaction takes place. While the above abstract gives the full argument of Mr. Collens' paper, the reader must be referred, with respect to the mathematical formulas, to the full paper, which will be printed in the Transactions of the Society.

Electrolyte Density in Storage Cells.

A paper by Mr. Lamar Lyndon, of New York City, was then presented, in the absence of the author, by Prof. Bancroft. The author refers in the introduction to the accepted reaction of the double sulphate theory, which is

$$PbO_2 + Pb + 2H_2SO_4 \rightleftharpoons 2PbSO_4 + 2H_2O,$$

the reaction going from left to right during discharge and from right to left during charge. When measurements of the density of the electrolyte between the plates were made in actual practice, it was found, however, that the higher the rate of discharge the less was apparently the abstraction of S and O from the electrolyte and the smaller the diminution of the density. Also with a given rate of discharge the change in density was apparently less toward the end of discharge than in the beginning.

This is not necessarily a contradiction of the validity of the above formula, since it leaves out of consideration the retarded diffusion of the electrolyte into the pores of the plates when the surface layer of active material has become practically sulphated, and thereby closes the pores in the surface of the plate. For this reason the electrolyte within the pores becomes more dilute than outside during discharge. If a cell is discharged at various rates and then allowed to stand discharged for several hours, the density of the electrolyte gradually falls (on account of diffusion into the plates), and in every instance approaches nearly to the value calculated from the above formula.

A plate, the active material of which is thin and porous, can be worked with an electrolyte of lower density than can one which has a thicker layer. Also in cells which are subjected to rapid discharges, the electrolyte density should be greater than in cells which are slowly and moderately discharged. The required density is, therefore, in a measure independent of the total quantity of electrolyte in cells having thick plates and subjected to rapid discharges, the active quantity being partly fixed by the amount of absorption in the pores of the plates.

High electrolyte density is therefore advantageous. It is also advantageous because the e. m. f. is higher. Very important factors, however, modify this deduction and fix the proper electrolyte density between 1.200 and 1.300 specific

gravity; *i. e.*, between 27 and 39 per cent of H_2SO_4. The author gave experimental results, showing that the maximum capacity is obtained with thicker plates at a higher density of the electrolyte, which is in agreement with his previous deductions.

If the density of the electrolyte is then still further increased the capacity falls off. This is partly due to a minimum of conductivity of H_2SO_4 at 1.224 specific gravity, as has been pointed out by Dolezalek. But Mr. Lyndon points out that the rate of diffusion must also be taken into account, and that this varies with the depth and porosity of the active material and with the rate of discharge.

As a rule, the higher the discharge rate and the thicker the layer of active material, the higher must be the acid density for minimum capacity and efficiency. The proper theoretical density for any cell is that which will most nearly average 1.224 throughout its range of variation in concentration within the pores, and manifestly each particular make of cell has its own best density for a given duty, which can be determined only by experiment. The factors of durability and length of time between successive charges must also be taken into account.

Mr. Lyndon then gives a formula for computing the quantity of electrolyte required. If S, the ampere-hours of discharge; X, the ounces avoirdupois of electrolyte required; D, the percentage of the electrolyte H_2SO_4 at the beginning of the discharge; and d the percentage at the end of discharge, then X may be found from

$$X = \frac{S\,(0.129 - 0.1053d)}{D - d}$$

The paper is concluded by the following table, which gives averages of general practice in this country:

Kind of Plates	Character of Service	Discharge Rate in Hrs.	Density — Initial.	Density — Terminal	Lbs. Electrolyte per 100 Amp.Hrs.
Pasted positive and negative	Motor car propulsion	4	1.300	1.100	4
Planté positive pasted negative	General use small sizes	8	1.210	1.187	22
Planté positive pasted negative	General use large sizes	10 8	1.210	1.177	15
Planté positive and negative	General use small sizes	8	1.200	1.180	25
Planté positive and negative	General use large sizes	10 8	1.200	1.165	15

Electrolytic Corrosion of Structural Steel.

In continuation of his former paper, presented at the Bethlehem meeting, Dr. Maximilian Toch, of New York City, the distinguished expert on paints, presented a further account of researches made by him on the important engineering problem of the electrolytic corrosion of structural steel.

Engineers have commented publicly on the electrolytic corrosion of structural steel, particularly those parts known as "grillage beams," supporting columns and base posts, which are either in the ground or surrounded by concrete and partly above the ground, with a view to determine beyond question at which of the poles corrosion occurs, and whether one pole is more active than the other. A series of experiments were started by Dr. Toch since the Bethlehem meeting, and definite results have been obtained.

The first experiment was performed, as in Fig. 6, by taking two sheets of high-grade watch-spring steel, which is extremely susceptible to corrosion, and connecting them with the ordinary bluestone telegraphic cell. A Pignolet combined volt and ammeter was placed in the circuit, and the two pieces of steel buried up to 5 inches in sand. The voltage was 0.05, the current varied from 0.02 to 0.05 amp., and the distance between the plates in the damp sand was 1½ inches. Careful observation was made every day to see that the current was uniform, and the sand was first moistened with salt water and then continually moistened with distilled water, so that the same strength of salt solution was maintained. This experiment was conducted for 100 days, and assuming that the current travels from plus to minus, or from anode to cathode, the anode being connected with the copper and the cathode being connected with the zinc, corrosion was noticed almost immediately at the anode, and the plate showed pitted surfaces at the anode and practically no corrosion at the cathode. The plates indicated some slight corrosion on the cathode, which, however, was probably chemical corrosion.

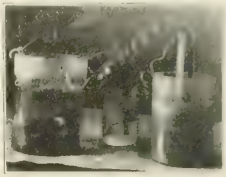


FIG. 6.—ARRANGMENT OF EXPERIMENT.

The next experiment was tried exactly in the same manner for a shorter period of time, but instead of using two plates three plates were used, the third one being designated as the "free" plate, in which chemical corrosion had full sway. It was not connected by any wire to the electric circuit. At the end of six days these plates were removed; the anode, as shown in Fig. 7, showed marked corrosion, the cathode plate showing practically no corrosion at all, and the "free" plate showed a fair average between the cathode and the anode, and it can be deduced that the difference between the cathode and the anode corrosion is equal to the "free" corrosion. In other words, there is many times more corrosion on the anode than there is on the "free" plate, and no corrosion on the cathode plate.

The plates were very carefully varnished all over to preserve them after the experiments were completed. The rust produced was first the green ferrous oxide, $Fe(OH)_2$, which, being a very unstable product, was quickly converted in the air into $Fe_2O_3(H_2O)$.

The voltage was 0.1 of and the current 0.1 of an ampere. The salt solution was four times as strong as that produced in the first experiment.

The third experiment shown in Fig. 8 was, however, of the greatest importance, owing to the fact that the author attempted to imitate the conditions exactly as they existed in

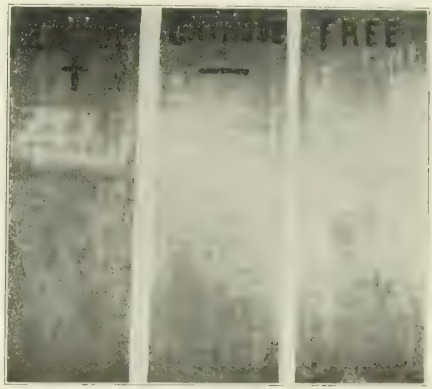


FIG. 7.—APPEARANCE OF THE THREE PLATES.

buildings. The same kind of steel was taken and bedded in various mixtures of concrete, starting from neat cement and going up to 1, 3, 5.

There is a well-known law in physical chemistry that reactions which take place with an increase of pressure are retarded by an increase of pressure, and the question has come up as to whether it is possible for steel to corrode when surrounded by concrete, many engineers holding that the alkaline nature of the cement will prevent the corrosion, and others

holding that in conjunction with this condition the pressure exerted by the concrete prevents chemical decomposition. Dr. Toch, in order to throw some light on this subject, made the following experiment:

In the first place, cement was taken of known composition, agreeing practically with the definition as quoted in the *Journal of American Chemical Society*, 1903, Vol. XXV., No. 7, July, when the question of the permanent protection of iron and steel by means of cement was thoroughly gone into. The cement for these experiments was what might be termed a tricalcic silicate and calcium aluminate. This is in contradistinction in the general classes of Portland cements containing dicalcium ferrite as a part of their composition and free calcium sulphate in excess. A cement of the calcium aluminate class, free from iron and free from calcium sulphate, is a well-known protector of steel and iron against corrosion, and this class of cement was used in these experiments. The pieces of steel were connected up with six elementary cells of

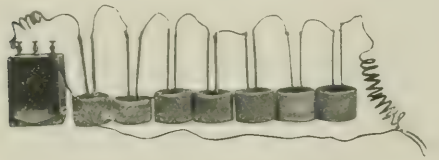


FIG. 8.—TESTING CORROSION OF STEEL.

sufficiently high voltage and amperage, and it was impossible to get a direct reading on the volt-ammeter, the instrument being too sensitive. The seven pats of cement containing the steel strips (see Fig. 8) were then put into the circuit and wet every few hours with solutions of 5 per cent sodium chloride and 1 per cent nitric acid and water, in order to increase their conductivity and produce corrosion as rapidly as possible.

The average voltage was 0.05 and the current 0.05 amp. throughout the entire experiment. Corrosion was immediately noticed on the anode pole, and the pat made of neat cement, which should have protected the steel most perfectly against all kinds of corrosion, showed a hair line split colored with rust at the end of the third day, which demonstrated that the chemical reaction of rusting had taken place at the anode; that the molecular increase had likewise taken place, and the pressure caused by the molecular increase had split the block.

The steel in each alternate pat was painted half the length, which was embedded in the cement with an insulating paint of known composition. The results obtained after these various briquettes were broken open, demonstrated that electrolytic corrosion takes place most violently at the anode unless the steel be coated with an insulating medium. Cement, concrete, or even neat cement, is, therefore, no protection against electrolytic corrosion unless the steel be insulated as heretofore mentioned. There was absolutely no corrosion where coated with insulating material. It must be noted that the cathode in all these experiments was perfectly free from any signs of oxidation.

The result of this entire series of experiments is to prove conclusively that electrolytic corrosion of structural steel embedded in concrete or steel tubes pipes only at the anode, and there with great violence; and, furthermore, that the cathode is protected by the electrical current. The popular impression that cement is a protector against corrosion of all kinds is fallacious, and that the anode does not only rust very violently, but a molecular increase of volume may take place which will split the concrete shell.

Another conclusion arrived at is that the electrolytic rusting of girders, beams or building steel need not be feared if the structural steel be protected by a good insulating material, but the insulating medium should form a bond with concrete.

Dr. Toch's paper produced quite an animated discussion, in which Messrs. Acker, Bancroft, Hering, Palmer, Richards and Rodman participated. Mr. Rodman said that if steel is protected as cathode, then the steel structure of buildings should be artificially made a cathode. Mr. Hering referred to a means for this purpose, which consists in burying zinc in contact with the steel in the ground. This represents a short-circuited cell, in which the steel is the cathode; of course, the zinc is gradually consumed, but the steel is protected. The method is exactly the same as for the protection of boilers. Prof. Bancroft asked whether there was not with this method a possibility of a leakage of current from one steel cathode to another steel cathode, whereby the former one would, of course, be corroded. Mr. Acker spoke of a case in which he had been interested, of the destruction of gas and water pipes by the stray currents from an electric tramway system. The trouble was successfully overcome by improving the bonds between the rails and installing special return feeders, for the purpose of preventing stray currents from passing into the earth. Mr. Hering referred to the method which is usually employed in reports on the effects of stray currents from railways, and which consists in measuring the potential difference between rails and pipes; but it is not the potential difference but the ampere-hours which determine the amount of corrosion, and it is not proper to assume a large current with a large potential difference, since the conductivity of the ground is hereby entirely neglected. It is, therefore, not sufficient to measure the potential difference alone.

Fused Magnesia Oxide.

A paper by Prof. H. M. Goodwin, of Massachusetts Institute of Technology, and Mr. R. D. Mailey, dealt with the physical properties of fused magnesium oxide. They first describe the methods which they have used to fuse magnesia. The first attempt was made in a graphite-resistance furnace. A pure Acheson graphite rod, 18 inches long and ¾ inch outside diameter, was bored with a ½-inch drill, and the ends fixed into large graphite block terminals for leading in and out the current. The furnace was mounted vertically, packed with coke to prevent excessive oxidation of the graphite tube, and the whole heat insulated with firebrick. From 20 to 30 kw. were used in the furnace to maintain the temperature required. Chemically pure magnesium oxide was fed in at the top of the furnace in the form of a powder, and packed down by means of a graphite rod. In this way it could be melted, and rods of the fused oxide were obtained about ¼ inch in diameter and 2 inches long.

In no case, however, could rods be formed as large as the interior diameter of the bored graphite rod, owing to an action between the oxide and either the walls of the tube or gases given off from or diffusing through the walls of the enclosing tube. This was indicated by the formation of a volatile product, which condensed to a black mass in the upper cooler portions of the furnace. To prevent this substance from completely stopping up the furnace it was found necessary to bore lateral holes through the graphite tube in the hottest zone, at which condensation could not take place. The composition of the substance condensed in the cooler part of the furnace was not determined, although tests indicated that it was not a carbide. Treated with acid, hydrogen sulphide was always evolved, showing that impurities from without the furnace proper, probably in the coke, must have diffused through the graphite to the oxide within. The graphite rods were tested and found to be very pure, having been especially prepared for this work by the Acheson Graphite Co., of Niagara Falls.

As it was found that rods of magnesia of the desired size could not be obtained by the method above described (at least when carried out on the scale feasible at the time), it was thought that an arc furnace might be more effective. A hollow rectangular graphite boat, 10 x 4 x 4 cm., was used as a receptacle for the fused oxide. Over the center of this was

arranged a powerful alternating current arc, into which the oxide was fed from above. The whole was enclosed in graphite bricks, around which were built up fire-bricks. In this way irregular masses of the fused oxide, as large as one's hand, were obtained, and from such masses, rods or pieces of the desired length were bored by means of carborundum. The large masses thus obtained were not, however, homogeneous. Certain portions or layers were composed of very fine compact crystals resembling marble; other portions were composed of larger crystals. Some excellent samples were obtained, however, so hard and compact that they were quite non-porous to water and to fused salts. The great difficulty experienced was in getting pieces sufficiently free from tiny bubbles or blow holes. Up to the present time the authors have been unable to completely eliminate these in pieces larger than two or three cubic centimeters. If the process were to be carried out on a much larger scale and a larger mass of oxide brought to state of quiet fusion and then cooled, better results could undoubtedly be obtained.

In discussing the physical properties of fused magnesia the authors first mention its appearance, which is pure white and of a very hard crystalline structure. The fused surface resembles glazed porcelain.

The hardness is very high, and is between that of apatite (transparent variety) and feldspar (white cleavable variety), though nearer the former than the latter.

The specific gravity of three samples was found to be 3.485 at 19° C. and 3.562 and 3.493 at 20° C.

Determinations of the melting point show that it is between

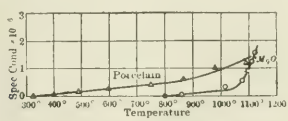


FIG. 9.—ELECTRIC CONDUCTIVITY OF MAGNESIA AND PORCELAIN.

1,890° and 1,940° C., so that in round numbers 1,920° C. may be taken as the approximate value. A more precise determination will be made in the near future by the authors.

The conductivity of fused magnesia was next determined, and may be seen from the following table, in which all the figures from 700° C. to 1,150° are the results of the authors' experiments, while the last figure, referring to 1,500° C., is taken from the Nernst-Reynolds-Landolt and Börnstein tables:

	10^5 x Conductivity.
800	0.01
900	0.10
1,000	0.20
1,050	0.34
1,100	1.00
1,150	2.60
1,500	85.00

[In the discussion which followed, Mr. Hering stated that the figures in this table were evidently given in meter-square millimeter reciprocal ohm units.]

Fig. 9 shows the change of the electric conductivity of fused magnesia compared with that of porcelain. Magnesia is a much better insulator than porcelain at temperatures below 1,100° C. Above that temperature the large temperature coefficient of magnesia indicates that its conductivity probably becomes greater than that of porcelain.

The measurements of the authors, which were made with direct current, indicated distinct polarization in the case of porcelain, but with magnesia no trace of such action was evident.

The last part of the paper of Goodwin and Mailey deals with their determination of the coefficient of expansion, and an interesting result of their investigation is the close agreement which exists between the value of the expansion coefficient of fused magnesia and that of platinum. This should prove a valuable property in the construction of apparatus involving joints of these two materials. The authors have found that platinum caps, ground to fit the ends of rods of the fused magnesium oxide, remain tight over wide ranges of temperature.

The following chemical properties of fused magnesium oxide are interesting: The fused magnesia possesses in a remarkable degree the ability to withstand chemical action of many neutral salts at high temperatures, and is, therefore, well adapted for use as vessels and apparatus for containing such salts when subjected to high temperatures. Qualitative tests of the chemical action of various fused salts on the oxide gave the following results:

Silver, sodium and potassium nitrates, sodium and potassium chlorides, bromides and sulphates, zinc chloride and barium nitrate showed no action on a polished sample of the fused oxide, when the latter was heated for an hour or more in the fused salt. Barium chloride had a very slight action; sodium carbonate, potassium sodium carbonate, potassium hydrate, and cryolite attacked the fused oxide energetically.

Dilute hydrochloric, nitric and sulphuric acids attack the fused oxide in the cold slowly. Concentrated acids are less active than the dilute acids.

In a communicated discussion, Messrs. FitzGerald and Bennie, of Niagara Falls, pointed out that the properties of fused magnesia depend enormously on the purity of the material. In some tests of different samples of fused magnesia, taken from different parts of an electric furnace, the density was found by them to vary from 3.52 to 3.57. Since a small addition of impurities alters greatly the melting point, the conductivity and other physical properties, an exact analysis of the constitution of the material should always be given. Mr. Carl Hering objected to the use of the term specific conductivity by the authors, and urged that either the terms conductivity or specific conductance should be used. At 1,500° C. the resistivity of magnesia is about 700,000 times that of copper at ordinary temperature, while the resistivity of the Nernst lamp filament is about 400,000 times that of copper. He referred to a statement of Heraeus, according to whom porcelain at 1,600° C. conducts injuriously; *i. e.*, cannot be depended upon as an insulator. Prof. Bancroft referred to a case in which a resistance furnace had been built with porcelain as resistor; he also exhibited various magnesia articles made in Germany. Photographs of fused-magnesia articles made by Messrs. FitzGerald and Bennie were also shown.

Cathodic Disintegration of Carbon.

A paper on this subject by Mr. George I. Kemerer (which, like the next two papers, was a contribution from the University of Wisconsin, and was presented in abstract by Prof. C. F. Burgess), called attention to the fact that graphite is unattacked, and that other forms of carbon are fairly durable when used as anode material for the electrolysis of fused sodium chloride, but that it is apparently not so generally known that the same material when used as cathode is rapidly consumed. While appreciable corrosion occurs with Acheson graphite, it is less than 1 per cent of that which occurs on amorphous carbon. With amorphous carbon rods as cathode in fused sodium chloride, the portion dipping into the electrolyte is completely disintegrated in from [illegible] to 2 minutes. The author supposes that the liberated sodium unites chemically with the carbon, forming a sodium carbide, which would immediately break down at the high temperature used to keep the bath molten. The amount of disintegration was not proportional to the quantity of current, as would be indicated by the application of Faraday's law, but current density and temperature seem to exert an influence.

In the discussion which followed, Dr. J. Forssell, of the National Carbon Co., Cleveland, Ohio, pointed out that ordinary baked carbon rods consist of particles of carbon cemented together by a binding material; the latter would naturally break down under the treatment of the rod as cathode in a fused salt. Prof. Bancroft objected to the connection of Faraday's law with cathode corrosion; it is impossible to assume the formation of a definite compound: there may be the break-down of a solid solution.

A New Silicide of Molybdenum.

A paper by O. P. Watts states that in 1893 only eleven compounds of silicon, with seven different metals, were known. The application of the electric furnace to this line of research resulted in a great extension of the list of metallic silicides, now thirty-two in number, and the end is not yet. A silicide of molybdenum, of the formula $MoSi_2$, has been known before. The author describes some experiments which make probable the existence of a silicide of molybdenum richer in silicon than Mo_2Si_3, and probably having the formula $MoSi_3$.

Reduction of Metallic Sulphides.

A paper by O. W. Brown describes experiments in which the method of treating zinc sulphide with lime and carbon (see paper of Brown and Oesterle, our Vol. III., p. 378) is applied to the reduction of other metallic sulphides. The experiments were successful with the sulphide of molybdenum.

The sulphide of molybdenum, which is the principal source of this metal, is usually roasted until the sulphur is completely expelled and the impure oxide of molybdenum obtained. This oxide may be purified by the usual chemical methods and then reduced to metal by smelting with carbon in the electric furnace. If only a commercial metal is desired, the impure oxide, obtained by roasting the molybdenite, may be smelted with carbon without being purified. A considerable amount of molybdenum is lost during the roasting process, as the oxide is volatile at the temperature required to expel all of the sulphur.

The reduction of the comparatively non-volatile molybdenite directly to metal would save the cost of roasting, and prevent the loss of molybdenum which takes place during the roasting process. The author has found that molybdenite is easily and completely reduced to solid metal by electric smelting with lime and carbon, if the relative quantities of the material in the mixture correspond to those of the reaction

$$MoS_2 + 2CaO + 2C = Mo + 2CaS + 2CO$$

The largest amount of sulphur which was found in any of the samples of molybdenum made in this way was 0.25 per cent. For the complete reduction of molybdenite to metal, an excess of lime in the charge seems to be of advantage, while an excess of carbon leads to the formation of carbide, which holds back some of the molybdenum.

The author then tried to employ an analogous method for antimony. As metallic antimony boils at the comparatively low temperature of 1,100 to 1,400° C., it appeared that its sulphide ore could be reduced to metal and distilled in a manner analogous to that which may be used with zinc blende, but the experiments of the author show that the reduction and distillation of antimony from a charge of stibnite, lime and carbon is incomplete, even when the mixture is heated in an electric furnace to a very high temperature.

Sodium Production.

The last paper of the Monday session was presented by Mr. Hamilton Y. Castner, the well-known and French electrometallurgist. The first part of the paper is a very interesting commercial review of the production of sodium, its present status, and future possibilities.

The production of sodium metal, in the United States alone, is approximately 1,200 tons per year, whilst in England, and also in Germany, nearly similar amounts are produced. The disposal of this considerable quantity of metal takes place somewhere about as follows:

	Tons.
Used for cyanide making	1,500
Used for peroxide making	1,500
Sold as metal	500
Total of the world's production	3,500

The figures, of course, are only approximate.

The price of the metal has gradually fallen during the last few years from 60 cents a pound, or thereabouts, to some 25 cents a pound, which is about the price to-day for large steady contracts.

The process due to the late Hamilton Young Castner has been in successful operation at Niagara Falls for many years. Many competitors have, however, arisen, and particularly, many attempts have been made to produce the metal by direct electrolysis of salt; but it is safe to say that none of these have, as yet, attained any substantial success, and, now that the older and successful process is becoming public property, through the expiring of patents, still less encouragement will be offered to possible competition in the future.

"In spite of these handicaps the author, like most inventors having a discursive kind of muse, has trayed into this field, and is now asking your kind attention to a brief account of a new method of production. Like a certain poor girl, when convicted of the crime of producing something which was not quite in line with the established order of things wanted, I feel inclined to plead in excuse, that the baby is 'only a little one.' I have, however, long felt that a genuine field for improvement exists in this growing industry, and my reasons are briefly as follows:"

Practically all the sodium at present made in the world is made by the Castner process, which consists in electrolyzing a pure form of caustic soda between electrodes of copper and nickel.

The costs of this process are known, and are, broadly, as follows, per pound of sodium produced:

Caustic soda	$0.05
Power according to locality	0.01 to 0.05
Labor	0.025
Up-keep and standing charges	0.02

Making a minimum possibility of cost under existing conditions of from 10.5 to 14.5 cents per pound. There are no by-products.

The new process of the author makes the metal by electrolyzing common salt in a fused state, by using a double electrolytic cell, in the first portion of which fused sodium chloride is electrolyzed with a molten lead cathode; an alloy of lead and sodium being formed, which is transferred to the second cell, where it is used as anode with an electrolyte of fused sodium hydroxide, yielding metallic sodium at the cathode. The sodium hydroxide is not consumed. [The first half of the process is, therefore, similar to that of Acker, the second to that of Castner.]

The cost of the Ashcroft process, when working on the same scale as the above estimate for the Castner process, is estimated as follows:

Common salt	$0.005
Power according to locality	0.01 to 0.05
Labor	0.01
Up-keep and standing charges	0.025

Making a gross cost of 5 cents to 9 cents per pound of sodium produced, according to the cost of power.

In addition to this very substantial reduction in cost of manufacture, the new process produces a by-product of pure

chlorine gas which may be worked up by well-known methods into any of the numerous chlorine products, notably zinc chloride, bleaching powder, potassium chlorate, stannous chloride, sulphur chloride, carbon tetrachloride or chloroform. The value of this (if worked up into the heavier products like bleaching powder) is approximately 2¼ cents per pound of sodium (net after deducting cost of working up the bleaching powder). The value would be considerably greater if worked up into some of the other products named.

The power factor in each of the above two cases is practically identical, the Castner process requiring 4½ total volts per cell and yielding 45 per cent real current efficiency, whilst the author's process requires 9 volts (7 and 2) per double cell, but yields twice the amount of metal (or 90 per cent real current efficiency) for a given current.

Mr. Ashcroft remarked here in parenthesis that "the low-current efficiency of the Castner process (a point, by the way, which has been often misstated) in comparison to the author's method (both delivering the sodium from caustic soda) is due to the chemical necessity which exists in Castner's method for decomposing an equivalent of water for every equivalent of sodium set free from caustic, a necessity which does not exist under the peculiar conditions of the electrolysis of caustic in the Ashcroft arrangement, as may be seen by a study of the reactions."

The commercial conditions (as to locality of works, etc.) for the two processes are practically identical, and great advantages are claimable by this industry for "cheap power" sites. The following table of Mr. Ashcroft shows the comparative cost of the several known means of producing power for such processes per horse-power years:

	Steam Engine Average	Gas Engine Average	Oil Engine Average	Niagara (Water) Average	Norwegian (Water) Actual
Cost of fuel.....			$15.00		0
Labor and upkeep	$40.00	$30.00	2.75	$20.00	$2.50
Capital charges..			7.25		5.00
	$40.00	$30.00	$25.00	$20.00	$7.50

Applying these figures of power cost to the manufacture of sodium, the result is approximately as follows per ton of metal produced:

Steam.	*Gas.*	*Oil.*	*Water, Niagara.*	*Water, Norway.*
$120.00	$90.00	$75.00	$60.00	$22.50

But the question of transport is also important to the commercial issues, and the site of the works should be chosen with this also in view. The metal is at present transported in packages of 200 pounds weight. The package consists of a sealed metal drum, into which the sodium is packed in 5-pound cakes, taken direct from the producing plant. The peroxide is packed in 10-pound tins, placed inside a wooden box. The cyanide is also exported in sealed packages. The packages are returnable (except from very long distances), and in actual practice are found to last well, so that the entire cost of packing and transport has been now brought to a very low figure.

No special precautions are necessary beyond the exclusion of air and water. Although a few years ago sodium and its products would have been regarded as dangerous chemicals for transport purposes, the experience of the last few years has removed nearly all restrictions, and cheap freights now prevail in most countries. The author believes the rates of carriage in this country, moreover, are sufficiently low to minimize the economical necessity which exists in most other lands for getting right on the seaboard with all such factories.

Mr. Ashcroft concluded this section of the paper by a brief statement of some of the special reasons which lead him to hope for a large extension of the sodium industry in the future:

(*a*) The metal is produced from common salt, which can be obtained everywhere at a low cost as compared to any other metallic ores, and which contains no less than 40 per cent of the metal with scarcely any impurities or other contents except the combined chlorine (60 per cent). These conditions are unique and ideal from the metallurgist's point of view.

(*b*) The metal can be produced by the new process with an expenditure of no kind of material at all, except the salt (corresponding to the ore in other metallurgical operations) and a trifling amount of incidental stores. When producing on a large scale, almost the entire cost of production will be for electrical energy, and, given "cheap power" conditions, the cost of production by the new process may ultimately be brought very low indeed. These conditions are unique and ideal from the purely commercial point of view.

(*c*) With a reduction of price of sodium greatly extended, uses will be found for the metal, since very special physical, chemical and electrical properties attach to this metal unique properties. It is the lightest of all common metals, not excepting aluminium. It decomposes water freely by mere contact, producing a large volume of hydrogen gas and a by-product of caustic soda. This property suggests at once the use of water and sodium as balloon ballast for long journeys, so that when the ballast was to be discharged a simultaneous refreshment of the gas could be secured as well. It has the greatest energy of combination weight for weight with chlorine, oxygen, etc., of all the metals (excepting aluminium, which is an inherently much more expensive metal owing to the prevailing impurity of its ores and the difficulties of its reduction), and on this property reasonable hopes are founded that it will some day find extensive application as a medium (either secondary or primary) for batteries having the object of producing or conveying to a distance cheap electric energy. It reduces most oxides or salts of other metals setting free the metal, and it is reasonable to suppose that this property may insure its finding further extensive employment in metallurgical industries (particularly iron and steel production and purification, or in the chemical reduction of sulphide ores in remote localities), when it shall be produced in large quantities at prices that will satisfy economical conditions.

Sodium has, weight for weight, the greatest electrical conductivity of all the metals. The most promising future for sodium lies in the already extensive and rapidly growing gold-cyanide industry. Before the war in South Africa some 10,000 tons of cyanide were used yearly in gold extraction, and since the resumption of operations there, and with steadily extending growth of this form of gold extraction throughout the world, it is estimated that the world's consumption of cyanide has reached considerably larger figures; probably some 25,000 tons per year or more. Although much of this cyanide is manufactured by processes not employing metallic sodium as a base, it is well known that by far the best processes in use do so. Only the high ruling prices of metal sodium have limited the extension of these processes to the exclusion of all others, and a careful comparison of all known costs shows that whilst cyanide finds a ready market to-day at 16 to 18 cents per pound, but little profit remains to makers at this figure. The sodium employed being valued at 25 cents per pound, nearly all the profit is then on the sodium.

As an outlet for cheaper sodium, however, this case is very different. The conversion of metallic sodium into cyanide is possible in several different ways and is a cheap operation. For a works able and willing to market the metal at, say, 12 cents per pound, effective competition can be instituted with any existing process of cyanide manufacture.

The second part of Mr. Ashcroft's paper contains the technical description of his own process (the first of his American patents was abstracted in our Vol. III., p. 481). The patents were acquired in the early stages of the invention by the United Alkali Co., of Liverpool, and the process has been

worked first by the author and then out in detail by the staff of this company at their central laboratory at Widnes.

The apparatus illustrated in Figs. 10 to 16 comprises a composite cell, the two components of which are united by means of a "regenerator or heat equalizing device."

Salt is charged into a hopper *A* and fuses in the bath at *B*. The main current before passing into positive electrode *C* from the terminal, makes several circuits around the coil of conductor *E*, placed behind the lining *F* in the interior of the

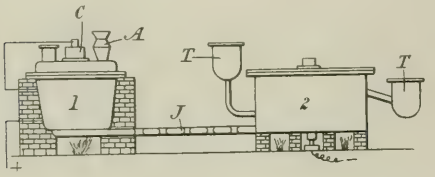


FIG. 10.—GENERAL ARRANGEMENT OF PLANT FOR PRODUCTION OF SODIUM.

decomposing cell 1, and thus sets up a strong magnetic field in the iron casing *C*, the lines of force tending to pass, as shown by the arrows, Fig. 12. The current crosses the electrolyte, decomposes the salt and then diffuses radially into the lead-sodium alloy *H*, forming the cathode on the bottom of the cell. The radial lines of current in the alloy cutting the lines of magnetic force nearly at right-angles, cause a motion of the metal in a plane again at right-angles. The direction of motion of alloy is indicated by the arrows, Fig. 14.

The effect of this is to maintain the liquid cathode and the electrolyte in circular motion, and as the rich alloy is formed immediately under the carbon anode, it is withdrawn from contact with the electrolyte by a skimming action on the surface of the lead, and passes into the division *I* of the composite pipe *J* through the holes *K*, the action being assisted if desired by small baffles *k* (Fig. 14). The adjustment of the magnetic exciting effect of the coil may be accomplished by varying the number of turns of the coil included in the main circuit, and for this purpose the several turns of the coil may be connected to a commutator or the like.

The main current, after entering the fused alloy (acting as cathode) in the decomposing cell 1, passes along the metal pipes *J* through the alloy (the pipe being preferably arranged

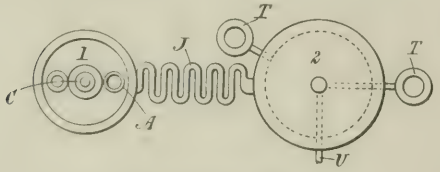
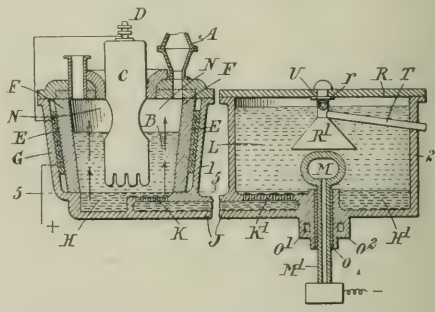


FIG. 11.—HORIZONTAL CROSS-SECTION.

as illustrated in zigzag form to secure large surface) to the second (or sodium producing) cell 2. Here the alloy acts as the anode *H*, and the current traversing the electrolyte *L* (of caustic soda or the like) enters the cathode *M*, which is preferably of metallic nickel and of globular form, and is connected to a stem *M*¹ of copper, preferably in tubular form, which passes through the bottom of the cell and serves to convey the current out of the bath. This cathode is insulated from the cell at *O* by an insulating sleeve (which may conveniently be composed of the chilled electrolyte), and through its tubular supporting stem a slight cooling effect may be obtained by air circulation (which can be further increased, at will, by circulating any suitable cooling liquid). This has the effect both of aiding in solidifying the insulation in the sleeve *O*, and also slightly lowering the temperature of the bath at the actual point where the sodium is deposited, and thus minimising any reaction between the caustic soda and the sodium.

A similar device, to that described for the No. 1 cell, consisting of small holes *K*, Fig. 13, leading out of the composite pipe, and backed if desired by similar small baffles, ensures the delivery of the rich alloy at the surface of the bath of molten metal in this cell, and thus in the most active region for the desired removal of the sodium therefrom, and causes the alloy to circulate gently in the cell also. After circulating in this cell, the depleted lead, now more or less freed from sodium, is withdrawn by the motion of the metal into fine holes placed in the side of division *O* of the composite pipe *J*; thence it passes along the pipe and returns through similarly arranged small holes *P* (see Fig. 14) laterally into the decomposing or salt cell, thus completing the circulation.

The working temperature of the metal in the salt (or decomposing) cell, is approximately 700° C. when ordinary commercial salt is being decomposed. By means of the regenerative device shown, the alloy passing through the composite pipe *J* in opposite directions in the two divisions, transfers heat and thus tends continually to equalize the temperature of the adjacent passage at both ends. By this means the hot alloy leaving No. 1 (or decomposing) cell, heats the cold



FIGS. 12 AND 13.—VERTICAL CROSS-SECTION.

alloy arriving from the No. 2 (or producing) cell, and the cold alloy leaving the No. 2 cell cools the too hot alloy arriving from the No. 1 cell, so that the desired temperature is maintained in both cells without waste of heat.

For the purpose of collecting the sodium deposited on the globular cathode *M* in No. 2 cell, the cell is preferably sealed from the air by cover *R*, and a funnel-shaped nickel screen *R*, terminating in a small pipe *r* of restricted sectional area is affixed to, and remains preferably to electrical contact with the cover *R*, and thus with the entire anode system of the cell. The effect of this somewhat unused arrangement is to preclude the possibility of sodium being set free in any part of the cell, except on the isolated globular cathode, from which it is immediately disengaged by gravity, and any sodium redissolved by anode at the point *X* will react with the caustic, according to the normal reaction $2NaOH$ (subjected to current) $= 2Na + H_2O + O$, and $2Na + H_2O + O = 2NaOH$ again, and not according to any of the reactions (such as $2Na + 2NaOH = Na_2O + H_2O$, etc.) which are found to take place when sodium metal remains long in contact with caustic soda.

Sodium is thus again set free at the cathode and normal caustic reformed at the anode, so that any leak current has no detrimental effect on the composition of the electrolyte, and, furthermore, prevents direct chemical action. All the metal set free on the cathode, as already stated, immediately floats by graviation to the surface and enters the restricted pipe *r*.

Being lighter than the caustic electrolyte its level rises, and it can thus be caused to leave the cell continuously as fast as formed, and collect in a suitable bath or receptacle by passing along a suitably placed pipe *U* (Figs. 13 and 15).

The restricted area of contact between the sodium and the caustic ensures that any possible corrosive action on the deposited sodium is reduced to a minimum, and the total efficiency of the cell thus increased to a maximum. This precise form of the above improvements relates to the fact that it has been found that the action of the sodium on caustic, when long exposed, tends to form oxides and sub-oxides, which reduce the efficiency of the caustic as an electrolyte. It may, therefore, sometimes be desirable to periodically renew the caustic by circulating it through pipes and reservoirs *T*. The caustic which has been long used in the cell and become inefficient, owing to solution of sub-oxides, etc., ca nbe regenerated and brought into a useful condition again by merely exposing it to the air and allowing it to absorb water, or by treating it with steam, or by cooling and dissolving it in water and refusing.

The lining of the first (or decomposing) cell is of any desired material, such as magnesia bricks, which are found to withstand well the action of the electrolyte, alloys and other products, but even silicious or aluminous bricks (which are cheaper) are mostly found to be sufficiently durable. The heat of the cell may be supplied either from within (by the current) or from without by means of the gas jets or fire shown, or both expedients may be resorted to. The second (or producing) cell, when caustic is employed, may be conveniently regulated in temperature by a small supplementary gas flame, which can be adjusted at will. The rigid exclusion of air from the melted sodium is not so necessary when caustic soda

FIGS. 14, 15, 16.—DETAILS OF CONSTRUCTION

is employed as secondary electrolyte, as it is with electrolytes fusing at higher temperatures, but it is, nevertheless, desirable to exclude air from the apparatus as much as possible.

Only those who have tried production of sodium by electrolysis of common salt—not on paper, but in factory-scale experiments, know how difficult a problem it is. In the first place, salt fuses at about 720° C., and sodium vaporizes at nearly the same temperature. Therefore, such processes must collect two gaseous products, chlorine and metal, and the apparatus must be arranged so as to keep them religiously apart. But at this and even at much higher temperatures the vaporization of the sodium is far from rapid or complete. It floats about in great blobbs on the surface and eagerly seeks every opportunity of meeting with its beloved chlorine. The real practical difficulty of avoiding all this and of simultaneously excluding all air is enormous.

These and other difficulties of a practical nature have never been effectually surmounted in spite of many paper processes and many immature patents; it would be rash, however, to predict that they never will. As sodium chlorine is a true electrolyte, and one which does not react with metallic sodium detrimentally (the sub-chloride theory at one time advanced has been exploded) to any great extent, the tendency to work in this direction will probably continue. The nearest thing to success in the direction of electrolyzing salt was found in the work begun by Rogers, of Milwaukee, many years ago, and perfected by Acker recently for another purpose; that is the manufacture of lead-sodium direct from salt. The Acker pro-

[illegible] Vol. I, p. 54) [illegible] the [illegible] does not go far enough [illegible]

Here the difficulty of practically dissociating sodium chlorides just disappears, but the [illegible] lead alloy, at the melting point of salt, although it passes to distil at a point very slightly higher, and for that reason the temperature and other conditions require very careful adjustment. "The process I have described takes full advantage of the work of Rogers and of Acker and in other matters in the same field, and acknowledgements are due to these."

Another fortunate circumstance for the Ashcroft process is the lowering of the fusing point of lead by the just necessary 50° or so, which a slight alloy of sodium brings about, thus enabling the fluid alloy to be used in the second cell without overheating the caustic soda. As caustic soda appears to be the one electrolyte from which sodium can be quite comfortably collected in fluid form, even whilst exposed to the air, and as it is, moreover, essentially cheap, there is a decided practical advantage in using this material for the unchanged secondary electrolyte, in preference to any of the other possible salts of sodium which might be employed. A further point of incidental advantage gained in the sodium end of this process is the avoidance of any liberation of hydrogen. Not only is the current efficiency thereby doubled but the continuous, automatic collection of the sodium is rendered possible, owing to the absence of the continual fusilade of popping hydrogen explosions, which is inseparable from the Castner process.

The paper produced considerable discussion, in which Messrs. Acker, Carrier and Hering participated. All of them thought that the electromagnetic method of stirring would be very inefficient, and that some positive mechanical device would be more satisfactory. Mr. Acker also referred to impurities occurring in common salt, and to the formation of a crust on the molten lead cathode in the first cell, which would offer great difficulties. The second half of the apparatus seemed attractive to Mr. Acker, but he thought it would need to be much larger than shown in the figures on account of the lower temperature. Mr. Hering pointed out that the ampere-hour efficiency of both cells must be absolutely the same, otherwise it would be necessary to shunt one of the two cells. Mr. Carrier thought a great difficulty and a large loss of heat would be found in transferring the lead-sodium alloy from one furnace to the other furnace. If the crust on the lead alloy referred to by Mr. Acker should be formed, it would increase the voltage very much. Mr. Ashcroft, in his reply, said that the first cell (to which the chief objections had been made) had been in continuous satisfactory operation for three weeks on a large scale, and that all practical difficulties which had first been experienced had been overcome. As to the larger size of the second cell and the necessity of shunting one of the two cells, he agreed with the previous speakers. With respect to the alleged inefficiency of the electric magnetic stirring apparatus, he replied pointedly that the power required for this purpose was quite as negligible as the power which is required for raising a teaspoon or a glass of beer from the table to the mouth. The chief point is that it gets there; the electromagnetic stirring apparatus is extremely convenient and does the work well, while the cost of power for this purpose measured in foot pounds is insignificant.

TUESDAY AFTERNOON

On the afternoon of May 2 the chemical laboratory, the dynamo laboratory, the engineering laboratories and the shops of Cornell University were inspected. The excellent equipment of the laboratories was highly appreciated by the visitors, especially as quite a number of interesting experiments were in actual progress during the visit, including electrolytic deposition of chromium and the manufacture of [illegible] and calcium carbide in an electric laboratory furnace, as well

as the electrolytic plant from which oxygen and hydrogen gases are supplied to the different laboratories. In the electrochemical lecture room, Prof. Bancroft described his new switchboard, which offers a possibility of getting any voltage and current at different parts of the table. He also showed some instructive electrochemical experiments.

The Electrochemistry of Chemistry.

At the last meeting of the Society at Bethlehem, Dr. Wilder D. Bancroft presented a lecture on the "Chemistry of Electrochemistry" (see our Vol. III., p. 370). His presidential address, presented on the evening of Tuesday, dealt with the reverse problem, namely, the electrochemistry of chemistry. He referred to the great assistance which electrolytic methods offer in chemical analysis, and then called attention in broad lines and in a most suggestive manner to the service which electrolytic methods may render in chemical researches when the same reaction can be produced in an electrolytic cell as in a chemical process.

At first it would seem that there is always a very distinct difference, since an electrochemical process always occurs at two distinct spots, the anode and the cathode, while a chemical process takes place uniformly within the reacting materials. But the difference is rather more apparent than real.

The author discussed a number of different reactions which "go" both chemically and electrochemically, and dwelt on such cases in which there is an apparent difference. In various cases he could show that an apparent discrepancy was simply due to the fact that the conditions of both cases were not exactly alike. He dwelt especially on the study of corrosion of metals, and emphasized a general rule, namely, that if a metal becomes passive as anode in a solution it cannot chemically rust in that solution. This rule, as far as applied, has always been found to be correct. Any solution in which the iron anode becomes passive does not produce rust. In sodium chloride solutions an iron anode does not become passive, hence iron will rust in this solution and the reaction velocity is normal.

Wilder D. Bancroft.

In this way the corrosion of iron, steel and other alloys may be studied. The chemical study of corrosion is not an easy thing, since it would require weeks or months, and it would be extremely difficult to maintain the conditions constant. But since we can produce the same results by making the metal the anode in an electrolytic cell, we can finish the test in 8 hours instead of eight weeks. Of course, the results must be checked by analogous tests of chemical corrosion.

In passing over to the discussion of solution of bronzes, Prof. Bancroft referred to the statement that it is the less noble metal which will always dissolve. He said this is a very nice statement but it is not true. The whole argument in its favor is based on the measurement of electromotive forces. However, as soon as corrosion begins, the original system is changed.

It seems that the whole problem of preventing corrosion depends very largely on the question whether a protective film is formed; now this side of the question is actually lost sight of in electromotive force measurements. In general, electrolytic corrosion and chemical corrosion are identical. All the exact evidence which we have points that way (see the paper of Curry abstracted below).

The electrolytic method has the advantages of convenience and reduction of time for the investigation of corrosion, but the rate of corrosion is, of course, different in both cases, and the results must be checked. At the conclusion of his address, Dr. Bancroft called attention to the use of the electric current in synthetic organic chemistry, where in recent years a large and promising field has been opened.

For details of the highly suggestive and interesting paper the reader must be referred to the full address, which will be printed in the "Transactions" of the Society.

After the address there was a most enjoyable reception at the Town and Gown Club. Refreshments were served in the upper rooms; the steins clinked freely and rival songs from the University Glee Club and from the table of the revived Free Ions held the company together until a very late hour.

Wednesday Session.

Laboratory Resistance Furnace.

The first paper of the Wednesday session was presented by Mr. G. R. White, of Ithaca, N. Y., who described a simple and cheap laboratory furnace useful for the production of calcium carbide, carborundum, etc., where not too much power is available. Samples of calcium carbide and carborundum were exhibited. Two types of cores for the resistor were described, first a carbon rod and second a core of crushed pressed carbon packed around with graphite powder for the sake of protection. Strong packing and a long time of heating are required in this case. Both the rod and the crushed carbon used for the laboratory furnace shown had been supplied by the National Carbon Co. The chief advantage of the construction is that it is cheap and enables one to make instructive experiments.

In the brief discussion which followed, Prof. Bancroft referred to the distinct circular line of siloxicon which is seen around the carborundum formed in such furnaces when it is broken up.

Electrochemical Processes as Station Load Equalizers.

A paper on this subject was presented by Mr. Elmer A. Sperry, of Cleveland, Ohio. He discussed the possibilities of coöperation of central station engineers and electrochemists. In a central station the peak load represents the investment, but since the peak load is only of very short duration the full equipment is utilized only for a few hours every day. This is what central station engineers call a poor load factor. The problem is to find uses for the electric current during the hours of light load, so as to improve the load factor, and the question suggests itself whether the current could be used for the manufacture of certain electrochemical products. During hours of light load central stations could easily afford to offer the power at a very low rate. As a matter of fact, central stations are now keen for business. Some times they want, however, the right to cut off the supply in an emergency case.

Mr. Sperry has made inquiries in several cities, and divides the central stations into three classes, according to the number of hours for which they would guarantee continuous current supply to electrochemical customers. Stations in the first class can supply current for 98 per cent of the whole year, those in the second class for 96.5 per cent, and those in the third class for 92.6 per cent.

From inquiries in different cities he has found that the rates which can be secured under such conditions may even be more favorable than from hydraulic plants. No transmission cost is, however, included in the prices, and it would be advantageous for a new electrochemical plant to locate right near the central station. Of course, such an arrangement would be specially suitable for truly intermittent electro-

chemical processes, and there are only a few of this kind. When no interruption is permissible storage batteries could be installed, but this would, of course, increase the investment.

In the discussion which followed, and in which Messrs. Ashcroft, Bancroft, Carrier, Howard, Richards and Sperry participated, it was first pointed out that the whole question is what are the rates that are offered by central stations. Mr. Carrier said that the best rate which could be obtained from the central station in Elmira, N. Y., was a little below 1 cent per kilowatt-hour. Mr. Sperry had been offered better rates, varying from 0.28 cent per kilowatt-hour up to 0.41 cent. There are no losses in conversion, and it should be noted that 0.31 cent per kilowatt-hour is $20 per horse-power-year, which is about the price at Niagara. Dr. J. W. Richards stated that almost all electrolytic processes are essentially continuous, but some electrothermic processes are intermittent, and they could be utilized. For instance, if a carborundum furnace could be operated so that the run takes 18 hours, it could be operated during the 18 hours of light load of a station.

Mr. Howard, of Boston, called attention to the fact that the large power stations have exhaust steam which they may sell for evaporating purposes. This would be an advantage for certain electrolytic processes. Dr. Bancroft referred to the Remington salt plant, where electric power is generated for light and traction and the exhaust steam, mixed with a little live steam, is used for evaporation. Mr. Sperry said that in Cleveland exhaust steam is transmitted two blocks from the station to the Cleveland Salt Co.

Electric Vacuum Furnace.

A very interesting paper on this subject was presented by Mr. W. C. Arsem, Schenectady, N. Y., who pointed out that a resistance furnace capable of working with a vacuum has distinct advantages, since the vacuum is good for the protection of the resistor and the temperature available is limited only by the evaporation of carbon. There is no possibility of chemical action of gases. He described a furnace developed by him in the research laboratory of the General Electric Co. It was already briefly described in our Vol. III., p. 196. It consists of an air-tight chamber having a refractory crucible placed centrally on a graphite stand. This crucible contains the material to be treated. Surrounding the crucible is the resistor of graphite, which is brought into the form of a tubular helix by cutting a helical slit into it (see our Vol. III., p. 218). It is provided at both ends with suitable terminals for the current supply. Around the heating resistor is placed a radiation screen consisting of an annular graphite tube filled with graphite powder. It prevents losses of heat by radiation. The crucible containing the material to be heated is preferably made of graphite, but magnesia crucibles and thorium oxide crucibles may also be used. A mica window on top of the furnace enables the observer to see the reaction.

In the furnace of the author it is easy to reduce the pressure to fractions of 1 millimeter of mercury, if all joints are tight. Alternating current is used with a rheostat. The conductors which lead to the resistor are made hollow, for the purpose of circulating water through them so as to keep them cool. The whole vacuum chamber is also surrounded by a water jacket.

The estimation of the temperature is most convenient, since there is a fixed relation between energy supply and temperature. By using the melting points of copper and platinum as fixed points it is easy to get at once the constants of the temperature scale. The relation is

$$n \log (y - 20) = \log a - \log x$$

where y is the actual temperature, 20 is the temperature of the cooling water, x are the kilo-volt-amperes. Up to 2,000° C. calibration by means of this formula is sufficiently satisfactory.

The principal source of error is variation [illegible] generator.

With the first type of furnace constructed it was found that the so-called Edison effect from the [illegible] caused considerable leakage of current from the crucible to the screen. This trouble was overcome later on by proper insulation of the screen and bottom supports.

It is now possible to reach with this furnace a temperature of 3,100° C., but in this case the life of the heater is very short on account of evaporation of carbon; 2,700° C. appears to be the vaporizing point of carbon in the vacuum. With temperatures below 2,000° C. the life of the resistor is practically unlimited. The furnace may be used for purifying metals in powdered form by fusing in vacuo, for determining melting points for synthesis and many other applications. In the discussion which followed all the speakers agreed that the furnace would undoubtedly prove very useful for many research purposes, and the remark was made that disadvantages of the construction of the design were the small capacity of the furnace (the crucible containing only a fraction of a pound), and the necessity of waiting until the material solidifies before it can be removed from the furnace. This brought out the reply from Mr. Arsem that a much larger furnace is in course of construction and one which will alloy the casting of the melt.

Electrolytic Corrosion of Copper-Tin Alloys.

A paper by Mr. B. E. Curry on this subject was remarkable in view of the enormous amount of experimental work which it must have required and in view of the experimental confirmation of Prof. Bancroft's general principles of electrochemical and chemical corrosion (see his address above). Mr. Curry first gave the equilibrium diagram for copper-tin alloys for temperature between 1,100° and 0° C., and for alloys containing from 100 per cent copper to 100 per cent tin. He then gave the results of his measurements of electrolytic corrosion in the following electrolytes: Sodium sulphate, sodium nitrate, sodium acetate, alkaline sodium tartrate, acidified ammonium oxalate and sodium choride solutions. The electrolytic corrosion was determined for each solution, and a diagram plotted giving the current efficiency of corrosion as ordinate with the composition of the alloy as abscissa, the composition varying from pure copper to pure tin.

For a sodium sulphate solution the curve looks like this: the current efficiency is 100 per cent for pure copper, when tin is gradually added, the efficiency decreases very slowly until at a certain point there is a sudden large drop; with further increase of tin the efficiency continues to decrease slowly until in a certain region there is almost zero corrosion (the passive state); with further increase of tin the current efficiency rises again to about 100 per cent for pure tin.

The curves for all the solutions studied show at certain points sudden drops in the efficiency curves. These are sometimes coincident with the formation of a new phase but sometimes not.

With a sodium chloride solution the corrosion efficiency is practically 100 per cent throughout. That means that all the bronzes dissolve in the chlorides.

The passivity was found in all cases to be due to a film. The film was removed and analyzed for copper and tin. Quite some copper was found but the film consisted principally of tin oxide with a small amount of copper in it.

To check the results some parallel tests were made on chemical corrosion extending over fifteen days.

As Dr. Bancroft pointed out in the discussion, all the evidence that we have is in favor of a perfect correlation between chemical and electrochemical corrosion. Prof. Miller referred to the sudden drop in the corrosion efficiency which sometimes occurs by very slight changes. Dr. Bancroft said that this is due to the formation of a protective oxide. This is the only reason he knows of.

Corrosion of Iron by Acids.

A paper by Prof. C. F. Burgess, of the University of Wisconsin, and S. G. Engle, gives the results of a study made under a grant from the Carnegie Institution of Washington. The corroding mediums employed were normal solutions of sulphuric and hydrochloric acids. Distilled water and chemically pure acids being used. The solutions were kept at room temperature (about 22° C.), and during corrosion additions of acids were made to keep the solutions at a nearly constant content of free acid.

Fig. 17 shows the relative rate of corrosion of five different samples of iron for successive hours during the progress of the tests. The striking facts which are brought out in the curves are that electrolytic iron (made by the process described in our Vol. II., p. 183) in the condition which it possessed just as it was taken from the tanks, corroded at a far greater rate than did the other samples, about six times as rapidly as the cast iron, four times as rapidly as the steel, and nearly forty times as rapidly as the transformer iron. It is also shown that the heat treatment to which the electrolytic iron was subjected, conferred upon it the property of resisting corrosion to a great degree, the heated electrolytic iron dissolving at a hardly appreciable rate as compared with the samples which had been unheated.

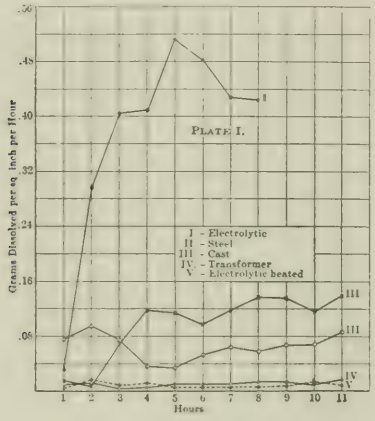


Fig. 17—Corrosion of Iron.

The rate at which an acid dissolves iron depends, of course, largely upon the elements with which the iron is associated. It is well known that the various metallographic constituents of iron and its alloys resist in differing degrees the attack of acids, and, consequently, different grades of iron show differences of durability.

The percentage of purity is, however, not the controlling factor, as is here shown by the observations that the heated and unheated specimens of electrolytic iron, essentially the same as far as chemical composition is concerned, exhibit the widest possible variations in rate of solution.

The crystalline or granular structure of iron seems to influence rates of corrosion to a marked degree.

The rapidity with which unheated electrolytic iron liberates hydrogen in a dilute acid solution suggests a useful application of this material in the production of pure hydrogen by replacing zinc, which is now commonly used. Iron and equal surfaces of exposed metal, electrolytic iron liberates hydrogen about four times as rapidly as does pure zinc, and twice as rapidly as the commercial grade.

Another advantage in the use of electrolytic iron in hydrogen generation lies in its higher purity, and the consequent increased purity of the resulting gas, as compared with that which is obtained from zinc. One pound of iron produces 16 per cent more hydrogen than does 1 pound of zinc, and if a sufficient demand for iron for this purpose should arise, it could be supplied at a cost materially less than that of zinc.

Traces of arsenic are shown to exert such a marked influence in protecting iron from the corroding action of acids that the employment of this element for protection against ordinary corrosion appears worthy of further investigation.

While the measurement of electric potentials of iron against corroding agents seems to afford some indication as to the rapidity with which it will become attacked, it does not seem possible to establish definite relations between electric potentials and corrodibility.

In the discussion which followed it was suggested by Prof. Bancroft that to test the possibility of the protection of iron against corrosion by means of arsenic it would be necessary to produce a thin film rich in arsenic on the surface of the iron. Mr. Hering asked whether it is the hydrogen contained in electrolytic iron which gives it those special properties. Prof. Burgess replied that there is no exact proof. Mr. Arsem had heated electrolytic iron in vacuo, and it was then no longer brittle but had poor mechanical properties, due to the presence of oxide. He thinks that the oxide plays a greater part than the hydrogen. Mr. Little asked whether hydrogen made from iron would be more expensive than the electrolysis of water. Prof. Burgess replied that if the demand should be greater, electrolytic iron could be produced very cheaply. Mr. Rodman stated that to get hydrogen gas for some purposes, like lead burning, ordinary iron is now already used in connection with potassium permanganate.

Development of the Nickel Plating Industry.

A paper by Mr. Isaac Adams (well known as the "father of electroplating") gave a sketch of the development of the nickel-plating industry in the sixties, with some interesting and amusing personal reminiscences. The paper is concluded by the remark that the art is to-day carried on substantially as in 1869-70, with the same solutions, the same anodes and the same details of shop manipulation. This remark was resented in the discussion, but nothing was brought forward to disprove it.

Prof. Bancroft said that rolled anodes had formerly proven unsuccessful, but that there is a great opportunity of improving nickel-plating. In the solution now generally used nickel is passive, hence it would not corrode. For this reason it is now general practice to use cast nickel anodes containing 6 to 10 per cent of iron. This yields corrosion, but introduces other difficulties. Prof. Bancroft thinks that all our nickel-plating contains iron, and remarks that all the nickel plating on Weston instruments rusts red in the chemical laboratory. It is never safe to leave nickel-plated bicycles out over night. He, therefore, suggests to take a pure rolled nickel anode and to add a slight percentage of nickel chloride to the usual electrolyte. This yields corrosion, and results in pure nickel deposits. Prof. Burgess said that nickel plating from a chloride solution is much softer, but Prof. Bancroft replied that he did not suggest the use of the chloride solution but only to add a little chloride to the electrolyte.

Lead from Acetate Solutions.

A brief note was presented by Mr. Ralph C. Snowdon, the object being to show that it is quite possible to get a good plating deposit of lead even from an acetate solution, and that the favorable conditions are the same for lead as for other metals, although he does not recommend the lead acetate solution for commercial purposes. He used a cathode revolving at a speed of 2,500 r. p. m., and added 1 gram of gelatine per liter to a solution of lead acetate and acid, normal with re-

spect to each. The temperature was kept at 30° C., the cathode density was 1.5 amp. per square decimeter, and a lead anode was used. A matted deposit of lead was obtained, very adherent and entirely satisfactory in every way.

The discussion which followed related to the question of the underlying reason of the beneficial effect of an addition of gelatine, which has been found so useful on a commercial scale by Betts.

Structure of Electrodeposits.

A paper by Prof. C. F. Burgess and Mr. O. P. Watts was read by Prof. Burgess, who exhibited numerous lantern slides showing the structure of electrolytic iron deposits. The research was made in connection with an investigation of electrolytic iron under a grant from the Carnegie Institution at the University of Wisconsin.

The lantern slides exhibited illustrated numerous important factors which influence the physical structure of deposits. Thus lack of circulation may cause vertical grooving, due to the rising of dilute columns of electrolyte upwards. Gases dissolve in the electrolyte and give rise to pitting. It has often been believed that hydrogen bubbles are mostly responsible for this effect, but the authors have found that air is more troublesome. The pitting effect may be prevented by boiling the electrolyte and keeping air off. The combined pitting and grooving effect was also shown. Then the peculiar warts, sometimes appearing on deposits, were shown; these are growths perhaps due to little pieces of dust coated over. These growths are not merely a surface effect but extend inside. By a sharp blow of a hammer these warts (like teeth) may be detached from the plate, leaving cavities.

In the concluding part of the paper some analogies were pointed out between the structure of electrolytic deposits and the structure of certain geological deposits of hematite, magnetite and various other ores from solutions. This suggests a method of studying geological problems in the laboratory, since with the aid of the electric current the growth of a deposit may be investigated in a short time when the corresponding geological deposit in nature requires a very long time.

Cadmium Standard Cell.

The Weston cadmium cell has become the pet standard cell in preference to the Clark cell for all ordinary laboratory work on account of the very low temperature coefficient of the former. It has repeatedly been suggested to authorize it as the fundamental primary standard.

Some six years ago Cohen, in Amsterdam, made a strong attack on the reliability of the cadmium cell as primary standard, but his arguments were exploded. Later on small troubles were still noticeable, but they were traced to the preparation of the mercurous sulphate, and it will be remembered that at the Washington meeting of the American Electrochemical Society an electrolytic method of preparing mercurous sulphate was described in two different independent papers, one by Dr. Wolff, of the Bureau of Standards, and the other one by Prof. Henry S. Carhart and Dr. Hulett (our Vol. II., pp. 174, 176, 407). It seemed at that time, and everybody hoped that all the difficulties with the cadmium cell had been overcome.

The paper which Dr. G. A. Hulett (Dr. Carhart's old co worker in this field and now at Princeton University) presented at the Cornell meeting, contains a great deal of strict evidence to the effect that those hopes were premature. It should be emphasized here that Dr. Hulett's paper is quite conservative, and that he does not make any claim like Cohen six years ago. But since putting up the first cadmium cells with electrolytic mercurous sulphate with Prof. Carhart, he has continuously followed up this research, and produces in the paper a great deal of evidence showing a drifting of the e.m.f. of the cadmium cell with the time. It is light, but [illegible] It is also evident from his research that the trouble is not with the mercurous sulphate, but must be with something else in the combination. It appears from his record that the cathode leg, as at present constructed, is not in a state of equilibrium. While the cadmium cell will always remain an ideal secondary standard it, therefore, would seem that, at least, with its present construction it would be premature to adopt it as a primary standard.

The paper was discussed by Messrs. Bancroft, Miller and Patten. Prof. Miller suggested that the trouble might be due to a reaction of the gases used at the cathode leg by Dr. Hulett did not think so.

Electrodeposition of Bronze.

A paper presented by Mr. B. E. Curry, dealt with the electrodeposition of bronze—the reverse problem of his corrosion experiments covered in the paper abstracted above. While chloride solutions give throughout almost perfect corrosion, it is impossible to get good electrodeposition of bronze from them. The best deposits were obtained from an ammonium oxalate solution slightly made acid with oxalic acid. The anodes were copper-tin alloys of varying composition.

The results of the author show that he was unable to get a good bronze deposit containing much less than, say, 80 per cent of copper. To get deposits containing less copper one needs solutions very low in copper, and the operation is easily thrown out of order. On the other hand, it is easy to get bronze deposits containing between 80 and 100 per cent of copper. This research, like that on corrosion, was made at Cornell University.

The paper was discussed by Profs. Bancroft and Miller. Prof. Bancroft pointed out that the deposition on the cathode was not the reverse process of the solution at the anode, since the corrosion of the anode depends on the anion while the cathodic precipitation depends on the positive metallic ions in the solution.

Ferromanganese Anodes in Alkaline Solutions.

The last paper of the Wednesday session was a brief note by Mr. G. R. White, of Cornell. According to Lorenz, a ferromanganese anode dissolves as permanganate. Mr. White found the green manganate. This is always formed at 95°, due to a secondary reaction, namely, the reducing action of nascent hydrogen at the cathode. Any reduction to the green manganate at ordinary temperatures is due to organic acids in the solution.

Wednesday Afternoon.

After the Wednesday morning session, as after the session of Tuesday, lunch was served by the courtesy of the University authorities in the Lecture Hall. Afterwards an excursion was made to the new Physical Laboratory (Rockefeller Hall) of the University, the splendid hydraulic laboratories, the power plant in the gorge and the works of the Ithaca Gun Co.

In the evening a subscription dinner was served in the Dutch Kitchen of the Ithaca Hotel, which was attended by some fifty members. The dinner was entirely informal and was very greatly enjoyed. Mr. H. B. Cushman was a splendid toastmaster, and introduced the numerous speakers in pointed and witty remarks. Among the speakers were President W. D. Bancroft, who gave a review of the work of the Society during the last year; Prof. J. W. Richards, who emphasized the important progress now going on in the electrometallurgy of iron and steel, Mr. E. A. Acheson, Mr. C. L. Acker, Prof. C. F. Burgess, Mr. Carl Hering, Mr. A. H. Cowles and Mr. C. E. Acheson. Mr. Acheson made a spirited speech, referring to the [illegible]

press in order to get legislation which would prevent the further use of the power of Niagara Falls for electric developments. He strongly emphasized that the general public alarm was unwarranted. Several of the speakers were introduced by little clever songs of the Mr. Dooley type, the alleged poets being Mr. H. B. Coho and Mr. H. T. Matthew.

After the banquet there was another informal reunion, with more singing, at "Zink's."

Thursday Session.

Colloids.

The first paper of the Thursday session was presented by Dr. H. E. Patten, of the Bureau of Soils, of Washington, D. C., on the migration and flocculation of colloids, considered as an adsorption phenomenon. The problem interests the Bureau of Soils, since the getting of a soil into proper physical condition in all probability involves the coagulation phenomenon.

Dr. Patten discussed the problem briefly, and stated that the paper was more philosophical than experimental, and that the object was to get a working correlation of the facts involving as few hypotheses as possible.

He first refered to the fact that it is possible to get a charge of electricity on a diaphragm and one on the liquid simply by filtering the liquid through the diaphragm. The converse of this phenomenon is found in electrical endosmosis, where a charge on the diaphragm and an opposite one on the liquid produces mechanical motion, the liquid passing through the diaphragm. These two processes take place for very poorly conducting liquids, and hence in no way require for their explanation the assumption that ions exist in these liquids.

Secondly, where solutions of salts, such as potassium chloride, are filtered through soils, through finely powdered solids, through a coagulated colloid, or even through gelatine, the salt will be split up, the potassium is retained and the solution gives an acid reaction.

Thirdly, when we have a manifestation of electrolysis it always occurs at the boundaries. The action of these small particles of fine suspensions in liquid media in retaining acid or basic radicles selectively, is like the separation of the products of electrolysis at the electrodes. Where such a splitting of a salt takes place we would expect the particle of solid to acquire a charge of electricity, just as the electrode acquires it. And where an electrical pressure is placed upon the liquid containing such a particle, it would, of course, migrate under this difference of potential.

Fourthly, we have the fact that in general suspended particles migrate in acid solution towards the positive pole and in alkaline solutions toward the negative pole.

These different phenomena hold together, and this gives us a working correlation of the facts.

In the discussion which followed, Mr. Hering referred to the possibility of seeing the migration of the particles under the microscope, and Dr. Patten confirmed this with the restriction that this is possible if the particles are large enough.

Fused Sodium Peroxide for Regeneration of Air.

A paper by Messrs. George F. Reinsch and Richard von Foregger was then presented in abstract, in the absence of the authors, by Mr. S. S. Sadtler.

In our April issue, page 128, as noticed already briefly, a preliminary paper presented in March by Dr. von Foregger before the New York Section of the American Electrochemical Society, in which the speaker made some interesting suggestions on the purification of the atmosphere of the New York subways by means of oxone. The present paper gives a full report of the experiments of the two authors on the regeneration of air by oxone with special reference to its use on submarine boats. (See also our May issue, p. 166.)

As has been repeatedly noticed in our columns (for instance, our Vol. III., pp. 255 and 377), oxone is fused sodium peroxide, and when brought in contact with water it yields the reaction

$$Na_2O_2 + H_2O = 2NaOH + O$$

The oxygen thus produced may be supplied to the lungs of living beings in a closed space, while the use of oxone has the further advantage that the carbon dioxide given out from the lungs is taken care of by the further reaction

$$2NaOH + CO_2 + Aq = Na_2CO_3 + Aq$$

For these reasons oxone is well adapted for the regeneration and purification of air, especially air that cannot be conveniently renewed by ventilation, and therefore has to be breathed over and over again. The object of the tests of the authors was to make this process entirely automatic, and their results show that it is quite possible to keep nine men alive in a submerged submarine for a period of 12 as well as 48 hours or more, probably within the full radius of action of such a vessel, depending merely upon the supply of oxone.

An average man requires 25 liters of oxygen per hour, and as 100 grams oxone furnish 13 liters of oxygen, 192.5 grams would give 25 liters. In other words, 1 kg. of oxone would sustain a man for 5 hours and 12 minutes, provided, of course, that he has to breathe the same air over and over again. These figures agree almost exactly with those obtained in the experiments of the authors.

Most of the experiments described by the authors were made with strong healthy rabbits, averaging in weight 7 pounds,

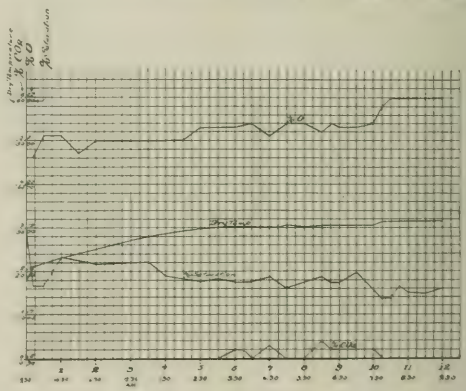


Fig. 18.—12-Hour Run with Six Rabbits.

and although the weight of a single rabbit was only one-twentieth that of a man, the authors found that one man exhaled as much carbon dioxide as ten or twelve rabbits. The experiments were made in a strong wooden box, 3 × 3 × 6 feet, containing a plate glass window, which also served as an entrance, and which could be hermetically sealed. Inside the box was placed the apparatus, consisting of an electrically-driven blower, so arranged that its discharge was connected to a horizontal sheet-iron cylinder, in which was placed the oxone.

In addition to this was a cyclometer, an incandescent lamp, wet and dry bulb thermometers, and the necessary food for the rabbits. An Orsat apparatus was placed outside on top of the box, and connected with the inside by means of a glass tube. A wash bottle was also placed on top, and connection made with the inside. The wash bottle enabled the experimenters to introduce water if necessary, and the fluctuation of its water level when gas samples were taken indicated whether the box was air tight or not.

In the first experiment of the authors they found that after the lapse of a certain time all reaction of the oxone ceased, owing to the formation of a crust of carbonate on the surface

of the oxone. This difficulty was successfully overcome by a mechanical agitator working the wire-netting cage which contained the oxone, and so arranged that the rate at which the cage was agitated could be regulated. The results were entirely satisfactory, and the device did not in any way interfere with the automatic nature of the process.

Of the numerous interesting diagrams given in the original paper we can reproduce only two on account of limitations of space. The first, shown in Fig. 18, represents the results of a run of 12 hours and 5 minutes, with six rabbits, 2½ kg. of oxone being used. This was the first run with the agitating device. The barometer was 750 to 752 mm., the outside temperature 14.4° to 21.0° C. It will be noticed that at the end of the 12 hours the oxygen was slightly above normal, but steady at 22 per cent, and the CO_2 was down to zero. One-half of the original oxone still remained in the cage at the end of the experiment. The slight increase in oxygen in this case was accidental, and due to the falling of small pieces of oxone through the wire netting of the cage into the caustic paste accumulated in the outside cylinder, and there acting on the water, generating an excess of oxygen. The occasional appearance of CO_2 in this run was entirely due to the attempts of the experimenters to determine the slowest rate at which the cage could be agitated.

Fig. 19 represents the results of a test in which one of the two authors entered the sealed box, to play the role of the rabbits in the previous runs. The test lasted 5 hours and 35 minutes, and 1 kg. of oxone was in the box. The barometer was 734 mm., and the outside temperature 21° C. The whole test is, in fact, extreme, since the man was sealed up in a space not over four times the size of his coffin would be. One of the interesting features of the test was the smoking of cigarettes by the man inside. The various points where the CO_2 curve goes up from 0 to 0.2 and 0.3 per cent, shows the times cigarettes were smoked. Another most noticeable thing was the absorption of the smoke and the drop of CO_2 to zero again within 15 minutes after the extinction of the cigarette. When the box was opened there was not the faintest smell of cigarette smoke. The man inside asserted that throughout the run he felt exceedingly well, in fact, better than before he entered. He described the conditions as being similar to a hot, dry summer's day.

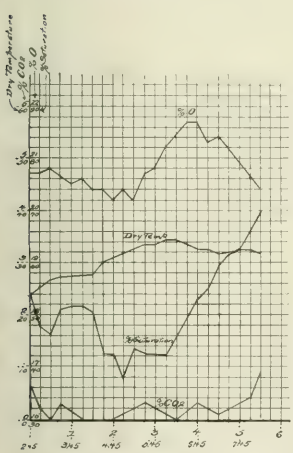


FIG. 19.—5½-HOUR RUN WITH ONE MAN IN BOX

The marked increase in humidity is in all probability due to the fact that for a few moments a wet towel was held in front of the fan inlet and 50 cc. of water placed in the bottom of the outside cylinder during the third hour's run. The presence of this water on the bottom of the outer cylinder was again a disadvantage, owing to small pieces of oxone falling into it and producing an increase in oxygen.

After 4½ hours nearly all the oxone had gone, only a few small pieces remaining in the cage, and the oxygen started to go down.

How quickly the air becomes vitiated in such confined spaces is shown by the fact that during the short time it took to get the man sealed up and the motor started, 0.6 per cent CO_2 had accumulated in the box. It will be noted, however, that within 30 minutes this was completely absorbed.

The final test of the authors shows curves not reproduced here of an exceedingly smooth and regular nature, since in this test the experience gained in previous runs was made use of.

In the conclusion of their paper the authors apply the results of their tests to the case of a submarine boat, the capacity of which is on an average 2,700 cubic feet, or 75,000 liters. With 9 kg. of oxone in all, nine men would be perfectly safe for fully 9 hours, and with 18 kg., or 40 pounds of oxone, they would be safe for at least 14 hours, so that if 40 pounds of oxone would supply nine men in a submarine boat for 14 hours, one man would require practically one-third of a pound.

In the discussion which followed, Mr. H. Howard, of the Merrimac Chemical Co., of Boston, Mass., referred to a case in which he had saved by means of oxygen the life of a man who had been overcome by inhaling gas. Respiration was restored by the use of oxygen generated from sodium peroxide and water. Mr. Ashcroft brought up a physiological question, and emphasized the great importance of the problem of regenerating air, while Mr. Carrier claimed that a catalytic agent is necessary to prduce the desired reaction between fused sodium peroxide and water, and that such a catalytic agent is really contained in the fused sodium peroxide sold as oxone.

SWITCHBOARD.

The next paper on the programme, on a lecture-room switchboard, by Prof. WILDER D. BANCROFT, was taken as read, since Prof. Bancroft had already explained the switchboard when the members visited the laboratory.

ALTERNATING-CURRENT ELECTROLYSIS WITH CADMIUM ELECTRODES.

Mr. G. R. WHITE, of Cornell, then presented his paper on a special case of alternating-current electrolysis. When an alternating current is passed through a solution of sodium hydrosulphite with cadmium electrodes, a precipitate of cadmium sulphide is formed. This problem has been studied by Richards and Roepper, who have taken out a patent covering this phenomenon as a process for the manufacture of "cadmium yellow."

Le Blanc and Schick have also studied the problem, and have determined efficiencies with various frequencies; they found that at the higher frequencies the efficiency drops practically to nothing. The present author shows that their efficiency can be very much increased. The general results are that in spite of a better efficiency its value is still too low for a commercial process. The precipitate, moreover, is apt to be polluted with bits of the brown oxide film, which would be objectionable in a commercial product, and render special treatment for its removal necessary. The efficiency is practically independent of the current strength and also independent of temperature and of stirring. The efficiency depends almost entirely on the electrode surface, but in spite of attempts of the author to produce a uniform electrode surface the quantity dissolved varied widely.

The paper was discussed by Messrs. Muller, Patten, Bancroft and Richards. Prof. Bancroft pointed out that it will be impossible to trace exactly the influence of all other factors as long as it is not possible to make a reproducible electrode so as to fix the chief influence.

ELECTROLYTIC CHROMIUM.

A paper by Prof. MAX LE BLANC, of Karlsruhe, Germany, was presented in abstract by Mr. C. L. Carrier, Jr. It is mainly a reply to some criticisms of Le Blanc's work, con

tained in papers by Carveth and Mott, and by Carveth and Curry (our Vol. III., pp. 176 and 273). The chief bone of contention is in regard to the patents of Placet and Bonnett, who stated that chromium could be electrolytically deposited in large quantities by electrolysis of a solution of chrome alum, to which had been added alkaline sulphates, plus sulphuric acid. Mr. Schick, working under Le Blanc, followed the patent specifications very closely, but could only get traces of chromium. Le Blanc concluded that a sample of chromium of 1 kg. weight exhibited by Placet to the French Academy of Science, was not obtained as stated, or that at least the essential condition of its production was omitted from the specification. It is to this conclusion that the above-mentioned gentlemen take special exception.

According to Prof. Le Blanc, the real question at issue is, "can chromium (of any desired thickness) be deposited from a solution of chrome alum in the way described by Placet and Bonnett?" This has not been answered by Carveth, Mott and Curry, for they did not work exactly according to Placet and Bonnett, and, moreover, their runs were so short that only a few milligrams of chromium were deposited.

Le Blanc has no doubt that chromium of any desired thickness will some day be deposited, and the work of Neumann and Glaser, supplemented by the three above-mentioned gentlemen, points in the right direction, but the problem is by no means solved. At least no precise data are available by which chromium can be deposited of any desired thickness.

Le Blanc then gives a review of extended work of Mr. A. Ruppert, who, at his request and under his supervision, endeavored to duplicate the work of Carveth and Mott. He discusses the best conditions for getting electrolytic chromium, the object being to obtain thick deposits on large scale. The best results were obtained with the following system: lead forming the positive pole, porous cup, chromium sulphate solution with an addition of boric acid, copper forming the negative pole; it was not rotated since the deposit was smooth without rotation, and since rotation seemed to have little effect on the efficiency. There was apparently rather a tendency to make it poorer on account of continuous removal of chromium sulphate from the surface of the negative plate.

Under these conditions the deposit was good at first, but cracks began to form as chromium became thicker, and finally tended to scale off.

The chief conclusions of the author are that thin deposits of fine quality of chromium can be plated on brass, copper, carbon; the thickest coat obtained was approximately 0.13 mm. Placet cannot have made his 1 kg. chunk of chromium which he presented to the French Academy in the way he claimed. This conclusion is reached from the character of the deposit.

In the discussion which followed, Dr. Carveth expressed his appreciation of the courtesy of Prof. Le Blanc to send his paper to the meeting at Cornell, where Carveth's own work had been done. While he will contribute a longer communicated discussion to the "Transactions" later on, he briefly stated that he had not considered that the production of any desired thickness was so very important, but he knew that his deposits were thicker than 0.13 mm. Prof. Bancroft suggested that Le Blanc should have used chromic acid as electrolyte. It works very well. Some samples of good deposits were exhibited. Prof. Bancroft said he could not believe that it is impossible to get deposits of any thickness. It is only necessary to maintain the original good conditions constant during the run. (as pointed out by David H. Browne, our Vol. I., pp. 338 and 386), and this is quite possible with chromic acid.

Mr. Curry stated that he had made good deposits 1 mm. thick at Cornell. Mr. Carveth pointed out that there was a difference in the two positions which Le Blanc, on one side and the Cornell people on the other side, were endeavoring to solve: the former wants to make metallic chromium just as Borgeaux makes electrolytic iron, and on the other hand, the latter try to get good electrical coatings for plating purposes.

Free Energy of Various Compounds.

A paper by Dr. M. DeK. Thompson, on the free energy of some halogen and oxygen compounds computed from the results of potential measurements, was then presented in abstract by Prof. Miller.

Dr. Thompson points out in the introduction that one of the most important chemical problems of the present time is the determination of the free energy of formation of compounds from their constituents, for it is this quantity and not the heat evolved that is the true measure of chemical affinity. No reaction can take place of itself that is not capable of doing external work, and the maximum amount of such work which a reaction can produce at constant temperature is called the free-energy-decrease of the reacting system. The author has computed this value for a number of compounds from potential measurements.

The chief results are contained in two tables, one of which gives the free and total energies of the solid halides of silver, mercury, copper, thallium and lead at about 18° C. Some interesting details of this chief table are still further condensed in the following summary of the free energies per equivalent of salt:

	Cl.	Br.	I.
Hg (ous)	24,700	21,400	13,400
Ag	25,000	23,000	15,900
Cu (ous)	28,100	23,700	16,600

This table means, for instance, that the free energy of Hg_2Cl_2 is 49,400. The table shows that the corresponding halides of mercurous mercury, silver and cuprous copper have free energies which differ from each other by less than 3,500 calories per equivalent, but which increase slightly in every case in the order mercury, silver and copper.

A second table contains the results for other compounds than halides, and is given below, requiring no explanation. It gives the free and total energies of the oxides of silver and mercury, of water, ammonia and hydrochloric acid at about 18° C.

Compound	Free Energy	Total Energy	Ratio of Free to Total Energy
Hg_2O	13,400	22,200	0.60
Ag_2O	3,900	5,900	0.66
H_2O (liq)	55,500	67,000	0.82
H_2O	53,800	57,500	0.93
HCl	22,300	22,200	1.01
NH_3	45,200	11,400	4.00

In the case of the solid salts investigated the ratio of the free to the total energy does not differ from unity by more than 7 per cent, except in the single instance of silver iodide. This result is of great practical importance, since it enables for approximate purposes the usually well-known heat of formation to be employed in the place of the often unknown free energy in the study of the equilibrium of reaction involving solid substances.

The paper was briefly discussed by Prof. J. W. Richards and Prof. Miller. Prof. Richards thought that the author laid too much stress on the importance of the free energy compared with that of the total energy. He also believed there was some hypothesis involved in calculating the free energy of the formation of water from the oxygen-hydrogen gas cell, the effect of the platinized platinum electrodes not being considered at all. Prof. Miller replied with respect to the relative importance of free energy and total energy that it all depends what one wants to calculate. If one wants to use the principle of conservation of energy, of course the total energy must be used; on the other hand, if one wants to know in which direction a reaction will go on or which one of several reactions will take place, the total energy does not give any exact information, and one has to refer to the free energy

Electrolysis of Caustic Soda.

A paper presented by Prof. J. W. Richards gave a useful and concise historical review of the various processes for electrolyzing caustic soda for the production of sodium, which have been either proposed or introduced into practice. Davy, in 1808, was the first to electrolyze caustic soda. Castner solved the industrial problem. The latter work of Rathenau, Becker, Carrier, Ashcroft, Scholl and others was discussed, and reference made to the research of Le Blanc and Brode as the best scientific investigation on the subject.

The paper was discussed by Messrs. Ashcroft, Carrier, Carveth, Forssell, Patten, Richards and Townsend. Dr. Forssell had once found globules of metallic sodium in a concentrated solution of caustic soda which he had electrolyzed the day before. In order to see whether these globules were protected by means of a thin coating he cut them into small pieces, but they did not react with the solution. Others had observed the same phenomenon. Dr. Richards confirmed Dr. Forssell's remark that sodium can be plated out from an aqueous solution.

One of the special results of the investigation of Le Blanc and Brode was the object of a remark of Mr. Carrier. They had found that at the anode hydrogen is really evolved together with oxygen, which sometimes results in explosions. Mr. Carrier used later the original apparatus of Le Blanc and Brode, and found that the above result was due to the construction of the apparatus, namely, to a bipolar effect of a screen. When the screen was removed the explosions stopped.

Potential Differences in Aqueous and Non-Aqueous Solutions.

A paper by Prof. Louis Kahlenberg, of the University of Wisconsin, and Mr. A. S. McDaniel, was then presented in abstract by Dr. Patten. The object of their investigation was to measure the differences of potential developed between electrodes of lead peroxide and manganese peroxide and solutions in various non-aqueous solvents, and to compare these results with those found when water is substituted as solvent, all other factors remaining the same. Among the solutions studied were pyridine, acetone, amylamine, when free from water, and when containing water in various percentages. The Ostwald calomel electrode was used. The results are given in tables and diagrams, and for details the reader must be referred to the original paper.

The chief conclusion which the authors draw from their results is that the difference of potential between a peroxide electrode and the solution into which it dips is to be ascribed to the chemical affinity; *i. e.*, the chemical strain or potential between electrode and liquid.

The paper was briefly discussed by Messrs. Arsem, Hunter and Miller.

Silver Coulometer.

A paper on a new electrolyte for the silver coulometer (or voltameter) by Prof. Henry S. Carhart, of the University of Michigan, and Dr. F. W. Willard, was presented in abstract by Mr. S. S. Sadtler.

The authors recommend silver perchlorate as electrolyte, and it is stated to eliminate the disadvantages of the nitrate. Silver nitrate is treated with perchloric acid and the nitric acid is driven off. A comparison was made between the use of earthenware diaphragms and filter paper, and the authors find that filter paper, as used by Dr. Guthe, is more satisfactory. The silver deposits have a very uniform appearance.

Errors in Pyrometry

The last paper of the session was on errors in pyrometry, Mr. E. S. Shepherd, of Washington, being the author. In the absence of the author it was presented in abstract by Prof. Bancroft.

After some notes on error in the use of [illegible] carelessness in handling thermocouples, the [illegible] his belief that chemists are too apt to [illegible] be used in the same haphazard manner as a [illegible] thermometer. As much as an error of 15° C. may be introduced simply by contamination of the [illegible] Further, the galvanometer needle may stick. [illegible] important that the cold junction of the thermoelement is kept at constant temperature. On the way to which some people use thermo-electric pyrometers the temperature of the cold junction sometimes varies by 1° to 20° from that for which the instrument was calibrated.

The thermoelement and the material which is [illegible] must be in very intimate contact. The temperature at various places of an ordinary platinum furnace is apt to vary from 10° to 50° in different parts of the furnace, and in a gas furnace the variation is even greater.

All metals have an appreciable vapor tension, and this vapor attacks the thermoelement, resulting in incorrect readings.

The temperature at which a reaction begins is an indefinite one in nearly all cases. The time factor is very important. For example, a certain mixture of lime and silica will combine slowly at 1,200°, more rapidly at 1,300° and fairly rapidly at 1,400°, while the melting point of this particular mixture is 1,512° C. It depends altogether on how much time is given whether or not one will conclude that the reaction takes place at any one of the above temperatures.

It is commonly stated that the temperature of formation of a slag lies above its melting temperature, but this is only an accident of shop practice. If the materials are finely ground and intimately mixed, combination will take place below the temperature of fusion, and the observed phenomenon is due entirely to the coarseness of the materials, rendering combination difficult.

At high temperatures, on account of large radiation, large variations of temperatures exist within the furnace.

The last part of Mr. Shepherd's paper deals with possible errors in radiation pyrometry. One fundamental error which can be made if the user does not understand the theory of the instrument is possible with pyrometers based upon the theory of radiation from "black bodies." If the bodies do not radiate as "black bodies" in the thermodynamical sense the instrument does not give the exact temperature. Further in certain cases when dense vapors are given off it is folly to try to determine the temperature of the substance which is emitting the vapor. In carbon tube furnaces, for example, there is a [illegible] carbon flame playing about the mouth or in the neck of the furnace, which may be several hundred degrees different in its temperature from that of the middle of the furnace.

The interest which is being taken at present in pyrometry was indicated by the lively discussion which followed. Mr. F. F. Schuetz, of New York, spoke at some length on the part of Mr. Shepherd's paper dealing with the thermoelectric type of pyrometer. Mr. Schuetz considers that the thermoelectric pyrometer is destined to be of the most important form for temperature measurements up to 3,000° F. or 1,600° C., being capable of both indicating and recording temperatures and adaptable to a great number and great variety of industrial applications. He emphasized that for industrial purposes a pyrometer should be used which eliminates the personal equation. With respect to the contamination of the elements composing the couple through contact with the [illegible] which it is inserted, or through vapors given off from [illegible] etc., he thought that it is not necessary to use [illegible] and platinum-rhodium at temperatures below, say, 2,600° F., and that better results can be obtained with cheaper elements composed of such materials as nickel, steel, [illegible], [illegible], constantine, nickel steel, german silver, etc. Such couples if properly protected and insulated, give good results. For this purpose each element should be [illegible] with silk thread and then coated with carborundum paint, using sodium

silicate as a binder. For protection against temperatures above 2,000° F. special tubes of nickel, plumbago or porcelain may be employed. For use in lead baths and the like the asbestos-carborundum-insulated couples may be further protected by an iron pipe welded together at the exposed end. These methods are used in the new Bristol low-resistance pyrometer (which was described in detail in our March issue, p. 115).

The Bristol pyrometer also avoids in an ingenious way the necessity of keeping the cold anode artificially at a constant temperature, since it contains an automatic compensating device (which was fully described in our March issue). Further, with the same instrument, there is no difficulty of the galvanometer needle sticking, since this is not likely to occur in a low-resistance system, owing to the greater motive power behind the needle. It is possible with this instrument also to readily estimate the temperature to 1° F. With respect to the point made by Mr. Shepherd, that the temperature at different parts of the furnace may vary, Mr. Schuetz recommends to place a number of couples at various parts of the furnace, to connect them to a common indicating instrument and to use a suitable switching device, by means of which the temperature of practically every point of the furnace may be quickly determined. Any lag, error or slowness in reaching thermal-equilibrium is to a great extent overcome by reducing the cross-section of the hot end of the couple, a practically instantaneous response to variations of temperature then being observed. Mr. Schuetz concludes that the thermoelectric pyrometer is now a thoroughly commercial and reliable instrument, able to withstand ordinary wear and tear and capable of being entrusted to workmen of ordinary intelligence.

In the further discussion, in which, among others, Messrs. Sperry, Burgess, Cowles and Bancroft participated, several speakers stated that they had used the Bristol pyrometer and had found it very useful and convenient, and that at the price at which the instrument is furnished quite a number of couples could be employed in practice. Prof. Burgess urged that the seller of thermoelectric couples should inform the purchaser concerning the composition of the couples, so that he may know with which metals and at which temperatures it is safe to employ them without fearing detrimental reactions. Mr. Howard briefly referred to the use of the Uehling pyrometer. Prof. Richards emphasized the importance of the time factor in investigating reactions, and agreed with the author of the paper that Seger cones must be handled very carefully to get good results.

On motion of Prof. Richards a resolution was then adopted expressing the thanks of the Society to all those who had made the meeting such a successful one, especially to Cornell University and the different departments visited, to the local committee and to the industries of Ithaca who had opened their shops to the visitors. This was the official conclusion of the ninth meeting of the American Electrochemical Society.

Thursday Afternoon.

A very interesting excursion was made Thursday afternoon to the works of the Remington Salt Co. The visitors studied with great interest the pumping machinery which lifts, from a depth of ½ mile, the brine, which is then evaporated in a large vacuum pan, this is followed by drying the crystallized salt by the centrifugal process and by final drying in a rotary drum. Two special and very enjoyable features of Thursday afternoon and evening were two lectures by Mr. E. G. Acheson and Dr. H. W. Wiley, which, however, were not directly connected with the meeting of the American Electrochemical Society.

Discovery and Invention.

Mr. E. G. Acheson, the distinguished electrochemical inventor, of Niagara Falls, addressed the engineering students of Cornell University in the Sibley Dome on "Discovery and Invention." In the introduction he spoke of Faraday as the representative of discovery without invention, of Bessemer as the representative of invention without discovery, and of Goodyear as both a discoverer and inventor. The main part of the address dealt with the author's own work on carborundum, artificial graphite and siloxicon. Mr. Acheson exhibited a great many of his products, and his lecture was full of interesting and suggestive remarks, and of reminiscences of his early work with Mr. Edison.

Foods.

In the evening Dr. H. W. Wiley, of Washington, D. C., spoke before the Cornell Section of the American Chemical Society on some special problems of the food question. His lecture was illustrated by lantern slides, and the author discussed the methods of testing foods and their adulterations.

The New President.

The new president of the American Electrochemical Society, Mr. Carl Hering was born 1860, in Philadelphia, Pa., and is one of the five surviving sons of the late Dr. Constantine Hering. He studied mechanical engineering at the University of Pennsylvania, graduating with high honors in 1880, with the degree of bachelor of science. Some years later he received the post-graduate degree of mechanical engineer. For a year after graduation he practiced as draughtsman and designer of machinery, and was then appointed instructor in mathematics and assistant in mechanical engineering at the University of Pennsylvania, and in 1882 assistant in physics, which position he took in order to begin studying electrical engineering. There being no college in this country at that time where electrical engineering was taught as a regular course, he went, in 1883, to Darmstadt, Germany, where he studied under Prof. Kittler, and was soon after made the latter's assistant.

Mr. Hering was assistant electrician at the International Electrical Exhibition in Philadelphia in 1884, and participated in the first extended comparative life test of the electric incandescent lamps at that time on the market. In 1885 he was chief electrical engineer for a large electrical company in Germany, and later for a company in Philadelphia. In 1886 to 1887 he taught electrical engineering at the University of Pennsylvania. In 1892 he was technical editor of *Electrical World*. Since 1886 he has been practicing as consulting electrical engineer in Philadelphia, being engaged chiefly with tests, reports, patent litigations and acting as consulting engineer for companies.

CARL HERING.

The high esteem in which Mr. Hering is held for his knowledge, experience and independence has often found recognition. He was a member of the Jury of Awards at the following expositions: Vienna, 1883; Philadelphia, 1884; Paris, 1889; St. Louis, 1890; Frankfurt, 1891; Philadelphia, 1899; Paris, 1900 (a member of the highest or superior jury); Buffalo, 1901 (chairman of Jury of Awards for the electrical exhibits and member of the Superior Jury); St. Louis, 1904 (chairman of group jury on electrochemistry and a member of the higher department jury for the Department of Electricity). In recognition of his services at the Paris Exposition in 1900 the French Government conferred on him the

decoration of the Cross of the Legion of Honor, popularly known as the "red ribbon."

Mr. Carl Hering was an active member of the following international electrical congresses: In Frankfurt, 1891, he was a delegate and vice-president. In 1893 he was an active member of the committee to prepare the preliminary programme for the Chicago International Electrical Congress. In 1900 he was a delegate and vice-president at the Paris Congress. In 1903 he was a member of the advisory committee of the International Electrical Congress of St. Louis of 1904, at which Congress he was the secretary of the Section on Electrochemistry.

Mr. Carl Hering is a member of numerous engineering and scientific societies. He is an honorary member of the New York Electrical Society, and was elected president of the American Institute of Electrical Engineers in 1900, president of the Engineers' Club of Philadelphia in 1904, and president of the American Electrochemical Society in 1906.

In spite of his extensive practice as consulting engineer, Mr. Hering has found time to present numerous valuable papers to the societies of which he is a member, and has been a frequent contributor to the technical press. In 1904 he published a book on "Ready Reference Tables," containing conversion factors of every unit or measure in use, enabling anyone to be readily reduced to any of the others and based on the accurate legal standard values of the United States. It is probably the first book of its kind published and based on the accurate legal standards. (An appreciative comment on this excellent book may be found on page 329 of our Vol. II.)

Mr. Hering's work in electrochemistry began with extended researches on accumulators, for which he obtained numerous patents. He also made extended researches concerning the commercial regeneration of spent battery solutions, including the recovery of the zinc, the reoxidization of the depolarizer and the separation of the acid and the depolarizer. Some years ago he made a complete recalculation of all electrochemical equivalents, based on the most recent, standard, fundamental values, and published the same in our Vol. I., p. 169.

In 1900 Mr. Hering made some very extended tests on a crude furnace for the recovery of arsenic from its ore, from which he deduced the figures for a properly constructed electric furnace, which calculation involved considerable labor and research to find out how much the losses could be reduced, as the losses in the crude furnace were very great. He has also designed other furnaces and has acted as consulting engineer in other industrial electrochemical work. He has devised and patented a method of getting zinc and similar metals out of their solutions when the products formed interfere with the process, as is the case when zinc is extracted from its sulphate. In this way he has obtained about 94 per cent of the zinc from the sulphate, with an ampere-hour efficiency of over 90 per cent. He was married in 1892, and has one young daughter.

In the following we give a complete list of the members and guests who registered at the Cornell meeting:

E. G. Acheson, Niagara Falls, N. Y.; Charles E. Acker, Niagara Falls, N. Y.; William C. Arsem, Schenectady, N. Y.; Edgar V. Ashcroft, London, England; Wilder D. Bancroft, Ithaca, N. Y.; John A. Black, Lima, Ohio; E. Blough, Pittsburg, Pa.; M. J. Brown, Tecumseh, Neb.; A. W. Browne, Ithaca, N. Y.; William Hand Browne, Jr., New York City; Hubert Buckley, Rochester, N. Y.; Charles F. Burgess, Madison, Wis.; H. G. Burnham, Ithaca, N. Y.; A. D. Camp, Ithaca, N. Y.; C. F. Carrier, Jr., Elmira, N. Y.; H. R. Carveth, Niagara Falls, N. Y.; G. W. Cavannaugh, Ithaca, N. Y.; H. B. Cohn, New York City; C. L. Collens, 2d, Niagara Falls, N. Y.; B. E. Curry, Bloomington, Ind.; M. Dennis, Ithaca, N. Y.; J. Forssell, Cleveland, Ohio; J. J. Frank, New York City; F. E. Gallagher, Ithaca, N. Y.; W. C. Geer, Ithaca, N. Y.; J. K. Gilis, Ithaca, N. Y.; C. P. Goepel, New York City; J. H. Goodwin, Pen Yan, N. Y.; F. X. Govers, Owego, N. Y.; Frank Hawkins, Hamden Junction, Ohio; L. F. Hawley, Ithaca, N. Y.; Carl Hering, Philadelphia, Pa.; Geo. A. Hulett, Princeton, N. J.; Henry Howard, Boston, Mass.; M. A. Hunter, Schenectady, N. Y.; F. H. Jennings, Ithaca, N. Y.; J. P. Magnusson, Ithaca, N. Y.; F. A. Mantel, Ithaca, N. Y.; F. C. Mathers, Ithaca, N. Y.; H. F. Matthew, New York City; W. E. McCourt, New York City; W. H. McLanahan, Bradford, Pa.; M. E. Mercer, Cleveland, Ohio; D. Lael Miller, Toronto, Ont.; H. [illegible]; H. H. Norris, Jersey, N. Y.; [illegible]; Patten, Washington, D. C.; J. L. Ramsay, Cleveland, Ohio; Rankin, Ithaca, N. Y.; J. W. Richards, Jr., Bethlehem, Pa.; George C. Robertson, Ithaca, N. Y.; H. [illegible]; Pa.; E. F. Roeber, New York City; S. S. [illegible]; Carl C. Schneiderhan, Ithaca, N. Y.; J. J. Sweeney, [illegible]; R. C. Spreeburg, Ithaca, N. Y.; Elmer A. Sperry, New York City; Henry P. Talbot, Boston, Mass.; J. E. Teeple, New York City; Edward C. Troy, Ithaca, N. Y.; Mechanical Engineer, New York City; Carlton Ward, Townsend, Washington, D. C.; J. W. Ward, Ithaca, N. Y.; W. H. Walker, Boston; G. Robert White, Ithaca, N. Y.; H. W. Wiley, Washington, D. C.; J. Lawson Wood, New York City; [illegible], Niagara Falls, N. Y.

Metallurgical Calculations.

By J. W. Richards, Ph. D.

Professor of Metallurgy in Lehigh University.

UTILIZATION OF FUEL IN THE BLAST FURNACE.

The blast furnace, in its simplest terms, may be regarded as a huge gas producer, producing in the region of the tuyeres pure producer gas from fixed carbon and heated air; the gas thus produced is partly oxidized in its ascent through the furnace by the oxygen abstracted from the charge (which latter item is almost a constant quantity per unit of pig iron made), and has added to it carbon dioxide from the carbonates of the charge. But, after all, the unoxidized and combustible ingredients of the gas escaping represent a large part, in fact, often the largest part, of the total calorific power of the fuel.

Problem 54.

A blast furnace uses 2,240 pounds of coke, containing 90 per cent fixed carbon and 350 pounds of limestone, containing 10 per cent of carbon (as carbonic acid, CO^2) to produce a ton of pig iron containing 4 per cent of carbon. The gases contain 24 per cent of carbonous oxide, CO, 12 per cent of carbonic oxide, CO^2, 2 per cent of hydrogen, 2 per cent of methane, and 60 per cent of nitrogen.

Required: (1) The volume of gas, as analyzed, produced per ton of pig iron made.

(2) The calorific power of the gas.

(3) The proportion of the calorific power of the coke which has been generated in the furnace.

Solution. (1) The carbon going into the gases will be that in the coke, less that in the pig iron, plus that in the carbonates of the charge.

Carbon in coke	2,240 × 0.90	2,016	pounds
Carbon in carbonates	350 × 0.10	35	"
Carbon charged		2,051	"
Carbon in pig iron	2,240 × 0.04	89.6	"
Carbon going into the gases		1,961.4	"

Carbon in 1 cubic foot of gas

In CO: 0.24 × 0.54

In CO^2: 0.12 × 0.54

In CH^4: 0.02 × 0.54

Total 0.38 × 0.54 = 0.2052 ounces Avoirdupois

= 0.012825 pounds

1,961.4

Gas produced per ton of pig iron = 153,035 cubic feet

0.012825

(2) Calorific power of 1 cubic foot of gas:

CO	0.24 ×	3,092	= 734.0 B. T. U.
H	0.02 ×	2,913	= 52.3 "
CH^4	0.02 ×	8,598	= 172.0 "
	Sum		= 958.3 "

[illegible]

per 153,035 cubic feet = 0.958 × 153,035 = [illegible]

(3) The calorific power of the coke, considering it to contain simply 90 per cent of fixed carbon, would be

$$8,100 \times 0.90 = 7,290 \text{ pound cal. per pound.}$$

The presence of CH^4 in the gases points, however, to there being probably some available hydrogen in it, which would increase its calorific power somewhat. A closer approximation to the calorific power of the coke could, therefore, be obtained by assuming at least as much available hydrogen in it as would correspond to the hydrogen in the CH^4 in the gas.

CH^4 in 152,935 cu. ft. of gas	=	3,059 cu. ft.
Weight of this = 3,059 × (0.09 × 8)	=	2,202 oz. av.
	=	137.7 lbs.
Hydrogen = 137.7 × 0.4 ÷ 100	=	34.4 lbs.
Available hydrogen in coke = 34.4 ÷ 2,240	=	1.54 per cent
Calorific power 0.0154 × 29,030	=	447 lb. cal. per lb.
Total calorific power of the coke	=	7,737 " "
Calorific power of coke used per ton 7,737 × 2,240	=	17,330,880 lb. cal.
Calorific power of the gases	=	9,168,450 "
Calorific power generated	=	8,162,430 "
	=	47.1 per cent

This figure, however, practically over-charges the furnace a little bit, because the pig iron with its 4 per cent of carbon really takes out of the furnace some unburnt fuel, whose heat of combustion may be utilized outside the furnace—as in the Bessemer converter. The furnace does not generate heat from this, representing:

$$2,240 \times 0.04 \times 8,100 = 725,760 \text{ lb. cal.}$$

leaving as actually generated in the furnace

$$8,162,430 - 725,760 = 7,436,670 \text{ lb. cal.}$$
$$= 42.9 \text{ per ceht} \quad (3)$$

Such an average blast furnace cannot, therefore, be accused of generating within it over some 43 per cent of the calorific power of the fuel put into it, while the heat rejected as potential energy of combustion of the waste gases amounts to more than half the calorific power of the fuel. Fifty years or more ago, when these waste gases were allowed to burn truly to waste, the blast furnace was indeed a devourer of fuel, but matters have been improved by the utilization of the waste gases to heat the blast, and thus one of the largest "leaks" of heat from the furnace has been patched up to some extent, although yet far from satisfactorily.

Problem 55.

Assume that in problem 54, one-third of the gases produced are burnt in hot-blast stoves, preheating the air blown in at the tuyeres, and that the blast is thus preheated to 450° C.

Required. (1) The amount of blast blown in per ton of pig iron made.

(2) The heat in the blast.

(3) The efficiency of the hot blast stoves.

(4) The increased efficiency of the blast furnace plant as a whole in generating the calorific power of the fuel, when thus provided with this hot-blast apparatus.

Solution.

(1) Volume of escape gases per ton = 152,935 cu. ft.
Nitrogen present in these (60%) = 91,761 "
91,761
Air containing this = 115,800 "
0.792 (1)

This is the volume of the blast per ton of pig iron produced, assuming no nitrogen to come from the coke used, and the blast to be dry. If the blast were moist, and its hygrometric condition known, the volume of moist blast could be calculated.

(2) Assuming the blast dry, it is heated to 450° C., requiring

$$115,860 \times [0.303 + 0.000027\ (450)] \times 450 = 16,430,975 \text{ oz. cal.}$$
$$= 1,026,936 \text{ lb. cal.}$$

(3) The hot-blast stoves receive one-third of all the gas produced, having, therefore, a calorific power of

$$9,168,450 \div 3 = 3,056,150 \text{ lb. cal.}$$

$$\text{Efficiency of the stoves} = \frac{1,026,936}{3,056,150} = 0.330 = 33.6 \text{ per cent} \quad (3)$$

(4) The blast furnace was primarily rejecting unused 57.1 per cent of the calorific power of the fuel, 4.2 per cent, however, as a necessary loss, to supply the carbon in the pig iron, but 52.9 per cent as combustible power of unburnt waste gases. If one-third of these gases are completely burnt in hot-blast stoves, then the combined plant—furnace plus stoves—is utilizing 17.6 per cent more of the calorific power of the fuel than before, or 42.9 + 17.6 = 60.5 per cent, and, therefore, rejecting undeveloped 39.5 per cent of the calorific power. (4)

[The net effect of the use of the hot-blast stove, upon the heat generation in the furnace, is practically to put back into the furnace, as sensible heat, 1-3 × 33.6 = 11.2 per cent of the calorific power of the waste gases, equal, therefore, to 0.112 × 52.9 = 5.9 per cent of the total calorific power of the fuel. This renders available, for the working of the furnace, 42.9 + 5.9 = 48.8 per cent of the calorific power of the fuel, an increase of available heat for reducing and smelting of $\frac{5.9}{42.9}$ = 13.7 per cent of the former available quantity.]

The practical conclusion is that a blast furnace generates in itself not much over 40 per cent of the calorific power of the fuel used, and rejects nearly 60 per cent; by using part of the waste gases to heat the blast, however, some of this rejected heat, to an amount representing net 5 to 10 per cent of the calorific power of the fuel used, is returned to and injected bodily into the furnace, thus rendering available for the purposes of running the furnace some 50 per cent of the calorific power of the fuel as a maximum. The efficiency with which the furnace applies this 50 per cent usefully to the objects of reducing and smelting, is another question for investigation.

Problem 56.

Assume that at the furnace of Problems 54 and 55 the two-thirds of the waste gases are burnt under boilers, raising steam which runs the blowing engines, hoists and pumps, and providing 10 effective horse-power for each ton of pig iron made per day in the furnace.

Required: (1) The efficiency of development of the calorific power of the fuel in the plant (furnace, stoves, boilers, engines) regarded as a whole.

(2) The thermo-mechanical efficiency of the boiler and engine plant.

(3) The power which could be generated if gas engines, at 25 per cent thermo-mechanical efficiency, were used in their stead.

Solution. (1) Since the stoves completely burn one-third of the waste gases, and the boilers the other two-thirds, all the combustible power of the waste gases is developed in the combined plant, and the only part of the calorific power of the fuel which is unused is the 4.9 per cent represented by the carbon necessarily entering into the composition of the pig iron. The plant as a whole, therefore, develops or generates 95.1 per cent of the calorific power of the fuel used.

(2) For each ton of pig iron, the heat developed under the boilers will be two-thirds of the calorific power of the gases, or

$$9,168,450 \times 2/3 = 6,112,300 \text{ lb. cal.}$$

There is generated thereby 10 effective horse-power days, equal to

10 × 33,000 × 60 × 24 = 475,200,000 ft. lbs

But 1 lb. cal. = 425 × 3.2808 = 1394.3 "

Therefore, thermal equivalent of

$$\text{work done} = \frac{475{,}200{,}000}{1394.3} = 340{,}800 \text{ lb. cal.}$$

Thermo-mechanical efficiency of boiler and engine plant:

$$\frac{340{,}800}{6{,}112{,}300} = 0.0557 \quad 5.57 \text{ per cent} \qquad (2)$$

(3) Gas engines, at 25 per cent thermomechanical efficiency, would give power representing per ton of pig iron produced:

6,112,300 × 0.25 = 1,528,075 lb cal.

Equal to 1,528,075 × 1394.3 = 2,130,645,900 ft. lbs.

or $$\frac{2{,}130{,}645{,}900}{33{,}000 \times 60 \times 24} = 44.0 \text{ horse power days} \qquad (3)$$

A quicker solution is:

$$10 \times \frac{25.}{5.57} = 44.9 \quad " \quad " \qquad (3)$$

Leaving net *surplus* power per ton of pig iron produced per day = 34.9 horse-power.

The preceding problems have elucidated the question of the small proportion of the calorific power of the fuel which is generated in a blast furnace, showing it to be, in usual practice, only 40 to, at most, 50 per cent of the calorific power. The discussion has not explained "why," but a further consideration will throw light on this question also.

The proportion of the calorific power of a fuel which is generated in a blast furnace is solely a question of how much of it is burned to carbonic oxide, CO^2, and how much to carbonous oxide, CO? If all the carbon were burned to CO^2, practically all the calorific power of the fuel would be generated; if all

were burned to CO, only $\frac{2{,}430}{8{,}100} = 0.30 = 30$ per cent of the

heating power of the carbon would be generated. If a blast furnace was filled with nothing but coke, and air blown in as usual at the tuyeres, carbon would be burned in the furnace only to CO, and but 30 per cent of its calorific power be generated and available for the needs of the furnace. The entire gain over this percentage is due to the oxidation of CO to CO^2 by the oxygen abstracted from the solid charges, that is, by the act of reduction. In Problem 54 we calculated that under ordinary conditions, between 40 and 50 per cent of the calorific power of the fuel is generated in the furnace; the excess of this above 30 per cent is due to the oxidation of CO to CO^2 during the reduction of the metallic oxides in the charge. From this standpoint it is advisable to strive to perform the greatest possible proportion of the reduction in the furnace by CO gas, because in this case the total generation of heat in the furnace per unit of fuel charged will tend towards a maximum. Since no carbon can be burned to CO at the tuyeres, it follows that, *from the standpoint of the generation of the maximum quantity of heat in the furnace from a given weight of fuel*, Grüner was right in formulating his dictum of the ideal working of a blast furnace, viz:

GRÜNER'S "IDEAL WORKING."

All the carbon burnt in the furnace should be first oxidized at the tuyeres to CO, and all reduction of oxides above the tuyeres should be caused by CO, which thus becomes CO^2. This dictum is not in Grüner's own words, but expresses their sense, and from the point of view of the present discussion, it is the correct principle upon which to obtain the maximum generation of heat in the furnace from a given weight of fuel. It practically directs us to generate at the tuyeres 30 per cent of the calorific power of the carbon oxidized in the furnace, and the rest that can be obtained from the combustion of CO generated during the reduction of the charge.

If we apply this principle to the furnace and case of Problem 54, we should first observe that the carbon required by the furnace is::

Carbon in coke charged	2010 pounds
Carbon in pig iron produced	84.0 "
Carbon oxidized in furnace	1926.4 "

Requiring, if all oxidized to CO at the tuyeres

$$1926.6 \times \frac{4}{3} \text{ lbs. oxygen} = 2569. \text{ pounds}$$

But, Problem 52 showed us that there was actually blown into this furnace 115,860 cubic feet of blast, containing, therefore,

115,860 × 0.208 × 0.44 ÷ 100 = 2.109 lbs. oxygen

capable of oxidizing to CO at the tuyeres

2.109 × 0.75 = 1.627 lbs. carbon

Proportion of carbon gasified burnt at the tuyeres,

$$\frac{1{,}627}{1926.6} = 0.844 = 84.4 \text{ per cent}$$

It is, therefore, true of the furnace under discussion, that if Grüner's ideal working be called standard, this furnace attains to 84.4 per cent of that ideal; and it is also true that this furnace generates from the carbon burnt at the tuyeres 84.4 per cent of the amount of heat which could have been generated if Grüner's ideal working had been attained.

It is always possible to find out for any given blast furnace, by similar calculations, how much carbon is burned at the tuyeres, and how much is burned above the tuyeres, and thus to determine how closely the furnace running approximates to Grüner's ideal working. This proportion or percentage will not necessarily express how efficiently the furnace is running, as regards fuel used per unit of iron made, but it will tell what proportion of the calorific power of the fuel used is being generated at the tuyeres, and in possibly nine cases out of ten this proportion indicates the general efficiency of the furnace as regards fuel consumption.

It will be next profitable to inquire when and under what conditions Grüner's ideal working does not correspond to maximum fuel economy, and why it usually does. The answer is not difficult to understand: if all the carbon gasified in the furnace is burned to CO at the tuyeres, 30 per cent of the total calorific power of the carbon burned is there developed, which is more than half of all the heat generated from carbon in the furnace. To this must also be added the sensible heat in hot blast, which may amount (as in Problem 52) to some 5 to per cent of the calorific power of the carbon, making, therefore, a total of 35 per cent of the calorific energy of the fuel generated at the tuyeres out of a total of about 50 per cent developed in the furnace. If, however, the blast be heated to a very high temperature, or particularly if it be dried, or if the ore and fuel are extra pure, so that a smaller quantity of heat is needed to melt down slag at the tuyeres, then there may not be needed at the tuyeres the generation by combustion of so much heat as Grüner's ideal working would require and gas to be produced, and to burn at the tuyeres all the carbon oxidized in the furnace would be wasteful of fuel. In this case, although less heat would be generated per unit of fuel by burning some of it above the tuyeres, yet economy in fuel consumption as a whole would be attained, because of the better distribution of the heat which was generated from a smaller total quantity of fuel.

Illustration. In Problem 55 we assumed that the furnace ran with blast heated to 450° C., and that there was developing at the tuyeres 84.4 per cent of all the carbon gasified in the furnace, smelted down pig iron and slag satisfactorily, and kept

the tuyere region at proper temperature. If the temperature of this blast were raised to 900° C., how much greater proportion of heat would be available in the tuyere region?

We have already calculated that the 115,860 cubic feet of blast used per ton of iron made, brought in at 450° C., 1,026,936 pound calories of heat, equal to 5.9 per cent of the calorific power of the fuel put into the furnace. If the temperature were 900° C., the heat brought in would be

$$115{,}800 \times [0.303 - 0.000027 (900)] \cdot 900 = 37{,}020{,}978 \text{ oz. cal.}$$
$$= 2{,}370{,}061 \text{ lb. cal.}$$

which equals

$$\frac{2{,}370{,}001}{17{,}330{,}880} = 0.137 = 13.7 \text{ per cent}$$

of the calorific power of the fuel, a gain of 7.8 per cent added to the heat available in the tuyere region. This causes a very great increase in the smelting-down power of the furnace, enabling the same work per ton of ore smelted to be done with much less consumption of fuel in the tuyere region. An idea of this increased smelting power may be obtained from the following comparison of heat available for smelting purposes in the tuyere region in the two cases just discussed::

Case 1.

Heat developed by oxidation of carbon per ton of iron made, 1,027 lbs. × 2,430	= 3,953,610 lb. cal.
Sensible heat in blast at 450° C.	= 1,026,936 "
Total heat available	= 4,980,546 "

Case 2.

Heat developed by same quantity of air burning same carbon	= 3,953,610 "
Sensible heat in blast at 900° C.	= 2,370,061 "
Total heat available	= 6,323,671 "

It is, therefore, seen that the heat generated and available at the tuyeres is increased 1,343,125 calories, amounting to 27 per cent of the amount disposable in Case 1. It follows, therefore, that the smelting down power has been increased 27 per cent, and that, if the 4,980,546 calories were sufficient for satisfactorily smelting down the iron and slag in the first instance, that the extra heat of Case 2 can all be utilized for smelting down 27 per cent extra burden. We can, therefore, charge 27 per cent more burden per unit weight of coke in Case 2, because we have the requisite smelting down power at the region of the tuyeres, which amounts to saying that we can charge 22 per cent less coke for a given weight of pig iron made.

In actual practice, as the amount of burden is increased and the temperature of the blast increased, the change causes more and more of the carbon to be oxidized above the tuyeres, and a smaller proportion to be oxidized at the tuyeres, thus obtaining less service in the furnace from oxidation of carbon as a whole, but compensating for this by the extra heat in the hot blast. Or, looking at it in another way, we may say that the same heat could be made available in the region of the tuyeres, when using hot blast, by the combustion there of a smaller quantity of carbon; therefore, we can burn more of it above the tuyeres and yet work more economically on the whole, than we were working in the first instance, with the colder blast.

Minimum Carbon Necessary in the Furnace.

Many writers have assumed that in the reduction of iron oxide, such as Fe_2O_3, the reaction of its reduction by carbonous oxide, CO, is expressed as follows:

$$Fe_2O_3 + 3CO = 2Fe + 3CO_2$$

If this were true, there would need to be burnt to CO at the tuyeres only 3C, or 36 parts of carbon, to ensure the reduction of Fe_2O_3, representing 112 parts of iron. The reaction does not, however, progress as shown, because CO_2 acts oxidizingly on Fe to such a degree that when $1CO_2$ is present in the gas for 1CO left unused, the reduction practically stops, even though the gases are moving slowly through the warm ore. The real reaction of reduction by CO gas is therefore more nearly represented by

$$Fe_2O_3 + 6CO = 2Fe + 3CO + 3CO_2,$$

which shows that 112 parts of iron would require at least 72 parts of carbon to be oxidized at the tuyeres to CO, in order to produce the gas necessary for its reduction. The presence of some CO_2 in the furnace coming directly from carbonates in the charge would neutralize still more of the reducing power of the CO gas, and cause still more of it to be theoretically required for reduction. The minimum amount of carbon necessary to be charged in the furnace will be that necessary to furnish fixed carbon enough for this reducing gas and for the carbon in the pig iron. This would be, per 100 parts of pig iron, containing, say 93 iron and 3 carbon, and using coke containing 90 per cent of fixed carbon:

Carbon burnt at tuyeres = 72 × 93 ÷ 112 =	59.8
Carbon in pig iron =	3.0
Total fixed carbon necessary =	62.8
Total coke to supply this = 62.8 ÷ 0.9 =	69.8

It results from these calculations that if "Grüner's ideal working" of a blast furnace were carried out to the practical extent of reducing all the charge by carbonous oxide, CO, and oxidizing no carbon at all directly above the tuyeres, that about 63 parts of fixed carbon would be required per 100 of pig iron made, requiring from 70 to 80 parts of fuel, according to its richness in fixed carbon (90 to 80 per cent). In practice, as is well known, *more* than this is commonly used, because of the larger proportion of unused CO in the gases than above assumed; and *less* than this has been regularly used, showing that economy of fuel can be attained without adhering to "Grüner's ideal working," in fact, by transgressing it as far as one dares.

The principle involved can be best grasped by a calculation of the amount of carbon which would be required by the furnace, supposing all the heat necessary for melting down the charge were supplied by electrical means, thus dispensing, for the purposes of this supposition, with the necessity of blast and the consequent necessity of oxidizing any carbon by any other agent than the oxygen given up by the ore. In this case the gases resulting would be, let us assume, of the same composition as before, that is, containing equal volumes of CO and CO_2, and since this oxygen is abstracted altogether from the ore, the reaction is

$$Fe_2O_3 + 2C = 2Fe + CO + CO_2$$

This would represent the utilization of carbon in an electrically-heated furnace, and would require per 100 of pig iron made, assuming it 3 per cent carbon and 93 iron:

Carbon for reduction 24 × 93 ÷ 112 =	19.9
Carbon in pig iron =	3.0
Total fixed carbon necessary =	22.9

Or only a little over one third as much as the minimum required when the smelting down is done by blast.

Aside from electrical furnace practice, however, this discussion proves that whatever fixed carbon burns or oxidizes above the region of the tuyeres, in a blast furnace, absorbs oxygen from the charges with three times the efficiency of carbon first burnt at the tuyeres. Every pound of oxygen abstracted from the charges by solid carbon requires the use or intervention of only one third as much carbon as that which is abstracted by CO gas; or, each pound of carbon abstracting oxygen directly from the charge takes from it three times as much oxygen as a pound of carbon first burnt to CO at the tuyeres possibly can.

The ordinary furnace produces at the tuyeres, in order to get

heat enough to melt down the charges, *more* CO gas than is needed to abstract all the oxygen from the charges; under these conditions it is uneconomical to oxidize any carbon at all above the tuyeres. The exceptional furnace, because of pure ores, small amount of slag, pure fuel, high temperature of blast, or dry blast, gets heat enough at the tuyeres to melt down the charges *without* producing enough CO gas to reduce all the charges; under these conditions more or less reduction is effected by solid carbon, and with the greatest economy in quantity of carbon required in the furnace. These are the conditions under which, having passed the turning point, the greater economy of fuel is attained the farther away one can get from "Grüner's ideal working."

Notes on Electrochemistry and Metallurgy in Great Britain

(From Our London Correspondent.)

Electrolytic Rectifiers.

A most interesting paper on "The Construction and Use of Oscillation Valves for Rectifying High-Frequency Currents," was recently read before the Physical Society by Prof. Fleming, in which the author recalled the fact that as far back as 1890, when investigating the Edison effect in glow-lamps, he had shown that the space between the incandescent carbon filament and an insulated metal plate placed in the vacuous bulb possessed a unilateral conductivity. Electrolytic rectifiers, such as the aluminium-carbon cell, were not available for high frequency currents, because a time element entered into the chemical actions involved. In 1904, however, the author discovered that if the carbon filament in an electric glow-lamp was surrounded with a metal cylinder connected to an insulated terminal by a wire sealed through the bulb, and if the filament was made incandescent by an insulated battery, then between the insulated terminal and the negative pole of the battery a unilateral conductivity existed which was operative with currents of any frequency, and the valve so made might be employed to render electrical oscillations measurable by an ordinary sensitive galvanometer. The author exhibited oscillation valves made on this plan, and also showed their uses. The advantage of using multiple spark-gaps in oscillation circuits was proved by the use of the valve. The author exhibited a multiple spark-gap discharger made with carbon rods, and claimed for it a much greater constancy of action than a discharger made with metal balls. Dr. Fleming also showed an experiment showing the copious emission of negative ions from the glower of a Nernst lamp at atmospheric pressure. The author referred to the employment of carbon-filament oscillation valves, such as he had described as wave detectors or cymoscopes in connection with wireless telegraphy. The amount of rectification produced by the valve could be measured, and might amount to 89 to 90 per cent.

Meeting of the Faraday Society.

The Electric Furnace for Iron and Steel.

The chairman, Prof. Huntingdon, opened the proceedings at the April meeting by remarking that the Society was endeavoring to keep pace with the electric furnace developments. People were, however, working hard at the practical side, and had no time to read papers on the subject. Mr. Harbord had kindly consented to read the three papers relating to this subject before them that evening. (They were abstracted on page 107 of our May issue.)

Mr. Harbord then read Mr. Keller's paper, practically verbatim.

Mr. Morrison said this paper seemed to him too academical. Nothing was said about the furnace itself. There were many questions he should like to ask, amongst which was, whether difficulties due to the tap-holes "freezing up" had been got over. Also what variation [illegible] the quantity of silicon in [illegible] last paragraph it appeared that [illegible] main in Mr. Keller's metal [illegible] form of furnace.

The chairman remarked that makers would naturally be reticent about the details of their [illegible] perimental stage.

Mr. Harbord said that when he was present at the working of this furnace it "froze up." The current [illegible] whether the metal flows or not. He considered that Mr. Keller had exaggerated the possibility of [illegible] electrical furnaces. Where anything like a large output was required this was impracticable. With regard to remelting of metal for castings, the metal would not, in the ordinary way, be taken direct from the blast furnace. The ordinary procedure is to obtain pigs of different percentage of silicon and mix them in the cupola.

Prof. Huntingdon said that in 1882 he had carried out experiments with the Siemens furnace, and that in that year a paper was read before the British Association, which seemed to be an effectual way of burying it. Experiments were made to find the maximum amount of carbon the pig would take up. Iron containing from ¼ to 9 per cent of silica was made, and its properties noted. That containing 2 per cent machined beautifully, producing an even surface. He also experimented on the elimination of sulphur, and thought that in these particulars he had anticipated Keller's experiments. In the course of these experiments they volatilized copper, and obtained also a mass of tungsten in crystals. This was the only solid tungsten which had been produced so far as he knew.

The paper on the Stassano furnace was next read.

Mr. Harbord said that the idea of rotating the Stassano furnace was to thoroughly mix the iron oxide with the metal. The inventor believes, and the speaker thought he was correct, that he can make steel direct from the ore in this furnace. The hearth was about 2 feet up from the base of the furnace.

Mr. Morrison asked if any figures relating to efficiency were available. Mr. Harbord could not at the time give figures, but believed the furnace to be not quite so efficient as Keller's or Héroult's. The electrodes did not touch the bath, and this he thought had disadvantages. Prof. Huntingdon surmised that the electrodes consumed more rapidly when touching the bath.

Mr. Harbord, in answer to Mr. Spiers, said he believed the rotation was at the rate of one or two revolutions per minute. He thought the rotation the weak point of the furnace.

In answer to Dr. Borns, he said the connections were made through an arrangement something like a trolley pole.

Mr. Harbord then read extracts from Mr. Gin's paper. He said the furnace was not commercially working anywhere when he was on the Continent.

Mr. Morrison had not heard of its being set up for that purpose, and thought it was in the experimental stage.

Mr. Weiss introduced the subject of power, and thought where water power was not available gas power would be the cheapest. This was supported in the report of the Canadian Commission. He thought the electric furnace might be useful for smelting copper ore.

Mr. Harbord said good steel can be made in the electric furnace. Prof. Huntingdon said that it might be used for high class steel, but so far as could be seen at present it would not come in much for commercial purposes.

Mr. H. S. Coleman's paper on "Cleaning Articles by the Electric Current" was read by the Secretary. The work to be cleaned (usually preparatory to electroplating) is [illegible] a hot solution of equal quantities of [illegible] and sodium hydrate in a wrought iron [illegible] used as anode for 5 or 10 minutes, the [illegible] current is then reversed for a short time [illegible]

repeated. The grease becomes saponified and goes into solution. The lighter dirt is skimmed off from the surface of the bath, while the remainder falls to the bottom.

Dr. Borns thought the surface of the tank getting covered with oxide would interfere with the conductivity. Mr. F. S. Spiers said all the better if it did. It would prevent the tank getting eaten away.

Metallurgical Papers at the Institution of Mining and Metallurgy.

Assay of Auriferous Tin-Stone.

Chief among the metallurgical papers read at the meeting held on April 19, was that by Mr. C. O. Bannister "On the Assay of Auriferous Tin-Stone." Finding various discrepancies in the assay results of different chemists, the author compared the results obtained by the five different methods of:

1. Scorification assay.
2. Crucible assay.
3. Wet method.
4. Concentration of the gold in part of the tin.
5. Collection of the gold in the whole of the tin.

Omitting the laboratory details of the steps taken in each process the following are the author's conclusions:

The scorification method invariably gives low results for these ores, and this is due to the unsatisfactory nature of the slag obtainable. In rich ores the results may be improved by taking smaller quantities of ore, such as 2.5 to 3 gram; but in this case, any error occurring in the assay is largely magnified in calculating the results, and thus it is not advisable to use this method.

The crucible assay gives excellent results, both for silver and gold present, and as 25 grams can easily be worked off it is the most suitable method for most ores.

The wet method gives good results for the gold, but silver is not determined; this method, however, is far more tedious than the crucible assay, and the results obtained compare about equally with those obtained by that method.

The method by concentration in part of the tin gives low results, partly owing to gold being left in the slag, and also owing to loss on the treatment of the tin buttons; moreover, the silver results are unreliable.

The method of collection in the tin from the cyanide assay is suitable for detecting the presence of gold, but gives slightly low results as a method of assay, and is unreliable for the estimation of the silver contents.

The other papers read at this meeting were: "The Bucket Dredging Industry," by E. S. and G. N. Marks; "The Production of High-Grade Bullion from Zinc-Box Precipitates," by C. N. Morris; "On the Use of the Impact Screen in Tin Dressing," by J. H. Collins; "The Mitchell's Creek Gold Mines, New South Wales," by W. F. Macdonald, and "Notes on a Prospecting Shaft in the Goldfield District, Goldfield, Nev.," by E. A. Collins. The first three of these I have laid aside for subsequent abstracting when the rush of metallurgical society meetings has quieted.

These are to follow closely on each other this summer: The "James Forrest" lecture at the Institution of Civil Engineers, by Mr. R. A. Hadfield, is to be delivered on May 2. A week later the Iron and Steel Institute holds its May meeting, to be followed in July by its joint London meeting with the Institution of Mining Engineers. Meanwhile other societies continue their regular meetings.

The Resistance of Iron and Steel to Reversals of Direct Stress.

The Institution of Civil Engineers, *facile princeps* among the world's engineering institutions, can hardly be accused of pandering to the press, technical or popular. In a paternal way it issues an official resume of each paper read to its selected press list, but absolutely bars the reporting of its meetings. From the official abstract of a paper on the above subject, by Dr. T. A. Stanton and Mr. L. Bairston, I have noted the following points for transcription. The need for further research upon this subject is urged for these cogent reasons:

"(1) Practically all the previous work, with the exception of Reynolds and Smith's experiments, has been done by subjecting the materials to transverse stresses, the intensity of which has, therefore, to be calculated by the ordinary theory of bending. (2) The resistance of the materials in common use by the engineers at the present day when subject to reversals of stress is imperfectly known, and there exists considerable difference of opinion as to the materials best suited for stresses of this kind. (3) Although it appears from Reynolds and Smith's experiments that the resistance of iron and steel is seriously diminished when the alterations are very rapid (*i. e.*, 1,500 to 2,000 per minute), it is not known if this reduction in resistance is considerable at those speeds which are common in high-speed reciprocating motors (*i. e.*, in the neighborhood of 800 reversals per minute), since the majority of experiments have been made at approximately 60 reversals per minute. (4) Although it is generally recognized that the effect of moderately rapid or sudden changes in section of materials subject to reversals of stress is to diminish their resistance, the amount of this reduction in strength for the various materials commonly used is not known. (5) The common assumption that, in cases in which the stress varies from tension to compression, but between unequal limits, the resistance depends solely on the range of stress and not on the actual values of these limits, has not been experimentally verified."

The effect of reversals of direct stress was, therefore, investigated on a number of bars of iron and steel, the alternating stress testing machine at the National Physical Laboratory being used. A microscopical investigation was carried out concurrently to observe the changes taking place in the crystalline structure of the materials, and "to determine if possible the manner in which ultimate failure occurs."

The materials subjected to the tests may be considered in three groups:

(1) Three samples of Swedish Bessemer steel and one sample of Swedish charcoal iron, presented by Mr. R. A. Hadfield for the purpose. The carbon content of the steels was approximately 0.17, 0.44 and 0.64 per cent.

(2) Four samples of steel presented by Messrs. Belliss and Morcom for the purpose. Of these, two were mild-steel bars, one was a bar of harder steel used for piston rods, and the fourth consisted of specimens which had been cut from a large steel forging.

(3) Two samples of wrought iron of British manufacture, bought for the purpose of the tests.

Although more uniformity in the results of the tests would no doubt have been obtained by subjecting all the specimens cut from any given material to an annealing process, it was felt that this would detract from the value of the tests, owing to the well-known effect of heat treatment on the resistance of steel. For this reason the tests were made on the bars as received; and in cases in which there were several bars of the same material, the specimens in any group of tests were not always cut from the same bar. This does not apply to the case of the specimens whose structure was examined microscopically, in which the actual resistance was of secondary importance.

The results of the experiments may be stated briefly as follows:

1. The superiority, in resistance to reversals of stress, of moderately high-carbon steels over low-carbon steels and wrought irons, which was discovered by Wohler to exist when the rate of reversals was 60 per minute, still holds when this rate is increased to 800 per minute, although, according to Reynolds' and Smith's experiments, this superiority no longer exists when the rate of reversals is in the neighborhood of 2,000 per minute. 2. As far as comparisons can be made be-

tween the results of the present experiments and those of Wohler and Sir Benjamin Baker, there is no marked reduction in resistance due to raising the rate of reversals to 800 per minute. 3. Experiments in which the ratio of tension to compression varied from 1.14 to 0.73 indicated that between these limits the value of the maximum range of stress was practically independent of the actual values of the limiting stresses in tension and compression. 4. The resistance of the materials in three typical cases of rapid reduction of area of the specimens has been determined. 5. The failure of iron specimens, due to the development of the slip-lines of Ewing and Rosenhain into cracks, has been determined for the case of direct stress; and the failure of moderately high-carbon steel, due to the development of cracks in the ferritic areas of the structure, has also been established.

MARKET QUOTATIONS DURING APRIL.

The continued activity in the textile industries has maintained the price of the alkali and coal-tar products.

As regards metals, the price of copper was very firm throughout, rising from £83.15 per ton to £86 per ton. Copper sulphate has further advanced to £26 per ton. Tin, which stood at £169 per ton on March 30, rose by April 30 to £183 per ton. As a result renewed interest is being taken in various Cornish mining properties, and the reworking of certain abandoned mines seems quite feasible.

There has been no particular change in regard to iron, Cleveland warrants closing at £2.9.6. Lead has been in fair demand, prices being practically stationary. Zinc sheets are £2 dearer, at £30.5 per ton.

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

ELECTRIC FURNACES.

Manufacture of Calcium Carbide.—H. L. Hartenstein, 819,218, 819,219, 819,220, 819,221, 819,222, 819,223 and 819,224, May 1. Applications filed July 23 and 26, 1902.

The Hartenstein process of making calcium carbide, or as he calls it, carbolite, is commercially operated in Constantine, Mich. This is the only case of competition with the Union Carbide Co., which, owning to the Willson patents, enjoys otherwise a complete monopoly in the manufacture of calcium carbide in this country.

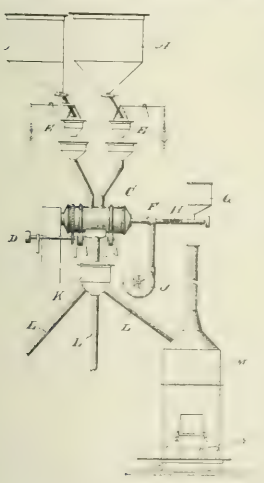


FIG. 1.—APPARATUS FOR MAKING CALCIUM CARBIDE.

The first five patents of Hartenstein, mentioned above, refer to the process of manufacture, the last two to the construction of his electric furnace.

The whole scheme is indicated diagrammatically in Fig. 1. *A* is a hopper for limestone, crushed to about 20-mesh, *B* a hopper for coke, pulverized to about 50-mesh. These materials are passed to the automatic dumping scales *E* and then introduced into the "preheating furnace" *C*. This is heated by means of the burner *F*, which delivers into the furnace coal dust (from bin *G*) and an air-blast, by means of the blower *J*. The furnace *C* is first heated to 1,500° to 2,000° F., and then the charge of limestone is introduced from *A*, and furnace *C* is slowly rotated. Carbonic acid gas and other gases are driven off, and the charge is heated up to incandescence. The charge of pulverized coke (1 part by weight for every 3 parts of limestone) is then introduced from *B* into furnace *C*, which is now rapidly rotated. During this "preheating operation" a small portion of carbide dust or fines may be added in order to thoroughly expel and drive off any remaining carbon monoxide and carbon dioxide.

From the preheating furnace *C* the charge is passed to the distributing tank *K*, but in order to maintain the high temperature, a "superheating compound" is first added, which also serves for removing any phosphorus that may be present. This superheating compound consists of 60 per cent calcium carbide, 20 per cent black oxide of manganese, 15 per cent bituminous coal, 3 per cent aluminium and 2 per cent chlorate of potash, all elements in pulverized or granular condition; 15 to 20 pounds of superheating flux are added per ton of material treated. The object is to maintain the charge at a sufficiently high temperature, so as to avoid the necessity of an expenditure of more electrical energy later on in the process.

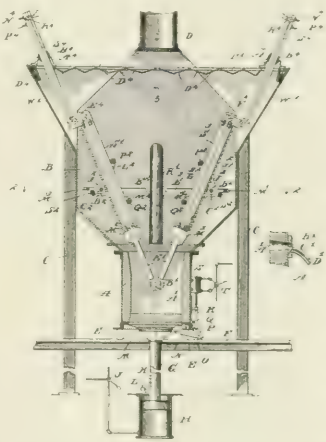


FIG. 2.—FURNACE FOR MAKING CALCIUM CARBIDE.

From the distributing tank *K* the charge is delivered through one of three pipes *I* to the electric furnace *H*, where the charge is heated to the melting point to effect the conversion into carbide, and finally tapped and run off into molds or over plates, so as to spread out into a comparatively thin sheet, so as to cool within a very few minutes' time. The object is to produce a carbide of a comparatively hard stone or flint-like structure, which is not objectionable, porous, and which, during the process of breaking or crushing, yields a small percentage only of fines.

The molds are covered with coal-tar, gas-tar or oil, mixed with coal-dust or coke. The object of the coal-tar or glucose is to apply to the carbide a thin coating of oil, protecting coating, so as to protect it against moisture. The finely-divided coke is used to prevent the admixture of particles of metallic calcium to the mass of carbide. In the reaction with water any particles of calcium would yield hydrogen gas which would detract from the illuminating quality of the acetylene gas.

The general construction of the electric furnace is shown in Fig. 2. One of the main objects is to permit adjustment of the voltage according to the amount of charge delivered to the furnace chamber or to the condition of the carbon electrodes. The carbons can be adjusted longitudinally, and the slant or inclinations of the carbons with reference to each other may be varied. *A* is the furnace chamber into which the charge is introduced so as to drop between the two electrodes. A^1 are the discharge openings, protected by slabs B^1, the construction being clearly shown in the small diagram at the right-hand of Fig. 2. For further details of the furnace construction see patent 819,224. The lining for the electric furnace (patent 816,223) is made by "mixing together cream of lime and pulverized coke and then adding thereto a mixture of asbestos and a hydrocarbon adherent, then forming the same into bricks, blocks or slabs, and then pressing and drying the same."

Tantalum.—W. von Bolton, 817,733, April 10. Application filed Jan. 5, 1904. (Assigned to Siemens & Halske Co.)

An impure amorphous powder containing oxides or carbides of a highly refractory material, such as tantalum, is reduced to the pure metal by the heat of the electric arc in a vacuum furnace. "Prior to my invention, so far as I am aware, there did not exist a relatively pure metal of a highly refractory type, such as tantalum, which was ductile or capable of being rolled or drawn, and such a product being new with me constitutes the essence of my improvement." The two claims are exceedingly broad; the first one reads: "A metal derived from tantalum compounds and possessing the quality of being homogeneous and ductile;" the second: "substantially pure metallic tantalum possessing the qualifications of homogeneousness and ductility."

Reducing Calcium Oxide.—T. L. Willson, 820,031, May 8. Application filed Jan. 28, 1896. (Assigned to Union Carbide Co.)

It took more than ten years to get this patent granted. It refers to the reduction of calcium oxide in an electric arc furnace, a carbon crucible serving as one electrode and a carbon pencil as the other one. The charge is calcium oxide and coke. "The reduced metal may alloy or combine with some other metal or element if present and may combine with an excess of carbon to form a carbide."

Electric Furnace for Iron Reduction.—M. Ruthenburg, 818,918, April 24. Application filed Nov. 28, 1903.

Mr. Ruthenburg's process for treating iron concentrated from magnetic separators has been repeatedly described in our columns (see the index of our former volumes under Furnace, Electric, Iron and Steel, Ruthenburg). The present patent relates to the combination of the different stages of the process. First, the comminuted particles of ore are fritted together in coherent porous lumps by passing the concentrates between two revolving rolls which serve at the same time as electrodes. The lumps drop into a reservoir below, with a reducing heated atmosphere. Then before the lumps have fallen below the temperature of reduction of iron oxide, they are subjected to the action of a reducing gas from a gas producer, and are then reduced without being fused. By applying more electrical heat the reduced metal is melted and drawn off. Under the conditions of the experiments of the inventor the fritting operation required 100 volts, and the smelting of the reduced iron about 15 volts. The two sets of electrodes used for both operations are connected in series.

Fusing Quartz.—C. O. Wingren, 817,212, April 10. Application filed Aug. 21, 1905.

In making articles of fused quartz in an electric furnace, the difficulty has often been experienced that after cooling the quartz contains air bubbles. These are due to lack of fluidity in the quartz, this being not sufficient to expel the air. The inventor connects his electric furnace with an air pump, and, while melting the quartz, he puts a pressure on the quartz; after the quartz is melted he exhausts the air from the furnace; finally he places again a pressure upon the mass while it is cooling. The claims refer to the construction of the furnace and pump.

Electric Muffle.—E. Zell, 819,625, May 1. Application filed Oct. 28, 1902. (Assigned to General Electric Co.)

Iron is a very suitable material for walls of muffles, heated by the passage of an electric current through the walls, but the iron must be protected, as it would quickly corrode and deteriorate otherwise. The inventor packs the iron in granular calcium carbide. The action of the heated chamber on the packing is to produce carbon monoxide and calcium oxide,

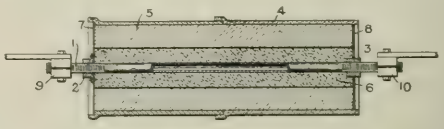


FIG. 3.—ELECTRIC MUFFLE.

both serving to protect the tube. "With such a packing an iron-chamber furnace can be run at a temperature of about 1,000° C., and the life of the chamber is of considerable duration." In Fig. 3 the iron pipe 1, threaded at both ends for making connections, forms the heating chamber. Around it there is the packing 6 of granular calcium carbide, and around this a lining 5 or lime or asbestos.

Dental Furnace.—J. F. Hammond, 817,767, April 17. Application filed April 13, 1905.

The interior of the muffle is wound with a heating platinum wire. A resistance wire of nickel is wound around the casing for the muffle. Both wires are electrically connected. The convolutions of the resistance wire of nickel are so bent, at portions parallel to each other, that the bend of one wire will lie within the bend of another wire. A sliding contact is provided along these bends for varying the resistance. In this way the amount of heat generated can be easily controlled. The furnace is specially adapted for the manufacture of jewelry and dental articles.

Dental Furnace.—D. G. Steinecke, 820,025, May 8. Application filed May 23, 1905.

Details of construction of a dental furnace, intended to be an improvement on the "Hammond furnace." The muffle is composed of two parts, and is readily detached from the furnace proper. Each of the two parts of the muffle may be electrically heated independent of the other part.

ELECTROLYTIC PROCESSES.

Smelting Lead Ores.—W. Valentine and A. G. Betts, 816,704, April 13. Application filed June 15, 1904.

The process described in this patent was fully and authoritatively covered in the article of Mr. A. G. Betts in our May issue, page 169. Claim 1 relates to "the process of recovering lead from lead-sulphide ores, which consists in fusing the ore and accompanying gangue into a matte and a slag, and electrolytically separating lead from the matte by bringing the matte into contact with a cathode in a fused electrolyte in which lead sulphide is substantially insoluble."

Electric Furnace.—A. G. Betts, 816,554, April 3. Application filed May 20, 1904.

Details of construction of the furnace which was described and illustrated (Fig. 2, p. 171) in Mr. Betts' article in our May issue.

Electrolytic Production of Caustic and Chlorine.—F. McDonald, 814,864, March 13. Application filed April 27, 1905.

The inventor, who has been quite active in this field in recent years (our Vol. I., p. 387, and Vol. III., p. 110), patents

the arrangement of plant shown in Fig. 4. The solution of sodium chloride is fed through pipe 20 to the anode compartment of the electrolytic cell 1. This supply is controlled by the valve 23, which is in turn controlled by the float 23', mounted in a tank 21. This is in communication with the bottom of the anode compartment of cell through pipe 22, so that the level of the brine solution is the same in the anode compartment of 1 and 21. The overflow pipe 30 leads to the gas collecting pipe 31, into which the overflow of the anode compartment of

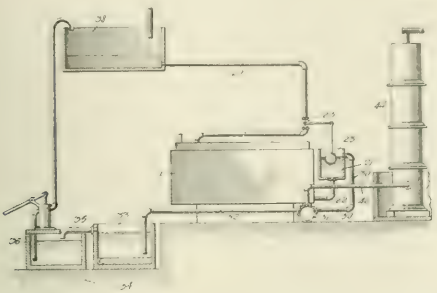


FIG. 4.—APPARATUS FOR ELECTROLYSIS OF SODIUM CHLORIDE.

each cell 1 discharges. The weak brine discharged into 31 is maintained in this pipe at a constant level, there being sufficient space above it to permit the chlorine gas to collect. The weak brine is discharged from 31 through 32 into the resaturating tank 33, which contains salt. The solution flows over to 36, from where it is pumped to 38. The chlorine gas is conducted from 31 through 40 to a series of absorbing towers 42, through which it passes in a circuitous course while being absorbed by lime water.

Electrolytic Manufacture of Metal Tubes.—O. Dieffenbach, 817,419, April 10. Application filed January 9, 1906.

Prof. Dieffenbach makes copper or nickel tubes by electrolytic deposition on a revolving horizontal or inclined cathode. He adds to the electrolyte a porous granular inert material. The rotary movement of the cathode carries this material upwards on one side, and lets it slide down again on the other side. By impinging against the cathode this material removes any small adhering bubbles of hydrogen and smoothes the deposit. The only suitable material of this kind for making tubes of any desired dimensions is kieselguhr.

Chlorine and Phosphate.—A. Clemm, 819,410, May 1. Application filed Feb. 27, 1905.

Phosphate—as, for instance, Florida Algier phosphate—bones or the like, are treated with hydrochloric acid, and the solution of phosphate is separated from the residue and any excess of hydrochloric acid is neutralized, until a precipitate begins to form at the surface. This solution of acid phosphoric acid salt and metal chloride is subjected to electrolysis. Chlorine is evolved at the anode, and at the cathode hydrogen is evolved, and a precipitate of phosphate ($Ca_2H_2P_2O_8$) is formed. This precipitate of phosphate is completely and readily soluble in a 2 per cent solution of citric acid. The composition of the liquid to be electrolyzed may be changed; for instance, one may mix together a solution of acid calcium phosphate with chloride of calcium (or chloride of magnesium or the like), the electrolysis of which gives the same result. The first mode of working is especially applicable where greater quantities of hydrochloric acid can be had at low cost, and the second where chloride of calcium or chloride of magnesium can be obtained at low cost, as by products or waste, and where a solution of acid calcium phosphate, such as that obtained in the extraction of low-grade phosphates, be treating the latter with dilute sulphuric acid, can be produced more cheaply than by treating them with hydrochloric acid.

Production of Ammonia.—J. A. Lyons and E. C. Broadwell, 816,928, April 3. Application filed Sept. 2, 1904.

Fig. 5 shows the crucible *A* forming the cathode and the anode *C* of carbon in the center within a pipe or annular jacket *D* of graphite, which serves as a diaphragm. The arrows *E* indicate the course of the current. The fusible mixture of the positive metals or a mixture of the same (such as of potassium, sodium, manganese, chromium, etc.) are maintained in fusion in the crucible. The passage of the current produces electrolysis and generates an intense heat at the carbon anode, due to the high anodic current density. At the same time nitrogen gas or nitrogen bearing gases are passed from the vessel *F* into the bath through the annulus or pipe *D* around the anode. "The electrolysis produces anion, boric anhydride and oxygen at the anode, where the intensely heated carbon acts as a reducing agent to chemically reduce the anion to boron. The pipe serves to prevent the boron thus formed from floating over to the cathode, and the nitrogen introduced into the pipe combines with the boron to form boron nitride. The metal of the borates will be deposited at the cathode." After sufficient boron nitride has accumulated it is subjected, while separated from the metals, to steam, at a temperature of 600° C., or above, yielding boric anhydride and ammonia, according to the reaction:

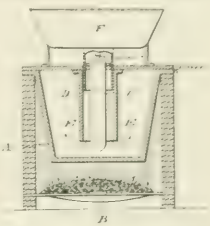


FIG. 5.—PRODUCTION OF AMMONIA.

$$2BN + 3H_2O = B_2O_3 + 2NH_3$$

The temperature of the bath is maintained at about 1,000° C.

Electroplating Tank.—C. G. Backus and G. L. Wallace, 817,832, April 17. Application filed July 27, 1905. (Assigned to Zucker & Levett & Loeb Co.)

The patent refers to a convenient and cheap electroplating tank with a rotatable receptacle containing a large number of articles to be plated at the same time. The left-hand diagram of Fig. 6 shows the complete apparatus, the drum being

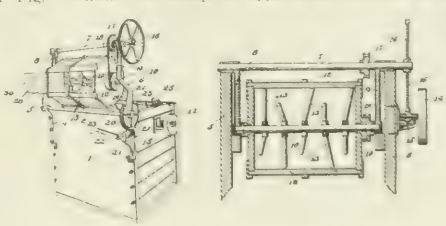


FIG. 6.—ELECTROPLATING TANK.

raised to be charged with the articles to be plated. The right-hand diagram shows the interior of the drum and the way in which it is mounted. The apparatus is very convenient to handle in electroplating practice. Its operation is obvious from the illustration.

Nickel Plating.—J. W. Aylsworth, 817,152, April 10. Application filed Sept. 17, 1904.

This patent refers to improvements in Lux's nickel process (described and illustrated in our Vol. III, p. 814) for nickel plating of long perforated strips of thin sheet iron or steel (which are subsequently cut up into blanks for use in the Edison storage battery). In his old process the only the passage of the strip through a chamber in which it is set

rounded by hydrogen gas and is heated by the current, or otherwise to a welding temperature, so as to cause the nickel film to be welded in place to remove any condition of tension therein. The inventor has now found that this is not necessary as a part of the plating process, and that superior results can be secured by placing a large number of the reels carrying the nickel-plated strips thereon in a muffle in which they are heated to a welding temperature in the presence of hydrogen gas, since in this way the operation can be very economically performed.

Electrolytic Decomposition of Water.—W. F. M. McCarthy, 816,355, March 27. Application filed Nov. 8, 1904.

Two receptacles, each containing one electrode, are connected by a conduit near the bottom. Each electrode is a plate of platinum coiled upon itself a number of times and has a projecting terminal portion directly opposite the end of the conduit. See also his former patents noticed on pages 113 and 192 of our March and May issues.

Water Purifier.—F. B. Hinkson, 820,113, May 8. Application filed May 31, 1905.

Claim 1 reads: "In a water purifier, the combination with a shell or casing having a water-inlet at one end and a water-outlet at the other end, of a series of annular electrodes located within the shell or casing and disposed, one within another, means for insulating said electrodes from each other and from the shell or casing, and electrical conductors connected, respectively, within the inner and outer electrodes."

Electrolytic Purification of Liquids.—L. Dion, 819,209, May 1. Application filed June 3, 1904.

Details of construction of apparatus for purifying a liquid by passing it between groups of positively and negatively charged electrodes.

ELECTRIC DISCHARGES.

Electric Discharges Through Air.—J. E. Mitchell and D. Parks, 817,082, April 3, Application filed Jan. 3, 1906. (Assigned to Alsop Process Co., St. Louis, Mo.)

The apparatus consists essentially of an electrifying chamber, two pump cylinders and an induction coil. In the middle of the electrifying chamber a stationary horizontal rod is provided, which is connected to one pole of the electricity supply. In line with this rod and on either side of it two movable electrodes are provided, which serve as the other electrode. These movable electrodes get a reciprocating movement by being mechanically connected with the piston rods of the pump cylinders which pump air into the electrifying chamber. Alternately one or the other of the movable electrodes makes contact with the corresponding terminal of the central stationary electrode. By breaking this contact an electric discharge takes place through air, while simultaneously air is pumped through the chamber. The machine is claimed to be compact, simple and durable and adapted to operate at a high rate of speed.

Bleaching and Sterilizing. S. Leatham, 816,482, March 27. Application filed July 16, 1904.

In order to bleach flour, cotton yarn, etc., or to sterilize milk or food products, the inventor passes air successively through two apparatus, in one of which it is subjected to the action of a silent electric discharge, yielding chiefly ozone, and in the other one to a spark discharge. Both the ozonizer and the sparking apparatus are operated from the same electric circuit and are connected in series.

Apparatus for Indicating the Character of Electric Effluvia. J. Gerard, 818,534, April 24. Application filed June 14, 1905.

In the production of electric effluvia for producing ozone etc., the efficiency of the operation depends upon the character of the effluvia. They may be best studied according to the inventor by shunting the discharge-gap by a vacuum tube and observing the rays emitted from the vacuum tube to physical analysis and observation (by spectroscopic methods, etc.).

BATTERIES.

Alkaline Storage Battery.—T. A. Edison, 817,162, April 10. Application filed Sept. 29, 1904.

In the Edison iron-nickel cell the negative active material when charged consists of a mixture of reduced iron and mercury. When the cell is shipped in this condition, the electrolyte being removed, the iron becomes oxidized, generating heat and driving off more or less mercury. This reduces the capacity of the plate. To safely ship the cell without electrolyte, the negative plate should be completely discharged, so that the iron is fully oxidized and no atmospheric oxidation can take place.

Alkaline Storage Battery.—T. A. Edison, 821,032, May 22. Application filed Sept. 28, 1904.

In the practical use of his iron-nickel battery, Mr. Edison found that bulk for bulk finely divided iron, obtained by reducing ferric oxide, is much more active electrolytically than the nickel hydroxide which he has so far been able to obtain practically. Consequently to present the best combination for practical use, the bulk of iron used need be only half that of the nickel. He uses pockets of the same size for both the nickel and the iron active materials, but uses twice as many nickel grids as iron grids. No insulating separators are required between adjacent nickel grids, which may therefore be allowed to swell, so as to touch each other without affecting circulation and the general operation of the cell.

Storage Battery Plate.—J. R. Macmillan, 817,498, April 10. Application filed April 1, 1905.

In claim 1 the inventor claims "in a storage-battery grid unit the combination of an inclosing frame, a plurality of horizontal ribs extending across said frame, a plurality of transverse ribs extending across said frame and intersecting said horizontal ribs to form compartments for lodgment of active material, projections on said transverse ribs extending into said compartments, said ribs and projections being quadrilateral, the top and lower edges thereof being disposed in a common vertical plane, and a web on said ribs and projections disposed in said vertical plane."

Battery Plate.—O. H. Fay, 820,040, May 8. Application filed Aug. 19, 1904.

Details of construction of the grid for battery plates of the Faure-Brush type. Claim 1 reads: "A battery plate, comprising end bars, a series of longitudinal bars provided with a series of rearwardly-flaring grooves, and a series of strips extending across the opposite faces of the plate, and in staggered relation to each other."

Storage Battery.—W. H. Palmer, Jr., 817,132, April 3. Application filed Feb. 14, 1905.

Details of suspension and connections of the positive and negative plates, alternately arranged.

Dry Battery.—C. J. Hirlimann, 820,047, May 8. Application filed May 25, 1905.

"A dry battery comprising a zinc cup or outer casing containing absorbent material, a carbon element therein having a straight cylindrical hole throughout its length and having narrow radial slots in the wall thereof, said slots extending from a point near the top of the carbon element to a point near the bottom thereof and of a width insufficient to permit the entrance of the sawdust or similar material therethrough, whereby the exciting material or reagent may be run into the hole to fill the same and afterward pass through the slot to impregnate the absorbent material to a predetermined degree of saturation, and a stopper or closure inserted into the upper end of the hole in said carbon element."

Binding Posts.—J. D. Warren, 819,389, May 1. Application filed May 2, 1905.

Mechanical details of connecting the electrodes of batteries with binding posts.

SYNOPSIS OF PERIODICAL LITERATURE.

THEORETICAL AND EXPERIMENTAL.

Osmosis.—The March issue of the *Journal of Physical Chemistry* contains a long and elaborate paper by L. Kahlenberg on the nature of the process of osmosis and osmotic pressure with observations concerning dialysis. The main conclusions are that whether osmosis will take place in a given case or not depends upon the specific nature of the septum and the liquids that bathe it; and if osmosis does occur these factors also determine the direction of the main current and the magnitude of the pressure developed. The motive force in osmotic processes lies in the specific attractions or affinities between the liquids used, and also those between the latter and the septum employed. These attractions or affinities have also at times been termed the potential energy of solution, etc., they are to the mind of Prof. Kahlenberg essentially the same as what is commonly termed chemical affinity. It has been emphasized that osmotic pressures are equilibrium pressures, and that in osmotic processes there is always a current in both directions, though the main current may in specific cases be so much stronger than the minor that the latter almost sinks into insignificance. In such cases the septum employed is termed "semipermeable." Vulcanized caoutchouc is a ."semipermeable" membrane when it separates pyridine solutions of silver nitrate, lithium chloride and cane sugar from the pure solvent. The necessity of stirring the contents of the osmotic cell and also the outer liquid during osmotic pressure measurements is pointed out, and a new apparatus for measuring osmotic pressures accordingly has been devised. The results of the osmotic pressure measurements show that the gas laws do not hold; and it is, consequently, pointed out that the latter cannot serve as a basis for a satisfactory theory of solutions. The advantage of stirring-in processes of dialysis is indicated, and it is shown that whether substances pass through membranes or not does not depend upon their colloidal or crystalloidal character, but solely upon their affinity for the membrane employed and for the liquids that bathe it.

A New Way of Fixing Doubtful Atomic Weights.—In a recent lecture by Prof. J. J. Thomson, before the Royal Institution in London, on the corpuscular (electronic) theory of matter, reference was made to the remarkable discovery that if the secondary radiation emitted by elements under the influence of Roentgen rays be measured by the ionization it produces within a millimeter of the surface of the substance, numbers are obtained that rise steadily with the atomic weight. If the ionization be plotted on an atomic weight base, a curve of such definiteness is obtained that it may prove useful in the future in fixing doubtful atomic weights.

METALLURGY.

IRON AND STEEL.

Blast Furnace Practice.—Director Oskar Simmersbach contributes a well-illustrated series of articles to *Stahl und Eisen* (March 1 and 15, April 1 and 15), on recent advances in this line. His descriptions of apparatus, however, are mostly confined to those in European practice, which are rather behind than ahead of the best American practice. Some of the most interesting items are the following: The method of opening up a frozen tap-hole by means of oxygen at 30 atmospheres pressure, is gaining in popularity. The frozen iron is first heated to redness by an oxy-hydrogen flame, then subjected to the oxygen blast alone, which burns iron, silicon, carbon, phosphorus and manganese to such good effect that each kilogram of material burnt will melt 4.5 kilograms of solid material. Holes a meter long can be melted in about 1 minute, in solid iron, at small cost. Opening a frozen tap-hole in this manner costs usually $0.75 to $1.25, seldom as much as $2.50. Copper or bronze are not injured by the oxygen blast At Pont-à-Mousson the workmen (perhaps rather *strikemen*) suddenly walked off on strike, leaving five blast furnaces in full operation. They were at once banked, and on again resuming three started all right, the fourth had frozen iron in the tuyeres, the fifth had frozen [illegible] tuyeres. Dr. Menne's oxygen specific was called into action, an ironmaster from Creuzthal melted out every tuyere of furnace number four in short order, and [illegible] once. The fifth furnace was, however, [illegible] temporary tuyeres were opened up above the "sow," and then the iron melted out at a lower point. The melting [illegible] peated at lower and lower points in the crucible, until on the fifth day the normal tap-hole was in operation, the normal tuyeres were melted out and started, and this *Lazarus* of blast furnaces was transacting business at the old stand. The remedy for the two furnaces was twenty-six "bottles" of oxygen, costing altogether $65.

Creuzthal has further distinguished itself, in the person of Director Dresler, by "banking up" a blast furnace fifteen months, and then having it in normal running order in 4 hours after putting on hot blast. The method is simplicity itself (when you know how); the furnace is simply allowed to "go out," and it might stay out fifteen years if [illegible] and start up just as promptly. As for details, the hearth is emptied, the charge line allowed to sink a little, and then the top of the charges covered with sheet iron, and a layer of wet clay stamped in on top until the furnace charge is shut up hermetically, when the bottom openings are also closed. Then one waits until the strikers come to reason, or a new iron-ore district is discovered, or a new ministry comes in and changes the tariff—and then the clay is dug out, the sheet iron removed, the tuyeres opened, hot blast turned on, and in 4 hours—*viola*.

Director Simmersbach finally discusses Gayley's dried blast, without developing anything new, and, in fact, without thoroughly understanding the question in all its bearings. But, afterwards, he closes with some remarks on the possibility of using air blast richer in oxygen, which are very much to the point. He points out that blast richer in oxygen would result in a higher temperature before the tuyeres, an easier reduction of silicon, a gas richer in carbonous oxide (CO), a quicker reduction of the charge, a larger proportion of CO in the gases (this prediction is just the opposite of what would occur), economy in fuel and increased output. With the parenthetical exception just made, we agree with all of these predictions; it only remains to enrich the blast in oxygen at a cost less than the money value of the above catalogued advantages.

Blast Furnace Profiles.—Prof. Bernhard Osann, of Clausthal, delivered an interesting lecture on this subject before the Southwest Luxemburg Ironmasters on March 18, which is reported in full in *Stahl und Eisen* for April 15. From a comparison of many modern blast furnaces the following general dimensions are deduced as those found best by experience:

Output in Metric Tons Per Day	Diameter of Hearth Meters	Depth of Hearth Meters	Height to Tuyeres Meters	Height to Slag Line Meters	Diameter at Meters
40-60	2 -2.5	1.5	1.2-1.3	two-	3 - 3.5
60-150	2.5-3.5	up to 2.0	1.3-1.8	thirds	3.5 -
150-350	3.5-4.0	" 2.5	1.3-1.8	height of	3.5-4
350-600	4.0-4.7	" 3.1	1.8-2.0	tuyeres	3.5-4

In all modern furnaces the angle of the boshes is [illegible] 75° or 76°, and the inclination of the shaft walls about 80°. Calling r the radius at the top of boshes, r_1 at the bottom of boshes, r_2 at the throat, and h_1 the height of the [illegible] height of the shaft, then

$$h_1 = 4\,(r - r_1)$$

$$h_2 = 14.3\,(r - r_2)$$

and the volumes of the shaft and boshes together [illegible]

$$V = 1.05\,h\,(r^2 + r^2 + r\,r_2) + 1.05\,h_1\,(r^2 + [illegible]$$

This facilitates very much the calculation of cubic contents of boshes and shaft, which are the effective [illegible] of the furnace. The time of treatment of a [illegible] (Durchsatzzeit) varies from 24 hours down to [illegible], the latter

only attainable by high blast pressure, good strong coke and when not making a high silicon iron. The lecture contained many other generalizations, but they apply only to regular practise with regular ores, so that the results possess no exact value for general application.

Cementation.—Dr. Eng. R. Bruch describes in *Metallurgie* for February 22 his very carefully conducted investigation of the cementation of wrought iron by gaseous carbonizing reagents. A bar of wrought iron 70 m. m. long by 11 m. m. diameter was heated in a porcelain tube, in each test for 7 hours, at temperatures of 600, 700, 800, 900, 950, 1,000, 1,050, 1,100, 1,150 and 1,200° C. The bar was afterwards turned off in five successive layers of 0.5 m. m. thickness each, and each carefully analyzed for carbon. The wrought iron contained 0.03 carbon, 0.023 silicon, 0.035 manganese, 0.014 phosphorus and 0.008 sulphur. The various tests resulted as follows: *Illuminating gas* (containing 33 per cent. CH^4 and 3.5 per cent. higher hydrocarbons) showed practically no carbonization below 700°, slow carbonization (up to 0.5 per cent.) up to 1050°, and rapid cementation (up to 1.5 per cent.) between 1050° and 1100°. *Petroleum vapor* showed slower carbonization, practically none at 700°, up to 0.25 per cent at 1050°, and rapid carbonization up to 1.5 per cent. between 1050° and 1100°. *Acetylene* showed very little carbonization below 900°, 0.25 per cent. at 950°, 0.6 per cent. at 1000°, 0.8 per cent. at 1050°, and 1.2 per cent. at 1100°. The strong carbonization commences sooner than with the first two reagents. Carbonous oxide gas (CO) produced no carbonization at all, in 7 hours, at temperatures between 500° and 1100°. The net conclusion is that illuminating gas, petroleum vapor and acetylene all act strongly carbonizingly on iron, very slightly at 700°, faster at 900°, and fastest at 1050° to 1100°; that the process is one of solution and diffusion of the carbon in the iron, and that it is being utilized practically on an extensive scale.

Graphitic Carbon in Cast Iron.—Engineer J. Kreiten gives us in *Metallurgie* for March 22 the results of tests to show the influence of tin on the separation of graphitic carbon in cast iron. Up to 3 per cent. of tin was added. In irons containing 3 per cent. and 1.3 per cent. of silicon, in which graphitic carbon was 70 per cent and 64 per cent of the total carbon, the effect of tin up to 3 per cent was negligible—that is, it did not practically increase the graphitic carbon. With 0.01 per cent of silicon present, the effect of tin was small until it exceeded 1 per cent. At 1.25 per cent of tin, 7 per cent of the total carbon became graphitic; at 1.81 per cent tin, 10 per cent; at 2.80 per cent tin, 37.3 per cent, or over one-third of all the carbon, became graphite. Higher percentages of tin lowered the total carbon, as well as the graphite, the former faster than the latter.

In *Metallurgie* for April 8, F. Wüst describes similar tests on the effect of sulphur. This element reduces the total per cent of carbon present, and strongly reduces the amount of graphite. With 0.6 per cent sulphur, even an iron with 2 per cent of silicon contained only 0.12 per cent graphite. Phosphorus, however, was shown to strongly increase the formation of graphite when it exceeded 2.5 per cent; in an iron with 0.4 per cent silicon, 6 per cent of phosphorus increased the graphitic carbon from 0.07 to 1.82 per cent; in an iron with 0.9 per cent of silicon, phosphorus up to 3 per cent had no effect, but above that increased the graphite per cent about 0.25 per cent for 4 and 5 per cent of phosphorus.

SILVER.

Alloys with Arsenic.—K. Friedrich and A. Leroux have studied these alloys in the Freiberg Academy laboratory and report their results in *Metallurgie* for March 22. They found it possible to get alloys with 25 per cent of arsenic, which, however, fell to 19 per cent while cooling slowly when melted. The cooling curves of all the alloys tested were very simple, an upper setting point and a lower resting point, the former gradually falling from 962° to 527° as the content of arsenic increased towards 25 per cent., the latter being constant at 527° for all the alloys, and therefore corresponding to a eutectic with probably 25 per cent of arsenic, and melting at 527°.

RECENT METALLURGICAL PATENTS.

REDUCTION OF FINELY DIVIDED ORES.

Two patents of the late Horace F. Brown (817,414 and 817,415, April 10) refer to his process for reducing of finely divided ores of iron, copper, lead and zinc, such as flue dust, mill dust, fine concentrates, oxide of iron sands and the like. The ore is first fed into the upper part of a vertical stack, there subjecting the particles while in suspension, and in a more or less segregated state, to the action of a reducing atmosphere moving in the same direction as the ore, whereby the latter is reduced, the particles being at the same time brought "into approximate physical contact." If necessary, this stack may contain two chambers, one above the other, of which the upper one carries a very high temperature with little or no reducing atmosphere, while the lower chamber contains an excess of reducing gas. It is vitally necessary that after reduction and before the second step of the process—separation of the reduced metal from the gangue—the metal shall be protected from contact with an oxidizing atmosphere or reagent. For this purpose the excess of reducing gas is provided at the bottom of the stack.

The deoxidized metal falls from the stack into a fused bath in a reverberatory furnace or forehearth, where it is protected against oxidation by a cover of molten slag while being further heated to complete fusion. On account of the differences of specific gravity the metal is separated here from the gangue, which goes into the slag. The reducing gas is mixed over the surface of the gas with an excess of oxygen to further its combustion and heat the bath. In practice carbonating and slag-producing ingredients may be fed with the ore to the stack if desired.

Iron-Sand Reduction. As applied to the treatment of iron-sand the process is as follows: In this case a comparatively high temperature is required for the reduction of the iron oxide, and the stack, therefore, contains two chambers, as described above. The upper one is for preheating the rain of finely divided iron oxide and silica particles with an excess of lime to a temperature in which the iron oxide is ready for deoxidation. The reduction begins at the lower end of the upper chamber and is completed in the lower chamber, while the lime combines with the silica and alumina of the charge, forming a silicate of lime and alumina. The iron reaches the lower end of the stack as metal. The excess of lime forms a basic slag, so that when the reduced iron is once beneath the surface of the bath it is protected from oxidizing action.

Treatment of Lead and Copper Oxide Fines.—In this case a lower temperature is to be maintained than in the case of iron. While descending through the stack the impurities are formed into a slag, of which iron, silica and lime form the main constituents. The reducing gases and the heat are so regulated that the iron is not deoxidized, but unites with the lime and silica, forming a slag of silicate of lime and iron, while the oxides of lead or copper are reduced to the metallic state.

FLAME REGULATION.

The "Eldred Flame." Our readers who are well acquainted with this subject from the articles of Mr Ellis (our Vol II., p. 505) and Mr. McElroy (p. 141 of our present volume) will be interested in two recent patents granted to Mr B. L. Eldred. One (819,045, May 1) refers in its second claim to "the art of heating materials in furnaces of the reverberatory type, which consists in transmitting through a body of ignited fuel a draft-current of air diluted with products of

combustion in such proportion as to check and cool the combustion of such fuel and to retard combustion of its gaseous products, passing said gaseous products into a refractory-wall chamber containing the material to be heated and there completing their combustion by radiant heat from the walls." The other patent (819,046, May 1) refers to means for controlling the gas velocity in reverberatory furnaces in which several fire-boxes are employed. The reverberatory furnace is provided with a plurality of successive flame-introducing means at different points along its hearth chamber, so as to successively increase the flame volume. Beyond each such flame-introducting means the internal diameter of the hearth chamber is increased to correspond to the increased volume of flame.

ROASTING FURNACES.

McDougall Roasting Furnace.—The work done by Mr. Frank Klepetko in recent years with respect to details of construction of the McDougall type of ore-roasting furnace is well known to our readers. Three patents recently granted to him refer to modifications of constructions patented by him before. In the first patent (814,297, March 6) he proposes to introduce the cooling medium into the hollow rabble shaft at a point where in the majority of cases the furnace is hottest. This is the middle of the furnace, the upper hearth being cooled more or less by the introduction of the charge and the bottom hearth receiving the material after its fuel contents has been consumed during the roasting operation. Under these conditions the introduction at the middle point results in the formation of two distinct currents of the cooling medium, one ascending and the other descending, suitable outlets for the discharge being located at opposite ends of the rabble shaft. The shaft is provided with a series of hollow rabble arms (each series extending into its corresponding hearth), and is divided into distinct compartments communicating with the arms; it follows that the currents of the cooling medium will be parallel in opposite directions along the shaft, and so far as the arms are concerned the circulation will be vertically in series through the arms and radially in multiple. The resulting circulation, therefore (from the point of introduction in the shaft), through the shaft and arms will be vertically "in parallel" and "in series," and radially "in multiple."

The second patent (814,298, March 6) proposes to provide the hollow rabble shaft with longitudinal division walls or diaphragms dividing the passage of the shaft into two or more independent conduits for the circulation of independent currents of cooling mediums, so that in the event of a leak or puncture of the walls of one of the conduits the remaining conduits may not be affected, thus obviating the necessity of an immediate shut-down of the furnace. The diaphragms also permit of the introduction of cooling mediums from independent sources, so that water may be made to circulate through one or more of the conduits and air through the balance, etc.

The third patent (814,294, March 6) refers to the division of the rotary furnace into a series of sections or compartments corresponding to the number of hearths of the furnace, each section being cooled independently of any adjacent section by means of a circulating medium injected from a common source of supply; the circulating medium is discharged from each section or compartment into a common return pipe or outlet leading from the furnace. The advantage of such a construction is that the circulating medium may be cut off for such hearts in which there is no longer any danger or injury to or destruction of the rabble apparatus from the heat inherent in the charge contained in the hearths.

GOLD AND SILVER.

Refining Finely Divided Hydrometallurgical Products.—In refining finely divided precipitates from syanide, chloride or hyposulphite solutions or electrolytic deposits or residuum, difficulties are experienced pri[illegible] losses. Charles W. Merrill (815,881, [illegible]) materials with litharge and adds a soluble lead salt and briquets the mixture. It is then fused in a cupel or crucible, whereby the litharge and soluble lead salt are reduced to metallic lead, fresh material being periodically added. The lead collects and contains the finely divided precious metals, and is then cupelled off and [illegible] in subsequent operations. A special advantage claimed for the process, besides preventing dust losses, is that the by-products are of very low value, for the reason that the lead is constantly percolating downward through the slag and carries the values with it.

Treating Ores Containing Selenium.—A process of A. C. Atwater (817,411, April 10) relates to [illegible] of precious metals containing selenium, [illegible] sulphide ores. The granulated ore is mixed with a th[illegible] amount of granulated carbon, so as not to have an excess of the latter. The mixture is placed in a crucible over which is placed a tight-fitting inverted crucible, which has a small hole in the bottom. The crucibles are then placed in a furnace, and commencing with a low degree of heat the latter is gradually increased to a clear red, and continued at a steady heat, say from 1 to 2 hours. The selenium is thereby driven off through the hole in the top. When no longer gases are escaping, the crucibles are taken from the furnace and the hole is tightly closed with fire-clay, to prevent oxygen from entering the crucible. When cold, the material is in a condition in which the precious metals may be readily recovered by amalgamation, cyanidation, etc.

Wet Process for Recovering Precious Metals.—John A. Just (820,000, May 8) proposes the following process for recovering gold and silver directly from the ores without preliminary roasting or chloridizing. The object is to convert the precious metals into oxides and simultaneously dissolve the oxides thus formed in a suitable acid to form soluble salts with that acid, then extract them with water or with an acid in which the salts will readily dissolve. The solution is then separated from the gangue or sands and the precious metals are recovered from the solution. The inventor uses sulphuric acid together with an oxygen salt mixed with the finely ground ore. The acid liberates oxygen from the oxygen salt, so that the oxygen will act in its "nascent state." As oxygen salts, oxide of manganese, native or artificial, potassium permanganate, sodium nitrate, lead peroxide and sesquioxide, red lead, etc., may be used. The chemical reactions are discussed at some length in the patent specification. Claim 1 refers to "the process of extracting precious metals from [illegible] terials containing said materials with nitrosulfonic [illegible]"

NICKEL.

Nickel-Copper Alloy.—A. Monell (256,677, Jan. 30) patents the production of a copper-nickel alloy directly from the nickel-copper ores without previous separation of the nickel from the copper. Ore containing sulphides of nickel and copper is smelted, the resulting matte is [illegible] and then calcined, to remove the sulphur and to leave the nickel or copper in the form of oxides. The oxides are reduced in a reverberatory furnace with carbon, so as to [illegible] containing about two parts of nickel and one [illegible] "Such an alloy is much cheaper to work [illegible] than copper, and therefore affords a less expensive [illegible] product, but equally non-corrodible, which [illegible] the same tensile strength and elongation as [illegible] the kind generally used for boiler plates." The alloy is much stronger than copper, and is freely malleable, [illegible] It can be used for the manufacture of [illegible] plates, castings, fittings, propellers and other uses where [illegible] strength and freedom from corrosion are required. It may also be used to advantage in the manufacture of [illegible] boiler tubes."

BOOK REVIEW.

Physical Chemistry for Electrical Engineers. By J. Livingstone R. Morgan, Ph. D. 12mo., VIII. 230 pages. Cloth, $1.50 net. New York, John Wiley & Sons.

The contents comprise 15 pages on fundamental principles, 18 on the general properties of gases, 16 on heat and its transformations, 47 on solutions, 36 on chemical mechanics, 39 an equilibrium in electrolytes, 38 on electrochemistry proper, and 10 pages of problems for solution. The aim of the author has been to discuss "those laws and generalizations of physical chemistry which form the basis of the subject embracing the chemical application of electricity and the electrical applications of chemistry." The guiding principle of the author has been to avoid "*the use of any hypothesis*," to place the subject "upon an absolutely experimental basis," and to draw no inference "not justified *in all its parts by actual results*."

Our opinion of the work is that the author's aim has been very satisfactorily realized, that he has furnished the clearest possible introduction to the understanding of modern physical chemistry; and as such we recommend it most heartily to electrical engineers desiring to lay a good foundation for mastering electrochemistry. As far as the author's guiding principle is concerned, he has stuck to it bravely, but has slipped up by admitting some hypothesis (such as Nernst's solution-tension theory of metallic electrodes). However, the author's intentions were so highly commendable, and in greater part so admirably adhered to, that the few slips can be cheerfully excused.

Gas Power.

By Oskar Nagel, Ph. D.

It is the purpose of this article to explain the causes of the immense progress of the gas engine industry.

In selecting a motive power for large power plants in most cases steam and gas power has to be considered.

For the small manufacturer the selection of the most convenient motive power is very simple. The small size boiler plant has long ago been replaced by the illuminating gas engine, while for very small powers and discontinued use electric motors, driven with current from electric central stations, are used to a very large extent. Since the development of producer gas plants, which are entirely independent of illuminating gas plants, gas engines are used to great advantage for continuous use and larger powers.

Producer gas power plants, on account of their independence of illuminating gas plants, are of importance in districts where illuminating gas cannot be had, thereby replacing steam, oil and gasoline engines. The two latter kind, however, are to be recommended for portable machines and for places where discontinued work is carried on and where quick starting is desirable.

More difficult is the selection of a power plant for large industries. Ten to fifteen years ago gas engines were used only in exceptional cases for larger plants, especially in densely inhabited city districts. Some pioneers already at that time were strongly advocating the use of gas engines and producers. However, the steam engine naturally had more friends on account of the long established reliability, and on account of being much more considered in technical high schools than the new gas engine. The gas engine builder had many difficulties to overcome. But the reliability and economy of the gas engine was fully proved on a number of medium sized gas power plants; then the iron industries, having at their disposal a low-grade fuel gas, the blast furnace gas, especially adapted for driving gas engines, developed a demand for gas engines of utmost size. While a few years ago gas engines of 100 hp. capacity were considered as exceptionally large, we find to-day a large number of gas engines running having a capacity of several thousand horse-powers each.

On account of the much lower running expenses as compared to steam engines of equal size, the gas engine has attracted wide attention of parties interested in large power plants. In small plants the difference in cost of running is very great, somewhat smaller in larger plants, but even there very considerable.

The utilization of heat in the gas engine is very much higher than in the steam engine, which is the main cause of the saving in running. Furthermore, the generation of gas in producers entails considerably less loss than the making of steam. In illuminating gas engines the constant readiness and convenience is of advantage. But in all gas power plants the supervision of boiler plants is done away with and also the smoke nuisance.

The first experiments for burning gas in engines were made in the first half of the nineteenth century. Lenoir, a Frenchman, was the first who built a practicable machine, however, without compressing the air-gas mixture. In 1875, Otto built the first four-cycle machine, which, however, was not used at that time. A watchmaker in Munich had built a four-cycle engine several years before Otto.

Four-cycle is called the alternate use of the same cylinders once as pump for drawing in and compressing the gas-air mixture and then as power cylinder and for driving out the exhaust. One impulse takes place at every fourth stroke (cycle).

In the beginning 30 to 45 pounds of compression was applied and 35 cubic feet of gas required to develop 1 hp-hour. Later on the compression was increased so that at present in the smallest engines only 50 per cent of the above-named quantity of gas is required. In larger engines only 40-35 per cent is used, and the efficiency increased to about 30 per cent, while in large steam engines 16 per cent is considered to be the best result.

Afterwards the gasoline and oil engines were developed. Gasoline, on account of the easy combustion with the air is very convenient, while the handling of oil is more complicated.

The above-mentioned gas producers were first built by Dowson. Hard fuels, as, for instance, anthracite or coke, are very well adapted for use in these producers; bituminous fuels, on account of their tar contents, require a more complicated arrangement.

If fuel is charged into a shaft furnace (producer), lit and air blown through the fuel, a combustion takes place first of all to carbon dioxide. In going through the upper fuel layer carbon dioxide is reduced to carbon monoxide, which is a combustible gas, and forms the main part of the so-called producer gas. The other components of this gas are nitrogen and small quantities of carbon dioxide, hydrogen and hydrocarbons.

Such producer gas is used in the industries to a very great extent, and is made as a by-product in all blast furnaces, which are, in fact, huge gas producers.

By adding steam to the gasifying air, the gas gets richer in hydrogen and thermal value, by the decomposition of steam into hydrogen and oxygen. The oxygen forms with the carbon carbon monoxide, and thereby decreases the amount of air and of the inert nitrogen to be introduced into the producer. Furthermore, the producer is running cooler and lasts longer.

One pound of carbon yields about 84 cubic feet of gas of 145-150 B. T. U., which is an efficiency of 80-90 per cent, a result which cannot be approached with steam boilers. This combined with the greater efficiency of the gas engine, causes a great decrease in fuel consumption as compared to steam plants.

In the first producer gas plants an independent boiler was used for furnishing steam to the steam jet blower. The firing

of this boiler means a loss in fuel, which, however, is equalized by preheating the combustion air by the hot gas which leaves the producer.

Efforts were made to do away with the boiler and to generate the steam in a vaporizer heated by the hot producer gas. Such plants have been largely introduced, especially in the so called suction gas producer plants.

In these plants the gas engine draws the air through the producer by sucking the gas through the plant. A negative pressure prevails in the system, so that the grate can be cleaned during running hours. This means a great simplification.

A fundamental difference, however, between pressure and suction type does not exist, as the gasifying process is the same in both. For large units and wherever some of the gas is to be used for fuel purposes, the combined suction and pressure system is used to advantage, the producer proper being kept at negative and the other part of the apparatus

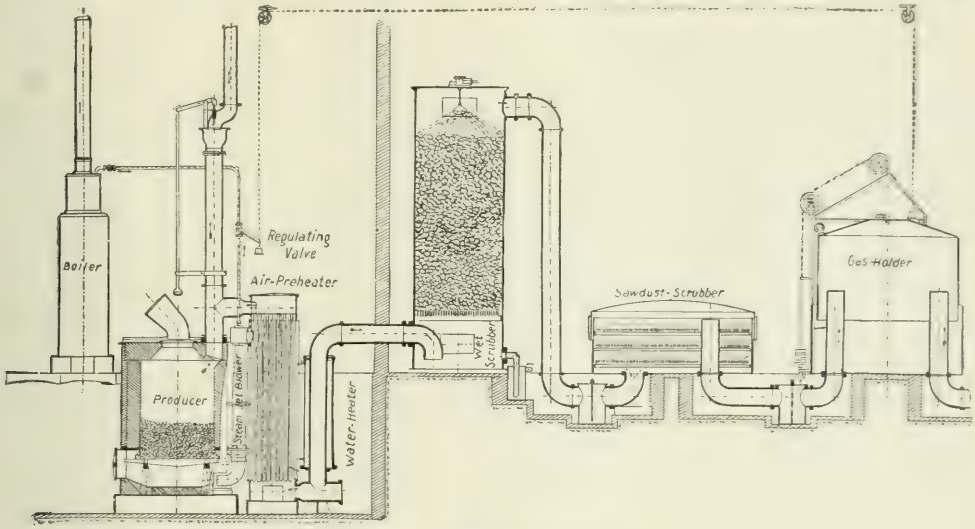


FIG. 1.—PRESSURE GAS PRODUCER.

at positive pressure, which is accomplished by inserting a fan behind the coke scrubber.

In the coke scrubber the gas is washed; ordinarily it is afterwards filtered in the so-called saw-dust purifier.

The size and construction of the purifying arrangement depends on the kind of fuel to be used; it has to be built in liberal dimensions, as otherwise considerable inconveniences and troubles are met with.

It is desirable to use in producers such fuels which, under ordinary conditions, yield a large amount of tar, such as bituminous coal, peat and lignite. The tar, besides being a loss as to its thermal value, is clogging up the pipes, and necessitates frequent cleaning.

A well constructed producer for such fuel has to gasify the tar, and utilize thereby the heating value of all its constituents.

For plants of large size, 2,000 hp. and above this is satisfactorily done in the Mond producer.

Of immense importance for the present gas engine industry is the utilization of gaseous by-products of certain industries, especially of coking plants and blast furnaces. While all of these gases were entirely put to waste up to a short time ago, they are now being more and more utilized in gas engines.

With every ton of iron about 157,000 to 158,000 cubic feet of blast furnace gas are generated. Of this about 40 per cent are used for heating the blast, 10 per cent are lost during charging, so that about 70,000 cubic feet are available. With 100 110 cubic feet of this gas [illegible] engine, which is necessary in gas furnaces to get the same result in the case of firing boilers.

The consideration of these facts has created a heavy demand for gas engines for the use in iron works. Koerting Bros., the leading German gas engine manufacturers, [illegible] over 150,000 hp. for this purpose. There are building even large engines 500 and above 2,000 hp., as the latter one gets too bulky in these units.

Up to about 500 hp., however, the four-cycle engine is at present exclusively to be recommended. To give an approximate idea of the wide introduction of the four-cycle engines and producers, I want to mention that the [illegible] in the last four years sold over 4,000 plants.

The following table will convey to the reader an idea [illegible]

At 250 hp. the saving in fuel in a suction gas power plant as compared to steam. A 250-hp. plant will be considered with 3,000 working hours per year:

COMPARISON OF 250-HP. STEAM AND GAS POWER PLANT.

	Steam.	Gas.
Cost of complete plant, including foundations	$13[illegible]	$11,25[illegible]
Coal consumed per B. H. P. hour, lbs.	3	1
Average cost of coal per ton	1.85	2.5[illegible]
YEARLY EXPENSES.		
Total cost of fuel including standby losses and loss for starting, 5 per cent for boiler, 10 per cent for producer	$[illegible]	$[illegible]
Attendance	[illegible]	[illegible]
Oil and cotton	[illegible]	[illegible]
12 per cent interest and depreciation..	1,500.00	[illegible]
Total yearly expense	$5,7[illegible]	$[illegible]
Cost per hp. hour	[illegible]	[illegible]

NOTE.—The building costs the same for steam and gas. Foundations for gas plant are 25 per cent cheaper than [illegible]

steam plant. Repairs on gas plant are only 50 per cent of the repairs on steam plant. As in most localities the difference between steam and producer coal is much less than shown in the table; the end result is generally more favorable for the gas plant.

These figures showing the superior economy of gas power will probably be criticised; one will find fault with the figures for the first cost, another with the figure for attendance, etc. This, however, is of no consequence, as sufficient figures are at hand showing actual results of German electric stations that are working with gas and stations using steam power. These figures, covering a great length of time, show clearly that 2-3

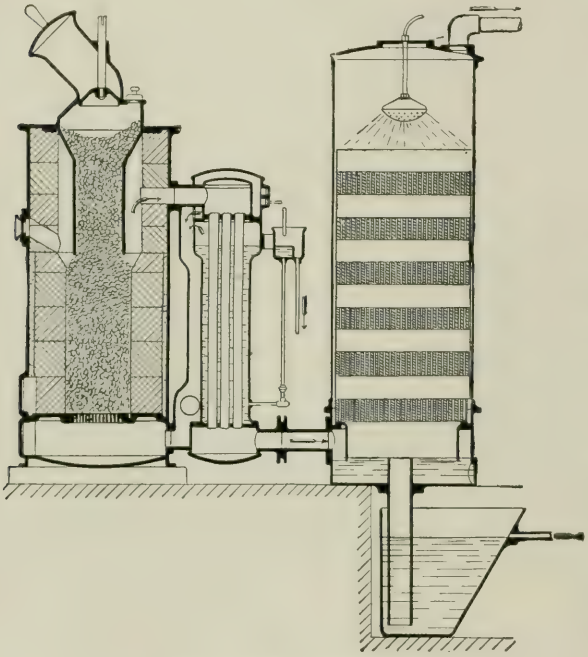


FIG. 2.—SUCTION GAS PRODUCER.

times as much total coal is used for steam power as compared to gas power.

An 80-hp. Koerting gas engine and producer, which is running at the electric central station in Lauterberg, Germany, has developed during the test run 1 hp.-hour with 0.77 pounds of coal. Such results naturally can be obtained only with the very best gas engines.

Another Koerting suction gas plant of 150 hp., which was started a short time ago at the electric central station in Doebelin, Saxony, showed the following results, using coal as fuel.

The generator, belt connected to engine, developed 98.27 kw. (guarantee 95 kw.). At an efficiency of 90 per cent of the generator and 5 per cent loss in the belt drive, this is equivalent to 134.2 hp. The coke consumption per B. H. P. hour was during the 10-hour test 1.085 pounds (guaranteed 1.1 pounds). The oil consumption was 1.86 gr. per B. H. P. hour.

The efficiency of the boiler is largely decreasing at low load and overload, while the economy of the producer remains constant within very wide limits.

Gas engines show the greatest economy at full load, while steam engines have it below this point. In both kinds of engines there is a point of highest efficiency. Up to and from this point the economy is decreasing.

It is frequently mentioned that steam engines are able to carry a greater overload than gas engines, in which 10-15 per cent is a maximum. However, as the steam engine is built to be loaded to the best efficiency, an overload means an overstraining of the working parts, which is to be avoided.

Sufficient experience was gained in the last ten years showing the absolute reliability of the gas engine, and a very large number of German industrial and electrical power plants depend entirely on gas power.

A change is taking place gradually in prime movers. The gas engine will more and more replace the steam engine to the advantage of our industries. Everybody, however, who thinks of installing this new motive power ought to examine carefully what is offered to him, as the comprehension and understanding in this field is much more difficult than with steam engines.

Electro-Cyanide Process.

An interesting combination of a cyanide agitator, amalgamating tables and electrolytic precipitation in a single compact apparatus is found in the Garvin cyanide machine, built by the Garvin Cyanide Extraction Co., of Portland, Ore. The adjoining illustration shows the construction.

The method of agitation may first be described. A tank *A* with a conical bottom is supported in a wooden frame, and is filled with ore and solution, the depth of solution above the ore being at least 1 foot (indicated in the diagram by the distance between "solution line" and "pulp line"). *B* is the precipitating tank, which will be described later on. The solution is drawn from the precipitating tank *B*, and its flow is regulated by valve *K*, a pipe connection being provided with the bottom of the tank *P*. At the latter place a second valve *L* allows drawing off a quantity of the settled pulp discharge into connection *P*, where it mixes with the solution drawn from the precipitating box *B*, and flows to the pump (shown at the left hand). There it is pumped up to the shaking amalgamating table *C*, which will be dealt with below. From the amalgamating table *C* the pulp passes into the hopper *J*, which discharges it at a given point into the tank *A*, as clearly shown in the diagram. A cone-shaped spreader *N* is placed in the bottom of tank, insuring a thorough circulation of the pulp and preventing it from funnelling or channelling.

It will be seen that the whole agitation is perfectly under the control of the operator by means of the two valves *K* and *L*. By opening the valves one can get as violent an agitation as may be desired. The amount of pulp that can be handled in the machine is large, the only limit being that the solution at the top of the tank must be less dense than what the pump will handle, so that it can be thickened to the consistency that will pump by opening the bottom valve.

The shaking amalgamating table *C* serves for amalgamating any coarse particles of gold that may be in the ore. The baffle boards *E* are arranged in this box so the discharge can be higher than the copper amalgamating plate, and the baffles will keep the pulp stirred up.

It remains to describe the precipitating box, which, in some respects, is the most interesting part of the apparatus. It is well known that electrolytic precipitation has been used successfully on a very large scale in South Africa, and in a few instances in this country and in Mexico. But the Garvin method of precipitation contains some peculiarities, mainly due to his using a mercury cathode and thus recovering the gold in form of amalgam. Of course, the mercury cathode is well

known to electrochemical engineers from the Castner-Kellner process of electrolysis of sodium chloride and from applications in the metallurgy of zinc, but it does not seem to have been employed before in the electrometallurgy of gold. At least, the mechanical arrangement of the mercury cathode in the Garvin apparatus is decidedly novel.

The solution is drawn or decanted from the top of tank *A* through outlet *O*, and passes then into the precipitating tank *B*. This is a box, rectangular in shape, with a circular bottom. Mounted in the bottom and the top of this box are two shafts with sprocket wheels on each end of shafts, and link chains or belts are fastened to them, and fastened to corresponding links of the chain are strips of copper, as shown by letter *G*. This is connected to negative pole of the dynamo, and is therefore called the cathode roller.

On each side of this roller is suspended a piece of sheet iron, which serve as anodes. In the bottom of this box is placed a quantity of mercury so that the lower part of the cathode roller *G* is immersed in it, and as the cathode is kept revolving it is continually being re-amalgamated. As the solution passes through this box the electric current precipitates the values from it, which are deposited in the mercury forming the amalgam.

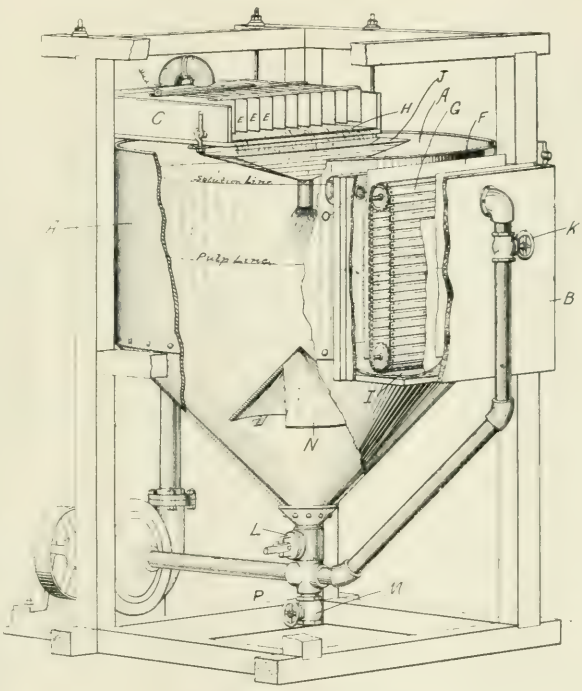


FIG. 3.—CYANIDE MACHINE.

The solution as it passes through this box is, of course, muddy, as all of the light slimes will not have settled in the main tank *A*, but as the roller cathode is continually revolving and passing through the bath of mercury a coating of slimes does not form on the cathode and prevent the values from getting to the mercury as the same is precipitated from the solution. There is nothing new in the idea of precipitating values from solutions into mercury, but in a good many methods the amount of mercury necessary to obtain a sufficient cathode surface is prohibitive; but in this case it will be seen that a very large cathode surface is obtained with a small amount of mercury.

Heretofore there has also been a good deal of trouble in precipitating during the agitating process, for the reason that if the ore was heavy in sulphides the action of the electric current would decompose them and change the chemistry of the solution. While with this machine, as the pulp is discharged into the main tank *A* the ore will settle to the bottom of tank, and only the very lightest slimes will pass through the precipitating box; hence the electric current is not applied to the main body of ore.

If we finally look at the apparatus as a whole it will be seen that it combines the following functions after ore and solution have been placed in it: Decantation; electrolytic precipitation of the values from the solution; at the same time drawing the ore from the bottom of the tank and mixing it with the precipitated solution and pumping the mixture back to the top of tank; [illegible] recovery of coarse gold particles [illegible] solution; and so on.

Ferro-Alloys.

The interesting illustrated booklet on High-Grade Alloys published by Messrs. Geo. G. Blackwell, Sons & Co., Ltd., the old well known ferro-alloy firm of Liverpool, has just appeared in its fourth edition. It contains much useful information. The following notes are abstracted from this booklet.

In the introduction it is pointed out that steel manufacture has now passed beyond the old "rule of thumb" methods of years ago, and has become a scientific process, a modern steel works being absolutely incomplete without its staff of trained chemists. For special work carbon steel does not answer requirements; and alloy steels have stepped into the gap. Armor plate, armor-piercing projectiles, guns, bullet-proof sheets, high-speed steel, motor car forgings and castings, etc., etc., are no longer made of carbon steel, but alloy steel. By alloying the rarer metals with steel, each metal having its own particular properties and effect, steel manufacturers have before them a field for making steel to answer practically any specification however severe.

The development of the electric furnace has enabled the manufacturer to place at the disposal of the steel men the metals and ferro-alloys practically unheard of a few years ago, and of a purity and excellence that would be impossible to surpass. Concerning electric furnace methods of making ferro-alloy see, for instance, our Vol. II., p. 349, 395, 449. Messrs. Geo. G. Blackwell, Sons & Co. state that they control the largest and most modern electric furnace works in the world, having a maximum of 42,000 hp., and that they own and operate their own ore mines.

In connection with the use of ferro-alloy the thermal treatment of steels is, of course, of the greatest possible importance, but this will not be referred to in the following notes, which are restricted to a summary of the applications of the various ferro-alloys:

Ferro-Chrome. It is used extensively for armor plates, armor-piercing projectiles, wire, bullet-proof steel, [illegible] high speed steel, high-grade castings, stamper shoes and dies, safe steel, tyres and axles, springs, [illegible] for many other purposes. As an instance of what may be accomplished with chrome-steel it is mentioned that it is possible to make armor piercing chrome-steel projectiles of

(-inch calibre, able to penetrate compound armor plates 10½ inches in thickness, thereby remaining practically unaltered in form, while projectiles 13.5 inches in diameter, fired from the 63-ton B. L. gun, were able to penetrate over 3 feet of steel or wrought iron. T. E. Vickers has formerly pointed out the difficulty of getting ferro-chrome containing a small amount of carbon. But this difficulty has been overcome in recent years. (Mainly a result of electric furnace processes, by which it is possible first to reduce the amount of carbon in the ferro-alloy and second to make a high-percentage alloy; the higher the percentage the less, of course, the impurities introduced into the steel with the alloy.)

According to L. Guillet the chrome-steels may be divided into three classes:

(1) Steels containing from 0.0 to 7.0 per cent Cr, are similar to ordinary carbon steels, except that the breaking strain rises with the percentage of Cr.

(2) Steels containing 7 to 22 per cent Cr, that the breaking strain and elastic limits are very high, and the elongation and contraction low.

(3) Steels containing over 22 per cent Cr, that the breaking strain is medium, the elastic limit relatively low, the elongation and contraction relatively high.

The shock tests made from Frémont's method, and the hardness test (Brinell) confirm the division into three classes. The 7 per cent and under samples, being hard, and resisting impact; those with 7 to 22 per cent Cr, yielding medium results in both cases, and those containing more than 22 per cent Cr, being highly brittle. The high percentage of Cr samples, notwithstanding their brittleness, give high elongation and contraction results.

Concerning the magnetic properties it may be mentioned that the coercive force is great, especially when the material is hard. In this state Hopkinson considers that it should make good permanent magnets—a fact confirmed by Bottomley's tests.

Ferro-chromium can now be made, containing some 60 to 68 per cent of chromium and small amounts of carbon. The following is an analysis given by Geo. G. Blackwell's Sons & Co. for their most refined ferro-chrome: 63.586 Cr, 35.576 Fe, 0.65 C, 0.14 Si, 0.028 S, 0.1 P. They furnish ferro-chrome in five quantities: *A* ordinary quality, about 9 per cent C; *B* quality, 6 to 8 per cent maximum C; *C* quality, 4 to 6 per cent maximum C; *D* quality ("No. 2 refined"), 1 to 2 per cent maximum C; *E* quality ("No. 1 refined"), ½ to 1 per cent maximum C.

Chrome-Nickel-Iron Alloy.—The analysis of a sample is 52 Cr, 18 Ni, 28.96 Fe, 0.50 C, 0.50 Si, 0.03 S, 0.01 P. This alloy is used for special work, chrome-nickel steel being principally used for armor plates.

Ferro-Tungsten. Tungsten largely increases the strength of steel. The hardness of tungsten steel is not impaired by heat, and as a consequence may be driven much faster than carbon steel when used as machine cutting tools.

A very distinctive characteristic of tungsten steel is its ability to retain magnetism.

Tungsten is more generally used for manufacture of self-hardening and high-speed steel for tools.

The question of the percentage of carbon is undoubtedly of great importance, and whereas it was customary in the old self-hardening steels to have high carbon contents, it has now been found that for the best high-speed steels a low percentage of carbon undoubtedly gives the best results. To get uniformity of results it is necessary that the content of carbon in the ferro-tungsten should be exactly known. Geo. G. Blackwell, Sons & Co. supply ferro-tungsten in three qualities, the figures in parenthesis giving the analyses of three samples: No. 3 (65.7 W, 31.00 Fe, 2.92 C), No. 2 refined (75.25 W, 22.097 Fe, 1.62 C), No. 1 refined (85.79 W, 13.50 Fe, 0.60 C, W and Fe together 99.29 per cent.)

Tungsten-Nickel Alloy.—Two grades, one about 75 per cent W, 25 per cent Ni, the other 50 W and 50 Ni (½ to 1 C and 0.25 to 0.50 Si) are used in works in Continental Europe in the construction of gun-carriage parts.

Ferro-Molybdenum.—With respect to forgings, molybdenum increases the elongation and elastic limit of steel very considerably, and is stated to be much more valuable than nickel in this respect. It has been found that by the addition of 0.25 per cent of molybdenum the elongation has been increased from 4 per cent up to even 45 per cent. It will be seen, then, that for all classes of forgings—particularly large crank and propeller shafts, etc.—this metal is of the greatest value. For large guns, rifle barrels, etc., where great strength and resistance to the erosion from the use of nitrate powders is required, molybdenum steel is invariable. There is also a future for molybdenum steels for shells. An exceedingly hard and tough armor-piercing shell can be made from this alloy. "Molybdenum-chromium shells are certain to displace chromium-nickel shells."

For wire-drawing the large increase in elongation at a comparatively small cost is very important. "Very high tensile tests can be obtained."

For plates for high-pressure boilers a steel containing 0.25 per cent molybdenum with chromium "is certain to give splendid results," especially for boilers for torpedo boats, where the maximum strength and elastic limit is required. Lighter plates could be used combined with greater strength.

Molybdenum steel is specially adapted for motor-car forgings and castings. It is also claimed that molybdenum steel is more suitable than tungsten for permanent magnets.

Finally with respect to the important subject of high-speed tool-steels it is stated that molybdenum, suitably alloyed, produces high-speed steels infinitely superior to the best tungsten-chromium steels. In support of this statement a paragraph is reproduced from Mr. J. M. Gledhill's valuable paper on "High-Speed Steel," read before the Iron and Steel Institute meeting in New York in 1904, and also the remarks made by Prof. J. W. Richards in discussing this paper. Mr. Gledhill says: "The influence of this element (molybdenum) at the present time is under investigation, and our experiments with it so far produced excellent results. It is found that where a large percentage of tungsten is necessary to make a good rapid steel, a considerably less percentage of molybdenum will suffice. A peculiarity of these molybdenum steels is that in order to obtain the greatest efficiency they do not require such a high temperature in hardening as do the tungsten steels, and if the temperature is increased above 1,000° C. the tools are inferior and the life shortened."

Prof. J. W. Richards said: "The mention of the use of molybdenum brings to my mind the difference between the practice in Europe and America. On looking over that very fine work on steel, recently written by Harbord, the word, molybdenum, is not to be found in the book, and the statement just made by Dr. Matthews that several hundred tons of molybdenum have been used in America would tend to show the difference in the practice in the two countries. Mr. Pye-Smith says that high-speed tool steel is very hard and brittle, and in Mr. Gledhill's paper the influence of the carbon in increasing brittleness is emphasized, it being pointed out that from 0.4 to 0.9 per cent is the most advantageous quantity of carbon to be used if brittleness is to be avoided. The particular advantage of using molybdenum is that it gives all the desirable qualities of high-speed steel with carbon kept below 1 per cent. So that the desirable qualities are obtained by the use of molybdenum and quite low in carbon, thus avoiding the hardness and brittleness caused by the higher carbon used in high-speed steels having no molybdenum—the ordinary high-speed tungsten steels."

Geo. G. Blackwell, Sons & Co. supply ferro-molybdenum in three grades, the analyses of three samples being added.

	Mo	Fe	C	Si	S	P
Ordinary 80 to 85% grade	85.80	10.963	3.070	0.110	0.050	0.007
Refined 80 to 85% grade	85.200	14.047	0.450	0.252	0.031	0.020
Refined 50% grade..	50.311	48.920	0.350	0.300	0.030	0.020

Molybdenum-Nickel.—Nickel, alloyed with molybdenum, renders the alloy very readily fusible, and permits of the manufacture of perfectly homogeneous steel. A small addition of a 75 per cent Mo, 25 per cent Ni alloy to high-percentage nickel-steel boiler tubes has a remarkable effect. This alloy is extensively used in Continental Europe for this purpose. Small additions of molybdenum are recommended to all high-percentage nickel steels.

Ferro-Vanadium.—The following analysis of a sample of ferro-vanadium is given: 50.50 V, 45.151 Fe, 4.07 C, 0.09 Si, 0.08 Mn, 0.07 Cu, 0.019 S, 0.020 P.

Ferro-Titanium.—Titanium introduced into steel increases the ductility, even in steels high in carbon. This increase is very much marked, even when only a small quantity of titanium is added.

It is suggested also that titanium acts in a similar manner to manganese, removing oxygen from steel in the Bessemer, open-hearth and crucible processes; and, as titanium also combines with nitrogen, it will probably be found to be a very useful deoxidizer.

A great deal has been written about the use of titanium in iron foundry practice. It is claimed that titanium gives a greater density to the iron, and increases the transverse strength considerably, and also gives a harder chill.

The following analysis of a ferro-titanium sample of Geo. G. Blackwell, Sons & Co. is given: 53.00 Ti, 41.87 Fe, 3.28 C, 0.30 Al, 0.29 Mg, 1.21 Si, 0.03 S, 0.02 P. (Concerning ferro-titanium and ferro-vanadium, see also the articles by Rossi in our Vol. I., p. 523, and Hans Goldschmidt in our Vol. III., pp. 168 and 226.)

Ferro-Nickel.—This is supplied in grades of 25, 35, 50, 75 and 85 per cent. Outside the nickel and iron, the impurities are 0.50 to 1.00 per cent C, 0.20 to 0.30 Si, 0.01 to 0.02 S, 0.02 to 0.03 P. The alloy is "in such a form that even very small percentages of nickel can be readily introduced into the steel or iron, the nickel becoming thoroughly mixed through the mass, the ingots being perfectly homogeneous." The alloy is malleable and homogeneous, and can be rolled, drawn or worked readily.

This alloy is also largely used in brass and bronze, to add nickel and iron to increase tensile strength, etc.

Ferro-Silicon.—Silicon pig-iron has been used for many years in steel practice and in the iron foundry. These irons contain from 9 to 20 per cent Si. They are now being replaced by electric-furnace ferro-solicon. Their advantages were pointed out in our Vol. II., p. 122, together with applications.

Geo. G. Blackwell, Sons & Co. give the following analysis of samples of their four commercial grades:

	25 to 35%	50%	75%	95%
Si	32.70	48.70	75.80	94.80
Fe	65.50	50.84	23.97	4.99
Al	0.13	0.17	0.08	0.10
Mn	0.31	0.13	0.11	0.08
S	0.04	0.03	0.02	0.02
P	0.05	0.04	0.02	0.01
C	0.27	0.09	0.00	0.00

Ferro-Manganese.—Manganese has occupied a very prominent position in metallurgy for many years past, and its value is largely due to its great affinity for oxygen. Mr. Hadfield is responsible for the discovery of the remarkable steel known as "Hadfield's Manganese Steel"—steel containing from 7 per cent to 20 per cent of manganese, combining great strength with toughness and hardness. Carbon steels are hardened by heating and quenching water; but these manganese steels are somewhat softened and deadened by heating and rapidly cooling in water. These steels are largely used for special castings, such as crusher jaws, mining and [illegible] machinery, tramway points and crossings, etc.

The use of manganese in foundry practice is a subject which is full of interest. Small quantities of ferro-manganese are used for softening, strengthening and purifying hard or chilling irons. The car wheel makers find manganese specially beneficial, and find it most convenient to use the ferro-manganese in a powdered state. Its action, when added in this condition in the ladle, is very powerful. It permits of the remelting of a large proportion of scrap car wheels and poor grades of pig. About 1 pound of powdered ferro-manganese to 300 pounds of metal is used, and should be thrown into the bottom of the ladle, and the molten iron immediately tapped on to it. In making chilled castings, the use of powdered ferro-manganese permits of the depth of chill being regulated. To increase the chill add slightly less ferro-manganese; to reduce the chill add slightly more.

Generally speaking, the use of ferro-manganese in a foundry acts:

As a deoxidizer, reducing the chance of blow-holes.

Assists in eliminating sulphur.

Regulates the depth of chill in chilled castings.

In small proportions softens hard or chilling irons.

In larger proportions (about 1 per cent) strengthens soft irons.

Generally improves and toughens all irons.

(An excellent article on the uses of manganese, by Dr. Hans Goldschmidt, was published in our Vol. II., p. 146. This article also contains interesting notes on the uses of chromium and on some special alloys.)

Manganese-Silicon Alloys.—Large quantities of silico-spiegel, or alloys of manganese and silicon, are being used by steel manufacturers for deoxidizing, and for certain classes of work—particularly steel foundry practice—manganese alloyed with silicon gives better results than manganese alone. Up to a short time ago the only silico-spiegels available were those made in the blast furnace, containing about 10 per cent silicon and 18 to 20 per cent manganese. The developments of the electric furnace, however, have resulted in the production of very high percentage silico-spiegels, which are much purer than those made in the blast furnace.

The same conditions apply to the use of these high per centage silico-spiegels as to the use of the high percentage ferro-silicons, and it is expected that these electric silico-spiegels will in time displace the lower grades made in the blast furnace.

These manganese-silicon alloys are supplied by Geo. G. Blackwell, Sons & Co. in the following five grades: 75 per cent Mn and 25 Si, 70 to 75 Mn and 25 to 20 Si, 25 Si and 55 Mn, 20 to 22 Si and 50 to 52 Mn, 20 to 22 Si and 36 to 40 Mn.

Ferro-Phosphorus.—Some two or three years ago, Messrs Geo. G. Blackwell, Sons & Co. were asked by some large steel manufacturers in the United States whether they could produce a quantity of ferro-phosphorus, to contain as high a percentage of phosphorus as possible. After considerable experiments they succeeded in producing this alloy, and have since delivered several thousands of tons to the steel works in America and elsewhere.

This ferro-phosphorus is useful in the basic steel process for enriching the slag. It has been found that in steel for sheet purposes, particularly tin plates, steel with phosphorus rolls with a better finish.

"It seems strange that phosphorus should actually be used in steel manufacture, whereas it was always previously considered a most objectionable element, but it is a fact that the presence of phosphorus in many steels is actually desirable, and it is only lately that these properties are becoming recognized.

It has been noted that some rails high in phosphorus have been remarkable for their long life. It has also been found

that under certain conditions carbon and phosphorus may replace each other, and that by lowering the carbon and raising the phosphorus the steel rolls with a better finish, and that the finished product from the rolls has a desirable surface (or skin) which is not found in lower-phosphorus or higher-carbon steel.

Geo. G. Blackwell, Sons & Co. have produced ferro-phosphorus in two grades: 10 to 20 per cent phosphorus and 20 to 25 per cent, of which the following analyses are given:

	P	Fe	Si	C	S	Mn
No. 1.....	24.00	73.30	2.47	0.03	0.08	0.10
No. 2.....	17.5	76.20	0.42	0.27	...	5.75

Besides notes on ferro-alloys, the booklet of Blackwell, Sons & Co. contains some information on the use of copper-silicon and manganese-copper, and on the employment of fluorspar in open-hearth steel works.

Notes.

Vanadium.—We have received from the Vanadium Alloys Co. samples of the vanadate of iron made by this company in their Colorado plant, which was described in our May issue, page 195.

General Chemistry.—Prof. W. Ostwald's six lectures, held at Columbia University in the beginning of this year, on the historical development of general chemistry, are now being printed in full in the *School of Mines Quarterly*. The January issue contains the first two lectures, the first dealing with elements and compounds, the second with combining weights and atoms.

Kryptol.—The Kryptol Stock Co. has sold out its whole business to the Machine Co. of the North German-Lloyd, in Bremen.

Chemical Control of Cane Sugar Factories.—W. D. Horne gives in the January issue of the *School of Mines Quarterly* notes on a uniform method of chemical control of cane sugar factories working under diverse conditions. These notes refer to standardizing apparatus, measurements, sampling and analyses.

Regeneration of Air.—In *Glueckauf* of May 12, M. Bamberger and F. Boeck discuss the chemistry of the use of potassium-sodium peroxide for regenerating air in closed spaces, and especially for use in mines. The article is mainly of a polemical nature replying to a recent article of Michaelis. The subject is of interest with respect to the use of "oxone"; *i. e.*, fused sodium peroxide, for the same purpose, as manufactured by the Roessler & Hasslacher Chemical Co., of New York. Some interesting notes on the use of oxone for this purpose were given in a lecture of Mr. FitzGerald, on page 160 of our May issue, while a more detailed account of extended experiments in this line, made by Messrs. Brindley and von Foregger, is given in their American Electrochemical Society paper published elsewhere in this issue.

Electric Testing Laboratories.—The purpose and equipment of the Electric Testing Laboratories of New York City—well known to members of the American Electrochemical Society from the most enjoyable meeting held there last year by the New York Section—was the subject of an interesting paper presented last November by Dr. Clayton H. Sharp before the American Institute of Electrical Engineers. This paper has recently been issued in form of a neat pamphlet, with numerous illustrations. In this connection it is interesting to note that Mr. N. D. Macdonald, of the Electrical Testing Laboratories, is devoting his time this month between the Locke Insulator Works, Victor, N. Y., and the Thomas Insulator Works, Lisbon, Ohio, inspecting and testing high-tension insulators for several transmission plants in the West.

Ithaca Meeting of the American Association for Advancement of Science and of the American Chemical Society.—The meeting will be held from June 20 to July 3. Some sections intend to make this essentially a field meeting, and the local committee has arranged excursions accordingly. For the American Chemical Society visits to various points of interest in the buildings and upon the grounds of Cornell University, in charge of several committees, have been arranged for the afternoon of Thursday, June 28. On Friday, June 29, the members of the Chemical Society will take a boat ride on Cayuga Lake, with dinner at Sheldrake. This excursion will take place after the dedication of the new Physical Laboratory (Rockefeller Hall).

American Foundrymen's Association.—The American Foundrymen's Association and the Associated Foundry Foremen will meet for their annual convention in Cleveland on June 4 to 7. A large attendance is expected. An excellent programme has been prepared; among the papers to be presented are the following: A. E. Outerbridge, Jr., on the beneficial effects of adding high-grade ferro-silicon to cast-iron; F. L. Antisell, on electricity in the foundry; W. M. Carr, on developments in the thermit process in foundry practice; Dr. R. Moldenke (the distinguished Secretary of the Association), on foundry tests of coke made by the United States fuel testing plant; C. Vickers, on alloys; R. H. West, on air-furnace construction and operation; C. H. Benjamin, on tests of cast iron. The entertainment programme includes a theater party on June 5, and automobile ride for the ladies on the next afternoon, while the gentlemen visit at the same time some factories in Cleveland; a visit to one of Cleveland's pleasure resorts the same evening; a steamboat ride to Loram, with a visit to the ore docks, blast furnaces and foundry on June 7, and a smoker on the evening of June 7.

Chemical Engineering.—A recent issue of *The Technical World Magazine* contains a suggestive interview with Mr. Edward Gudeman, the well-known consulting chemical engineer of Chicago, on the ever-widening opportunities in the comparatively new field of chemical engineering. "The chemist can give the promoter the formula for bringing about the chemical action he needs in order to produce his article. The engineer can give him the plans required for the machinery. The engineer cannot tell what formula is needed. The chemist cannot give him the plans for the machinery. The engineer with chemical knowledge not only knows how to deal with the elements necessary, but he also knows how to design the machinery needed to utilize the elements and direct the forces. It might be said, and is said, that the chemist can be secured to do his part in solving the joint problem, and that the engineer can be found to do his. This is true; but we assert that the most satisfactory work will be produced by the man who can take the problem in its entirety and handle all of it." Specific examples are given from railway practice, the iron and steel industry, etc. Mr. Gudeman urges that a thorough knowledge of chemistry should be a part of the average engineer's training. He finds that one reason for Germany's supremacy in some commercial and manufacturing lines is the willingness and readiness of its manufacturers to utilize chemical discoveries.

Utilization of Blast Furnace Gases.—Many of the steel mills, iron works and other furnace plants are making great progress in the utilization of blast furnace gas. This development (which we have carefully covered in our columns) has chiefly become possible because within the last few years the gas engine business has made enormous strides. A gas engine unit of 100 hp. was, about five years ago, a curiosity, while to-day such engines are manufactured with a capacity that runs as high as 5,000 hp. Gas engines have found a great field of utility in mills, mines and factories where their introduction offers many points of advantage. It saves much space, because boiler room is eliminated. Its operation is more uniform and less costly than the steam engine. The manufacture of pro-

ducer gas, which has also reached a state of high development recently, is another feature which makes the use of the gas engine desirable. In the mills, however, the blast furnace gas, which would otherwise not be fully utilized, is now used for the operation of gas engines, which furnish power in various forms throughout the mills. The Lackawanna Steel Co. was the enterprising pioneer in this field. More recently the United States Steel Corporation began the operation of gas engines with producer gas, and that company has a number in use. In the rail mill at Braddock, Pa., gas engines run several immense electric generators, which furnish power to the rolling mills and other machinery, and these gas engines are operated by blast furnace gas. In addition, the American Steel & Wire Co., which is an underlying company of the United States Steel Corporation, is now equipping the plant at Worcester, Mass., with an 800-hp. gas engine to operate the mill machinery. All of these gas engines are manufactured by the Westinghouse Machine Co., of East Pittsburg, Pa., which firm makes a specialty of their production, and produces gas engines as large as 5,000 hp.

Gas Power.—The important place which the large gas engine has won for itself in America is plainly evidenced in the attitude of the foremost engine builders in the country toward it. The Allis-Chalmers Co., of Milwaukee, has undertaken the production of large gas engine units as well, and has developed within the past year a distinctive Allis-Chalmers type of gas engine. The latest evidence of the growing favor toward large gas engines is the recent purchase by the Carnegie Steel Co., for its Homestead Mills, of four gas-driven blowing-engine units, each consisting of a horizontal twin tandem, double-acting Allis-Chalmers gas engine of 3,000 hp., having two direct blowing tubs with a capacity to deliver 30,000 cubic feet of free air per minute against a pressure of 18 pounds per square inch. These blowing tubs, which are so designed and proportioned that they can be operated at any pressure up to 30 pounds, are of the slick type, covered by patents owned by the Allis-Chalmers Co. The Carnegie Co., in addition to the four units above described, has ordered an electrical generating unit for installation in the same mills, consisting of a horizontal twin tandem Allis-Chalmers gas engine, direct connected to a 2,000-kw, 25-cycle, three-phase generator, wound for 6,600 volts. The engine ends of these five units are to be exact duplicates with the exception of the fly-wheels, which, in the case of the blowing engines, where the demands for close regulation are not so severe, will be somewhat lighter. The generator for this unit will be built after the well-known designs of Allis-Chalmers revolving field machines, at the company's electrical works, Cincinnati, Ohio. This order followed shortly that of the Illinois Steel Co., of Chicago, calling for two 2,000-kw. Allis-Chalmers gas engine generating units of similar design.

Niagara Bill.—While we are going to press it is reported from Washington that, without debate or division, the House of Representatives passed the Burton bill for the preservation of the scenic beauty of Niagara Falls, as reported from the committee on rivers and harbors. The measure provides that the Secretary of War shall be authorized to grant permits for the diversion of water in the United States from the Niagara River to companies, individuals and corporations which are now actually producing power from the river; also for the transmission of power from Canada to companies legally authorized therefor, but only to the amount now actually in use. The Secretary of War, however, may grant revocable permits from time to time for the diversion of additional water to such amount as, in connection with the amount diverted from the Canadian side, shall not injure or interfere with the navigability of the river and its integrity and proper volumes as a boundary or the scenic grandeur of the falls. The amount of power which the bill permits to be transmitted to the United States from Canada is fixed at 160,000 hp., provided that the Secretary of War may issue [illegible] for the transmission of additional electrical power so generated in Canada, but in no event shall the total amount together with what may be generated and used in Canada exceed 350,000 hp. This amount is fixed as a limitation, and it is not to be construed as directing the Secretary of War to issue permits to this extent. Violation of the law is declared to be a misdemeanor, and the penalty $500 to $2,500 fine, or imprisonment, in case of a natural person, not exceeding one year, or both in the discretion of the court. The act is to remain in effect for three years, and the President is requested to open negotiations with Great Britain for the purpose of effectually providing by treaty for regulation and control of the waters so as to preserve the scenic beauty of the falls and the rapids to the river. The bill will now go to the Senate.

Personal.

Mr. HERBERT HAAS, consulting metallurgical engineer, with offices formerly at 604 Montgomery Street, San Francisco, informs us that since his office became an entire loss in the disastrous conflagration which befell San Francisco, he has now temporary offices with the Union Iron Works Co., at Patrero, San Francisco.

Dr. OSKAR NAGEL, who is well-known to our readers by his excellent and authoritative articles on gas power, is now connected with the De La Vergne Machine Co. as special engineer of their rapidly extending gas power department. This company builds the celebrated Koerting gas engine.

Prof. WILHELM OSTWALD has resigned the professorship of chemistry at the University of Leipzig. According to a rather mysterious cablegram, published in daily newspapers, this step is "a result of his displeasure at the lack of support accorded his chemical researches." Dr. Ostwald will establish a private laboratory. Prof. Ostwald was the first German professor selected to come to this country under the arrangement for the mutual exchange of professors between American and German universities. He delivered, last winter, a series of lectures at Harvard and later a shorter series at Columbia. The very enjoyable reception tendered to him by the New York Section of the American Electrochemical Society just before his departure for Europe is certainly still fresh in the memory of many of our readers.

Mr. EDGAR A. ASHCROFT sailed for Norway on May 23 on the steamship "Helig Olaf." He was accompanied by Mr. I. J. Moltkehansen, who will assist him in starting the Norwegian plant for the electrolytic production of metallic sodium by the process described by Mr. Ashcroft in his Electrochemical Society paper published elsewhere in this issue. Mr. Ashcroft intends to come again to this country in autumn, and will attend the meeting of the Electrochemical Society in New York in October.

Dr. HENRY NOEL POTTER, special engineer of Mr. George Westinghouse, sailed for Europe on May 30 on the "New Amsterdam." Dr. Potter expects to return in the beginning of August.

Dr. PAUL HEROULT sailed with his family for France on May 31 on the steamer "La Provence." He will return to this country in about two months, since the developments of his electric furnace process in the iron and steel industries are rapidly making progress. His representatives in this field are Mr. R. H. Wolff for the United States, and Mr. R. Turnbull for Canada. Mr. Wolff has been for many years a maker of high grade steels and wires in this country, and was during the last five years the foreign representative of the Crucible Steel Co. of America. Mr. Turnbull is intimately acquainted with the Heroult process, having been connected as a special

engineer with Dr. Héroult abroad as well as at the recent successful experiments at Sault Ste. Marie (see our April issue, p. 124).

At the Celebration of the seventy-fifth anniversary of the Royal Prussian Institute of Technology, of Hanover, Germany, the honorary degree of Doctor of Engineering was conferred on Mr. Ernst Koerting, the noted European engineer, and of the well-known firm of Gebr. Koerting, A. G. Koertingsdorf, Hanover, for his scientific researches and discoveries in gas engines and other important branches of engineering. Dr. Koerting lives in Pegli, Italy. He is interested in a number of large enterprises in this country, among them the De La Vergne Machine Co., of New York, as well as the Schutte-Koerting Co., of Philadelphia, and is at present sojourning in this country.

Mr. F. W. Haskell, the well-known president of the Carborundum Co., of Niagara Falls, has contributed an interesting and sarcastic article on "Hysterical Insurance Reform" to the May issue of *Moody's Magazine*. It will be remembered that Mr. Haskell was a member of the Frick committee investigating the Equitable Life.

Digest of U. S. Patents.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

Discharges Through Gases. Ozone.

(*Continued from page 204.*)

No. 427,744, May 13, 1890, T. F. Colin, Pittsburg, Pa.

Produces chlorine derivatives of methane by burning a mixture of chlorine and natural gas in suitable proportions. To produce chlormethane, burns a mixture of equal volumes of chlorine and marsh gas, passes the products through washing-jars or coke towers, wherein the hydrogen chloride is absorbed, thence through a water-cooled condenser, wherein higher methyl chlorides are liquefied, and finally passes the residual chlormethane through an alkaline solution, to remove traces of hydrogen chloride, through sulfuric acid to dry, and compresses at from 3 to 5 atmospheres in iron cylinders. For dichlormethane, burns a mixture of chlorine 2 volumes and marsh gas 1. Chloroform is produced from a mixture of chlorine 3 parts and marsh gas 1, passed through an externally-heated combustion chamber, through which a high-tension electric discharge is passed to induce and maintain combustion. The product, after the hydrogen chloride is washed out, is condensed and rectified by distillation, the fraction coming over between 61° and 62° C. only being collected.

No. 430,387, June 17, 1890, J. C. Kennedy, Detroit, Mich.

Places disc electrodes with opposed points on the ends of the conductors of a Holtz static generator. One conductor is hollow, and the ozone is withdrawn through it by a suction pump, and delivered, under compression, to a reservoir.

No. 468,326, Feb. 9, 1892, Francois Broyer and Paul Petit, Tournus, France.

Ages alcoholic liquors by ozone. The ozonizer consists of a glass tube surrounding and inclosed, respectively, by the electrodes, two aluminium spirals. The whole is inclosed in a tubular glass air conduit. Oxygen from a holder is dried by calcium chloride, passed through three ozonizers in series, and thence in series through three wooden tanks containing the liquor. Each tank receives the ozone through a glass tube, so arranged that the escaping ozone produces a gyratory motion of the liquor. The spent gas may be directly returned to the oxygen holder, or reclaim, reozonized and passed through other tanks before return.

No. 470,425, March 8, 1892, Frederick M. Grunbacher, New York, N. Y.

Ozonizes air for use in theaters. Forces the air through a chamber containing drying material, thence through a number of Siemens ozone tubes, and finally through an essential or volatile oil held in the bend of a U-tube. The pungent odor of the ozone is thereby removed without impairing its action.

No. 481,676, Aug. 30, 1892, C. C. Sharp, Chicago, Ill.

Ozonizer consists of an oval glass bulb, having a flaring mouth at one end. The bulb is partly lined with a ring of metal, serrated at its edge and constituting one electrode. Within the bulb and spaced away from its metal lining is a hemispherical metal cup, marginally serrated and having inwardly-projecting points. A patient to be treated is seated on an insulated stool, having a metal platform connected to a Wimshurst static generator, and inhales the ozone, the terminals of the ozonizer being connected to the same generator.

No. 483,745, Oct. 4, 1892, James W. Moliere, San Francisco, Cal.

Utilizes a Toeppler-Holtz static generator for therapeutic purposes, by providing its glass case with flexible ozone inhalation tubes. A wire connected to one terminal of the machine leads through one tube and into its glass mouthpiece, to convey the electric discharge to the patient, seated on an insulated chair.

No. 487,390, Dec. 6, 1892, Oscar Frolich, Berlin, Germany.

Locates both electrodes on one side of the dielectric layer, a third insulated conducting body being placed near the opposite side. In one form the dielectric is a flat plate, having on one side spaced rectangular electrode coatings separated by a straight insulating rib. A metal plate is placed on the opposite side of the dielectric, being spaced therefrom to provide a passage through which the air to be ozonized is passed. Illustrates three modifications. In one the electrodes are ring-coatings on the outside of a dielectric tube and separated by an annular rib, a tubular metal body being arranged within and spaced away from the dielectric tube. Another form is similar, but the electrodes are arranged longitudinally outside the dielectric tube and separated by longitudinal ribs. In the last modification two separate electrodes, each consisting of a short dielectric tube with metal coating, the two supported at their ends in ebonite rings, surround but are spaced away from a tubular chamber coated with tin, through which water is passed for cooling. The dielectric material may be porcelain, earthenware, gelatine, enamel, wood, paper with insulation, papiermache, celluloid, guttapercha, caoutchouc or ebonite, and may be protected by a mixture of paraffin and wax.

No. 494,199, March 28, 1895, Thurston G. Hall, Chicago, Ill.

A heated fire-brick chamber contains a loose pile of thermo-electric cells, each comprising two concentric porous sand crucibles. Between the inner and outer crucibles is a mass of iron or carbon, and within the inner crucible a conductor of opposite polarity, such as copper or zinc. The metallic electrodes in the several cells are connected by wires, and the whole constitutes a battery, the current of which may be externally utilized. Gases, such as air or steam, are said to be subjected to an electrochemical or catalytic action by passing them through the heated chamber containing the battery.

No. 494,200, March 28, 1893, T. G. Hall, Chicago, Ill.

Utilizes the thermo-electric battery of the preceding patent for the production of illuminating gas, by passing a mixture of steam and products of combustion through the fire-brick battery chamber. The elements of the gaseous mixture thereby "separate or become separated into a nascent condition, and will then immediately recombine into a permanent inflammable compound gas."

No. 499,572, June 13, 1893, Marie P. Oudin, Paris, France.

Into one end of a flattened glass tube is sealed a binding post, in contact with hydrogen or partially-exhausted air therein, constituting one electrode. At the sides of the tube are metallic plate electrodes. Air rises between the plates and tube and the ozonized product escapes at the top through an inhaling tube.

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Announcement.

With our present issue, *The Iron and Steel Magazine*, for the past eight years so ably conducted by Dr. Albert Sauveur, of Harvard University, becomes part of ELECTROCHEMICAL AND METALLURGICAL INDUSTRY. All subscribers to the former will be furnished with this journal for a period of two and one-half times their unexpired subscriptions, corresponding to the yearly subscription rates—$5.00 and $2.00, respectively. Those who were also subscribers to this journal will have their subscription extended for corresponding periods.

Greeting.

We extend editorial greetings to our new readers, former subscribers to *The Iron and Steel Magazine*. From the birth of ELECTROCHEMICAL INDUSTRY in 1902, this journal has devoted a large portion of space to the electrometallurgy of iron, steel and ferro-alloys. Since the enlargement of its title to ELECTROCHEMICAL AND METALLURGICAL INDUSTRY in 1905, we have endeavored to keep our readers abreast of current developments in the iron and steel field as reflected in the literature on the subject in all its phases. The absorption of a magazine especially devoted to this line of metallurgy, brings to us privileges and lays upon us obligations—the privilege of addressing more particularly than before the strong men of the iron and steel industry, and the obligation of living up to our responsibility to record more completely than ever before the information of most value to this class of metallurgists. While pledging our sincerity to the fulfillment of this programme we ask the cooperation of the leaders and of the rank and file of the iron and steel industry. We must get, in order to give.

This journal is devoted to chemistry and metallurgy—to their theory, engineering and practice. While absorbing *The Iron and Steel Magazine*, we intend to give no less attention than heretofore to the important broad field which we have covered in the past. The metallurgy of iron and steel—whether its theory, engineering or practice, chemical or electrical—finds its logical place within this broad field. The iron and steel metallurgist should be a chemist, an engineer and a metallurgist in general, the more he is of all these the stronger he is in his own specialty. Chemical engineering, calculations, combustion, principles of heating and reduction, fusion and casting, mechanical working and metoscopical examination, measuring temperatures and quantities of heats, chimney draft, radiation, gas producers, regenerative firing and the other subjects—all are as widely important to the iron and steel metallurgist as to the lead, copper or zinc smelter and to the manufacturing chemist. All these subjects we aim to summarize, recapitulate and expound, and to exert in this way a broadening influence on workers and students in chemistry and metallurgy.

If technical necessity can reinforce this argument for broader interest and more liberal technique, let us cite the fact that the iron and steel industry is the chief consumer of zinc (for galvanizing), of tin (for tinning), of sulphuric acid (for pickling black iron), of nickel, tungsten, vanadium, molybdenum, manganese and chromium (for special steels), besides being a large consumer of other metals and salts. To be a broad man in the iron industry alone necessitates a considerable degree of familiarity with these correlated fields. We shall aim to do our share towards this end in the columns of this journal. In conclusion, we may recall the immense progress with which American pluck in chemical and metallurgical practice is astonishing the world in our days. To be a worthy, consistent, and—if hard work can assure it—the best exponent of this advance, is our desire and determination.

The Metal Market.

The great industrial development in the United States is forcing the consumption of metal to an unprecedented extent. The heavy buying movements in pig iron, steel, copper, spelter and lead manifest in the month of June are easily explainable. The purchasing agents desire to be covered for Autumn's requirements before they hie themselves to the seashore or mountains for their vacation, and the fact that this is done shows that in their judgment consumption is going on at a faster rate than production. Especially in copper is the market indicative of reserve strength. The price of lead is phenomenal, and at the end of May the St. Louis price of lead crossed that of spelter—a fact that three years ago would have been scouted as chimerical. All this growth, pleasing and agreeable as it is, is somewhat too extravagant and too enthusiastic to be regarded as entirely satisfactory. In times of prosperity the wise man prepares for the lean years that come on the backward swing of the pendulum of trade, and while at the present it is largely a question of maximum output, the time is coming when the maximum efficiency is requisite.

Tar as a Liquid Fuel.

Within the next few years there is pretty sure to be enormous developments of the by-product coke oven and also the gas retort to the corresponding decrease in relative importance of their respective competitors, the bee-hive oven and the water gas producer. This progress will have the pleasant effect of reducing the smoke nuisance in our large centers. It will also increase the profit from soft coal and conserve the resources with which nature has so generously bestowed North America. A most important by product from these two operations will be tar. Naturally, much of this tar will find increasing consumption for roofing and tarring paper. And it is to be expected besides that the fractional distillation of tar will be practiced and the aniline dye trade will be established in the United States. These and other uses will absorb a large part of the tar produced in the future. But there is a large unlimited market for tar as a substitute for fuel oil in heating furnaces used everywhere in the mechanic and metallurgical arts. The hydrocarbon burner can be made to burn tar with modifications and by preheating the tar to a point where it is fluid. In point of fact, this is being done in some plants in America successfully. We can, therefore, conclude that a great market exists for tar at from 2 to 4 cents per gallon (it must be remembered, too, that tar has higher specific gravity than oil), as a substitute for fuel in large establishments where the necessary technical ability to burn the more difficult fuel is at hand. This concatenation of facts is likely to effect most favorably the metallurgical industry both by cheapening coke and furnishing a local supply of convenient liquid fuel at a low cost.

Slags.

In this issue we publish a description of a new apparatus for the determination of the melting points of slags by Mr. Woolsey McA. Johnson. The author describes a piece of apparatus made out of Acheson graphite, to measure what he calls the "dripping point" of slags and also their rates of reduction by coke and their flow through small apertures. He follows here the same line of work as in his determination of the reduction temperature of the oxides of zinc described in our Vol. II., page 185. It is always hard to make small-scale experiments or measurements which can be applied to regular practice. But we believe that Mr. Johnson has taken special pains to meet this requirement in his apparatus, and we hope that one of the metallurgical research laboratories of our universities, perhaps aided by a grant from the Carnegie Institution, would continue these researches which relate to a subject of utmost metallurgical importance.

Advances in Machine Shop Practice.

The revolution brought about in recent years in machine shop practice can be traced to two distinct causes, the use of high-speed tool steels—intimately connected with the new industry of ferro-alloys—and the general application of the electric motor in the machine shop. In both directions the limit has not yet been reached. The present tendency to overpower machine tools indicates the possibility of further developments of high-speed steels. In all cases the development has been towards greater directness and compactness and towards the elimination of complications. In harmony with this, the general tendency is now to drive each machine tool directly by an electric motor, so that all shafting and belting is eliminated. However, there has been one singular difficulty in this development. On the one hand, to make the very best use of high speed steels requires variable speed. On the other hand, there was not available an electric motor with suitable speed characteristics which could be built for variation of speed within the wide limits required by the best machine shop practice. This necessitated complications: either speed variation by mechanical means or a multi-wire distributing system by which a variable voltage could be delivered at the terminals of the motor and its speed thus adjusted. The first means mechanical complications, the second electrical complications. Both methods mean extra expense. How this difficulty has now been solved, is an interesting illustration of the progress of electrical engineering, and it is perhaps not superfluous to emphasize that this progress was essentially based on theoretical research. By properly studying the problems of commutation, electrical engineers have been able to develop what is needed

in machine-shop practice -a direct-current motor whose speed can be varied within wide limits. It is the new "interpole" motor to which we refer.

* * *

It is evident that the recent introduction of the auxiliary-pole motor or interpole motor—the speed of which can be changed within the widest limits—solves the problem of the machine shop in the most direct way. It is simply a motor with shunt characteristics, and its speed is varied by varying the field. Now, with the ordinary direct-current motor design large variations of the field would lead to difficulties in commutation, which would manifest themselves in sparking at the brushes and destruction of the commutator. Close analysis of the theory of commutation has, however, shown that a remedy may be found in providing auxiliary "interpoles" between the main poles. The motor is operated from an ordinary two-wire distributing system, and its speed may be varied without trouble within the widest limits required in practice. This type of motor has already become very popular in Europe. Its introduction in this country should have a far-reaching effect on machine shop practice.

The Atomic Weight in the Light of the Electronic Theory.

We have repeatedly referred in these columns to the indefatigable and magnificent work which J. J. Thomson has performed in recent years in the development of the electronic theory. Two distinct periods may be distinguished in his activity. The first period was experimental and devoted to building up our knowledge of the existence and the properties of negative electrons or, as Thomson says, corpuscles. Starting with experimental researches on cathode rays, he established the existence of corpuscles and determined their charge and mass. He further proved them to be identical when produced from different kinds of matter and under different conditions. On the basis of the facts, thus established by experiment, Thomson was then enabled to investigate theoretically by deduction what universal rôle the corpuscles play in the constitution of matter. This is the second period of his activity. His fundamental and most remarkable paper of two years ago—on which we commented at length in our April issue 1904—was devoted to the establishment of a model of the atomic structure. According to his model the atom consists of a sphere of uniform positive electrification; within this sphere a number of corpuscles or negative electrons are oscillating, arranged in a series of shells, one shell inside the other and the number of corpuscles within each shell decreasing gradually from shell to shell towards the center. Thomson showed that the stability of the structure of the atom necessitated the electrons to arrange themselves in shells, and he further showed that the properties conferred to the atom by this shell structure are analogous in many respects to those possessed in reality by the chemical elements. It is this fact which makes his model of such fundamental importance and of such timely interest.

* * *

Thomson showed especially that the properties of the atom will depend upon its atomic weight (which is proportional to the number of corpuscles in it) in a way quite [illegible] the periodic law. As a matter of fact, Thomson has [illegible] to "explain" the periodic system, to explain the [illegible] reason for the groups of the periods (each [illegible] corpuscles corresponding to a new group), to explain the variation of valence and the change from the electropositive to the electronegative character of the elements when passing from the left to the right in a horizontal row of Mendeléeff's table. As far reaching as these results were, there still was a certain deficiency in Thomson's model, as is indicated by the above statement that the atomic weight is "proportional" to the number of corpuscles in the atom. To make the model complete we need the numerical value of the constant of proportionality between atomic weight and number of corpuscles in an atom. This problem has now been solved by Thomson in a paper published in the June issue of the *Philosophical Magazine*, which paper is again remarkable for the ingenuity of the mathematical methods employed and by the extremely simple result to which Thomson is led by his analysis.

* * *

Thomson uses three very different methods for determining the number of corpuscles in an atom of an elementary substance. The first method is based on the consideration of the dispersion of light by gases, the second on the scattering of Roentgen rays by gases, the third on the absorption of beta-rays from radioactive substances when passing through matter. All of these three methods lead to the conclusion that the number of corpuscles in an atom of an elementary substance is of the same order of magnitude as the atomic weight. Two of these methods show in addition that the ratio of the number of corpuscles in an atom to the atomic weight of the element is the same for all elements. Thomson agrees that the evidence is rather indirect, and that the data are not very numerous, so that further investigation is necessary before we can reach an exact conclusion. But the evidence at present available seems to make it highly probable that the number of corpuscles or negative electrons in an atom is directly equal to the atomic weight.

* * *

This remarkable result puts the atomic weight into a new light. By giving an electrical meaning to this fundamental chemical conception, our attention is forcibly called to the fact that the electronic theory is about to reduce all theoretical chemistry to electrochemistry. This numerical result also greatly simplifies the electronic theory of the constitution of matter. We have, indeed, the simplest possible theory which may be imagined from the electronic point of view; this is indicated by the result that a hydrogen atom contains one single corpuscle. This simplicity should facilitate further research. Finally, if the whole theory proves true, it will bear out the correctness of the views of those who, like Prout, have contended that atomic weights must be whole numbers. They must be whole numbers because the atoms of two different elements differ from each other only in the number of the corpuscles constituting them. But the corpuscle itself is the same in all elements and is the fundamental thing—unless in the course of human events complications should arise which would necessitate a modification of this beautifully simple theory.

German Bunsen Society—Report of Dresden Meeting.

By H. Danneel, Ph. D.

The thirteenth annual convention of the German Bunsen Society was held in Dresden from May 20 to 24. The first session, devoted to the reading and discussion of papers, was held on Monday, May 21, and was opened by the usual graceful speeches of welcome.

Production of Nitrogen Oxides from Atmospheric Air.

A symposium of papers on this timely subject was opened by an introductory address of Prof. F. Foerster, on *the significance of the problem of the fixation of atmospheric nitrogen.*

Prof. Foerster first spoke on the chemical properties of nitrogen. Nitrogen is extremely slow in entering chemical combinations at ordinary temperatures, while oxygen combines very easily with most elements. Since oxygen may be removed from air by oxidation processes without the simultaneous formation of nitrogen oxides, it would appear that nitrogen and oxygen cannot combine with each other at ordinary temperatures. As a matter of fact, there exist a great number of stable nitrogen-oxygen compounds of which HNO_3 is technically most important.

At present we are provided by Nature with all HNO_3 used in industries and agriculture, in form of sodium nitrate and potassium nitrate. In the industries, HNO_3 is used specially for the manufacture of explosives, of aniline and other compounds of the chemistry of colors. Of greatest importance is HNO_3 for agricultural purposes, in the form of fertilizers. Nitrates of heavy metals are very unstable; they decompose by giving off NO_2. This is brown at ordinary temperature and colorless at higher temperatures, according to the formula:

$$\underset{\text{(low temperatures)}}{N_2O_4\ \text{(brown)}} \rightleftharpoons \underset{\text{(high temperatures)}}{2NO_2\ \text{(colorless)}}$$

At low temperatures the equilibrium favors the brown N_2O_4, at higher temperatures the colorless NO_2. The brown gas, when introduced into water in presence of oxygen, gives nitric acid.

The direct combination of nitrogen and oxygen has first been observed by Bunsen, who found that nitric oxide is formed by explosions of mixtures of hydrogen and oxygen gases with air. To oxidize nitrogen it is therefore sufficient to employ a sufficiently high temperature, and the assumption that there is a specific electric effect in the oxidation of nitrogen by electric discharges through air cannot be maintained.[1] The nitrogen of the air was first oxidized by means of electric sparks by Cavendish and Priestley, 1780. Such a method is now used on a commercial scale. By producing discharges between iron wires with direct current at 60 to 70 volts, the brown N_2O_4 may be easily observed.

In contradistinction to other oxidation processes, the oxidation of nitrogen does not give out heat. The flame, when started, would, therefore, be extinguished, if new electrical energy was not continually supplied. The electric discharges render the nitrogen and oxygen active by producing a high temperature.

A second method for the fixation of atmospheric nitrogen is based on the property of various metals, like cerium, boron, titanium, magnesium, calcium, etc., to combine with nitrogen already at red heat, forming nitrides. In contact with water the nitrides decompose, resulting in the formation of metallic oxides (hydroxides) and ammonia. However, it is still too difficult at present to win the metals back from the oxides.

A third very simple method for the fixation of atmospheric nitrogen has been invented by Adolf Frank and his collaborators. If nitrogen is blown into hot calcium carbide, calcium cyanamide $CaCN_2$ is formed.[2] This compound is also decomposed in contact with water and yields ammonia.

The following figures illustrate the commercial importance of the problem. The consumption of Chili saltpeter was one and one-third million tons in 1900, one and a half million tons in 1905, of which Germany imported about half a million tons. The world's production of ammonium sulphate in 1900 was one-half million tons, of which Germany produced 150,000 tons. About one-fourth of all nitrogen compounds is used for industrial purposes, three-fourths for fertilizers, the value of the latter amounting in 1900 to some $64,000,000.

Prof Walter Nernst then presented a paper *"on the equilibrium and the formation velocity of nitric oxide."* Prof. Nernst is at present endeavoring to secure, with the assistance of some of his students, a safe scientific foundation for the industrial fixation of atmospheric nitrogen. The present paper gives a concise review of what has so far been accomplished.[4]

There is still some dispute on the question whether the combination of N and O is a pure thermal effect or whether some specific electrical effect is involved. But the former view is undoubtedly the correct one. This is proven by the fact that the reaction $N_2 + O_2 = 2NO$ has been observed with explosions (see Bunsen's observations, mentioned above in Foerster's paper).

Such purely thermal reactions can be discussed by applying the simple principles of physical chemistry. Two problems must be answered: first, that of the equilibrium; second, that of the reaction velocity. The combination of the two gases proceeds only to a certain maximum concentration of NO, and for industrial purposes it is of importance to make this concentration as high as possible. On the other hand, the process would be useless for the industry if the reaction is very slow.

The equilibrium is determined by the law of mass action:

$$K = \frac{[NO]^2}{[O_2]\,[N_2]}$$

The chemical symbols in brackets represent the pressures of the gases after equilibrium has been reached. K is a constant independent of the relative quantities of the gases at the beginning of the reaction. K depends, however, on the temperature. By using as high a temperature as possible the concentration of NO in the gas mixture is increased.

For engineering purposes it is of fundamental importance to determine the most favorable temperature. It should be remembered that the cost of producing high temperatures increases at a higher rate than the temperature itself. For this reason it is important for industrial purposes to know how the concentration of NO in the gas mixture depends on the temperature.

This problem was investigated by Nernst by passing air into a platinum globe which was heated in an electric furnace to a certain temperature. At very high temperatures a globe of iridium was used instead of platinum. After equilibrium has been reached (i. e., after the concentration of NO has reached the maximum corresponding to this temperature), it is necessary to cool the gas mixture very suddenly, so that during the cooling period it has no opportunity of assuming a new condition of equilibrium. In other words, the cooling must be done so quickly that the original equilibrium (corresponding to the high temperature) "is frozen." Already at 1,500° C. the reaction velocity of the decomposition of NO is so slow that no further decomposition may be feared.

To produce this quick cooling process, Nernst passed the gas mixture from the hot globe through a capillary tube and then analyzed it. The concentration of the constituents of the gas

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[1] See also our April issue, pages 129 and 136.—Ed.

[2] See the report of the meeting at Bonn mentioned on page 122 of our April issue.—Ed.

[4] The fundamental standpoint of Prof. Nernst in this paper is the same as that of Prof. Guye in his article in our April issue, page 136 [on Prof. Nernst see corrected further in several details, while Prof. Guye's article covers a broader field, so that both papers supplement each other in a very gratifying way.—Ed.

mixture gives the value of K in the above equilibrium formula.

There is also another method applicable. Nernst used it to check the figures obtained by the first method. This second method consists in exploding a mixture of hydrogen and air (the latter in excess). During the explosion the temperature first increases very quickly, remains then constant for a short time, and then falls very quickly. By carrying out the explosion at different pressure, and by extrapolating the resulting curves to infinite reaction velocity, it is possible to calculate the pressure from the curves. The equilibrium constant K is thus obtained up to 2,500° C.

By integrating van't Hoff's equation,

$$\frac{d \ln K}{dT} = \frac{q}{RT^2}$$

and by assuming the reaction heat $q = -432{,}000$, an equation for x *is* obtained, where x is the percentage of NO in the mixture after heating. The integration constant is found experimentally to be 0.326.

The values of x, calculated from this equation, agree extremely well with the values found by measurement. In the following table a few values of $\sqrt{K} \cdot 10^5$ and x are given, calculated from the above formula:

T	$\sqrt{K} \cdot 10^5$	x
1500	2.48	0.10
1800	8.51	0.34
2000	15.3	0.61
2500	45.5	1.79
3000	93.0	3.57

By further theoretical considerations Nernst finds the following values of the time t, in which one-half of the equilibrium concentration is reached at different temperatures, T:

T	1600	1500	1900	2100	2500	2900
t	81.6 years	1.3 days	2 min.	5 sec.	0.001 sec.	0.0000345 sec.

The velocity is extremely high at high temperatures and very low at low temperatures, below 1,700°. Prof. Nernst considers it to be improbable that a gas mixture may be obtained on a commercial scale containing more than 3 per cent NO, since in the removal of the gas mixture from the arc the cooling is not quick enough to prevent completely the decomposition of NO.

In the discussion the question was asked whether the concentration of NO in the mixture in the condition of equilibrium might not be essentially increased by using, instead of air, a mixture of O_2 and N_2, containing 50 per cent of each, thus corresponding to the above equation of the reaction. Prof. Nernst replied that the output would then increase by about 20 per cent, but this would hardly cover the cost for producing such a mixture of oxygen and nitrogen. Not only the equilibrium constant increases in this case but also the reaction velocity.

Prof. F. FOERSTER then presented a paper giving a summary of the industrial processes of oxidation of nitrogen which have so far been proposed. The theory clearly indicates that two main conditions must be fulfilled: to heat the gas mixture to a very high temperature, and after equilibrium has been reached to cool it as suddenly as possible.

Muthmann and Hofer, who used the electric arc, found that the output of nitric oxide is the smaller the greater the energy which produces the arc. This is explained by Brode as follows. The electric arc consists of an interior portion, which is extremely hot, and in which the nitrogen and oxygen combine. Around this interior part of the arc there is the so-called aureola, which is at a much lower temperature, but still sufficiently hot to permit the decomposition of the NO passing off from the interior part of the arc. The greater the energy supplied to the arc the greater is the aureola, and therefore the decomposition and redestruction of NO formed in the interior part of the arc.

It would, therefore, be much better to use spark discharges, [illegible] But since the energy of a spark is small, a great many sparks are required. Haytheim used for the purpose Lodge and Lovejoy. Their apparatus gave with a high speed the passage of an some 4 or 5 per cent NO, yielding [illegible] grams of nitric acid per kw-hour. To reduce [illegible] the apparatus consumes 1 amp. per spark. If larger currents are used the sparks become surrounded with the disadvantage [illegible] use sufficiently large amounts of energy it is, therefore, necessary to employ a large number of these [illegible] Prof. Foerster thinks that this has been the reason of the commercial failure of the process.

The current of a spark discharge may be increased by using it in combination with condensers,[6] but the condensers required for this purpose have not proven sufficiently durable for commercial work. If it should become possible to make [illegible] durable condensers, the spark discharge will undoubtedly represent the ideal method for the oxidation of atmospheric nitrogen.

In the meantime it has been necessary to return to the use of the arc discharge, and the endeavor is to overcome the disadvantage sketched above by causing the arc either to change its place continually or to be broken continually. The process of Birkeland and Eyde[7] uses this method, and yields on a commercial scale 500 to 600 kilograms of anhydrous nitric acid per kw-year. The one and a half millions of Chili saltpeter, consumed per year in the world, could be produced by some 2,500,000 hp. Of these, Prof Foerster remarks, 500,000 hp. could be produced by utilizing in gas engines the "waste gases" of the blast furnaces in the iron district of Rhineland, Germany.

Prof. Foerster then discussed whether the combination of nitrogen and oxygen is really a purely thermal (no electrical) effect, and described experiments of his own which prove that this is the case. He finally pointed out that it will scarcely be possible to get a much higher efficiency than is now obtained in actual practice, if not a form of discharge is discovered which yields much higher temperatures. Any progress in the industrial fixation of atmospheric nitrogen must, therefore, come from advances in electrical engineering.

It might be possible to improve the efficiency by using a higher pressure of the gases, since this increases the conductivity of the air; but the equilibrium constant is not changed by the pressure.

A second improvement might be derived by using a mixture of 50 per cent of oxygen and 50 per cent of nitrogen, and liquid air might be made use of for this purpose.[8]

Prof M. LE BLANC then presented a paper on the *analytical estimation of NO in air*. A method for this purpose consists in passing the N_2O_4 into water and determining the quantity of nitrate and nitrite formed. The reaction is assumed to proceed according to the equation $NO_2 + HO = HNO_2 + HNO_3$; *i. e.*, equivalent quantities of nitrite and nitrate.

The author has found, however, that the ratio of nitrite to nitrate ($i:a$) is not always unity, but differs according to the conditions of the experiment. If the gas mixture is shaken with the water for a short time only, the ratio is indeed unity, but after shaking for 3 minutes it was found to be 1:1.7. If a KOH solution is used instead of pure water, more nitrate is obtained than nitrite. If the gases come from the reaction in a high-tension flame, the conditions were found to be reversed. In one case [illegible] 1.07. The deficiency is greater when the gas is is sucked off directly from the arc.

[6] See our vol. I., pages 20, 100, 289, 386, [illegible] 250; vol. III., page 285.—Ed.

[7] For instance, in the process of Kowalski [illegible] page 36; vol. II., page 152.—Ed.

[8] See our vol. II., pages 309, 507, vol. III., [illegible] 17, [illegible] 23, and 126.

[8] See our April issue, page 139.

Prof. Le Blanc discussed possible theoretical explanations of these facts.

The last paper of this symposium was presented by Prof. Klandi, who discussed *the industrial change of nitrogen oxide gases into nitric acid and nitrates.* When it is stated that on a commercial scale 238 kg. NO_2 = 600 kg. of anhydrous nitric acid is produced per kw-year (Birkeland-Eyde process in Norway), it should be remembered that the gas is obtained greatly diluted, so that the cost of concentration must not be neglected. On a small scale it is possible to get per kw-year 900 kg. HNO_3, or 200 kg. of combined nitrogen in form of a gas mixture of 5 per cent concentration. We may expect that it will be possible later on to get the same results on a large commercial scale, but probably no better results.

For some water-power plants of Norway the cost of the horse-power-year is stated to be as low as $4.00; in large water-power plants of Germany and Austria this cost is about $10.00, at Niagara Falls it is $20.00. With smaller plants the cost is still higher, so that they cannot be considered for the production of NO. Low rates are possible with producer gas plants, and especially with gas plants using the waste gases of blast furnaces.

According to the above figures, the kilogram of combined nitrogen (in form of NO_2 diluted in air) is obtained on a commercial scale in Norway for 3.75 cents, while its costs 29 cents in form of Chili saltpeter and from 42 to 48 cents in form of HNO_3, according to the concentration.

For the use of NO_2 diluted in air we may first consider the direct chemical applications of NO_2, especially in the manufacture of sulphuric acid. Well conducted plants use 7 kg. $NaNO_3$ per ton of H_2SO_4, corresponding to 3.79 kg. or 1.976 liters of NO_2. This quantity is contained in 100 cubic meters of air, containing 2 per cent NO_2. But this dilution is deleterious in the manufacture of sulphuric acid, while a concentration of 5 per cent permits direct introduction into the chamber.

The following method would be suitable: absorption of the nitrogen dioxide by sulphuric acid and introduction into the Glover tower. The author calculates that a plant producing daily 250 "metric centners" of H_2SO_4 consumes 175 kg. of saltpeter, costing about $8.31. The same quantity of combined nitrogen, in form of NO_2 diluted in air, costs only $1.09, if cheap power is available as in Norway. But even if the cost of power is four times that given above for the cheapest Norwegian water-power plants (for instance, if gas engines or oil engines are used), the above cost is only $4.38 against $8.31 for the use of Chili saltpeter. This means a saving of about $1,200 per year. Even if the cost of power is eight times the cost in Norway, or $40.00 per kw-year, the use of NO_2 gases diluted in air would not be more expensive than the use of Chili saltpeter.

The most important application is, however, for the production of fertilizers, and for this purpose the NO_2 gases, diluted in air, must undergo further treatment.

Germany's agriculture now consumes 400,000 tons of saltpeter per year, the chemical industries 100,000 tons. Under the present conditions of the industrial oxidation of atmospheric nitrogen 1,125,000 hp. would be required to fix the atmospheric nitrogen needed for Germany alone. To produce sodium nitrate from the NO_2 gases, some 25,000 carloads of caustic soda would be required per year. This would require doubling the present output in Germany.

The further treatment of the NO_2 gases is rendered very difficult by their dilution. The best method of procedure would appear to be at present to try the absorption of the gases according to the formula

$$2NO_2 + O + H_2O = 2HNO_3$$

This has so far not been possible, not even with the aid of catalysers. However, if large quantities of water are used for absorption, the reaction is

$$2NO_2 + H_2O = HNO_3 + HNO_2$$

If through this mixture of solutions we now pass NO_2, which has been brought previously to a concentration of 50 per cent, a 60 per cent HNO_3 solution is obtained.

This reaction takes place with the development of NO, which changes again to NO_2 in contact with air. Finally, a mixture of NO and NO_2 is obtained which acts on water like N_2O_3; *i. e.* yields HNO_2 when introduced into water.

If the process is carried out successively in several absorption towers, the first tower yields HNO_3 of 60 per cent without HNO_2. In the other towers one gets HNO_3 of a smaller concentration and containing HNO_2.

Nitric acid is on the market in Germany in two concentrations: 90 per cent and 60 per cent. For four-tenths of the consumption of nitric acid in Germany 60 per cent acid is suitable, and this may contain HNO_2. The more concentrated acid which is used for explosives, should not contain more than 1 per cent of HNO_2. For transportation of the acid a change into nitrates, especially Mg $(NO_3)_2$ and Ca $(NO_3)_2$, should be considered.

In view of the large difference between the cost of the kilogram of combined nitrogen in form of NO_2 diluted in air (3.75 cents), and in form of Chili saltpeter (29 cents), the manufacture of fertilizer may be profitable even if the treatment of the gases is not very cheap. To make calcium nitrate with the aid of limestone will be the cheapest. (See page 127 of our April issue.)

Even if the cost of power should be such as to render the profit per ton of output small, yet the process may be profitable on account of the enormous consumption. Due to its hygroscopic properties, it is impossible to ship calcium nitrate in bags. Farmers are also said not to like it very much. Basic calcium nitrate is reported to prove satisfactory.

At present it is still impossible to state definitely how well this new industry will pay. The financial results also depend on the situation of the plant, freight charges, etc. Further progress may certainly be expected in the engineering problems involved.

(To be Concluded.)

Why Do Fluid Slags Cause Slopping in the Bessemer Converter?

To the Editor of Electrochemical and Metallurgical Industry:

Sir:—No doubt many of your readers have, like myself, been puzzled as to why it is that pig iron rich in manganese causes serious slopping in the Bessemer converter, and the throwing out of large quantities of molten metal. The manganese should have the effect of making the slag thinner rather than thicker, and at first sight it seems that a thin slag should be less subject to slopping than a thick one. But after puzzling over this a great many different times I have at last seen an explanation, and I should be glad if any of your readers could throw light on its reasonableness.[1]

The explanation is simply this: that a thin slag, when a considerable quantity of it is momentarily submerged beneath the surface of the metal by the violent boiling, may, by breaking up into many small particles of drops, suddenly expose so large a surface to that metal that the evolution of carbonic oxide caused by the reaction between the iron oxide of the slag and the carbon of the metal, may be so violent as to cause slopping. In other words, the liquidity of the slag causes slopping by bringing about a very extended surface of exposure of the submerged oxidizing slag to the carbon of the metal in which it is submerged, with the consequent sudden formation of carbonic oxide, sudden generation of gas and consequent explosion.

Henry M. Howe.

Columbia University in the City of New York.
May 26, 1906.

[1] See also the abstract of the paper by Doelter, on page 280 of this issue. Editor.

Metallurgical Calculations.

By J. W. Richards, Ph. D.
Professor of Metallurgy in Lehigh University.

THE HEAT BALANCE SHEET OF THE BLAST FURNACE.

Twenty-eight years ago, Sir Lothian Bell first constructed a satisfactory heat balance sheet for a blast furnace. His observations were largely, and his experience altogether, confined to the reduction of the argillaceous siderite ores of the Cleveland district, England, and although he made numerous attempts to draw general conclusions from the data at hand, applicable to iron smelting in general, yet many of his deductions remain true only for the particular ores and manner of working characteristic of the Cleveland district.

No treatment of this subject, however, can be based otherwise than upon Bell's researches, following the lines laid down in his "Principles of the Manufacture of Iron and Steel."

Heat Received and Developed.

The items on this side of the balance sheet are:

(1) Combustion of carbon to carbonous oxide (CO).
(2) Combustion of carbon to carbonic oxide (CO^2).
(3) Sensible heat of the hot blast.
(4) Heat of formation of the pig iron from its constituents.
(5) Heat of formation of slag from its oxide constituents.

(1) and (2) *Combustion of carbon in the furnace.* There is but one satisfactory way to determine with exactness the amounts under this heading. From the balance sheet, the total amount of carbon passing into the gases is obtained; from the analysis of the gases, the weight of carbon per unit volume of gases is calculated; the first divided by the second gives the volume of gases per unit weight of pig iron produced. The amount of CO and CO^2 in these gases is then obtained by use of the gas analysis, and if from the total CO and CO^2 in the gases there be subtracted the CO and CO^2 contributed *as such* by the solid charges, the difference is the CO and CO^2 which have been formed in the furnace. The heat evolved in the formation of these quantities can then be calculated.

Illustration: In Problem 51[1] it was calculated that per 1,000 kilos. of pig iron produced, 534.09 kg. of carbon went into the gases; also that the analysis of the gases showed 0.20736 kg. of carbon in each cubic meter of gas. The quotient indicated, therefore, 2575.6 cubic meters of gas produced per ton of pig iron. From the analysis of the gases there was in this volume,

$$2575.6 \times 0.231 = 595.0 \text{ m}^3 \text{ of CO}$$
$$2575.6 \times 0.148 = 381.2 \text{ m}^3 \text{ of } CO^2$$

whose weights were

$$595.0 \times 1.26 = 749.7 \text{ kg. CO}$$
$$381.2 \times 1.98 = 754.8 \text{ kg. } CO^2$$

The balance sheet shows, however, 49.1 kg. of CO^2 contained in the limestone flux used, which can be assumed as entering the gases bodily. Subtracting this we have 705.7 kg. of CO^2 formed in the furnace, and 749.7 kg. of CO, containing respectively

$$705.7 \times \frac{12}{44} = 192.5 \text{ kg. of C in CO}$$

$$749.7 \times \frac{12}{28} = 321.3 \text{ kg. of C in CO}$$

The heat generated in the furnace by the oxidation of carbon is, therefore,

$$192.5 \times 8100 = 1,559,250 \text{ Calories}$$
$$321.3 \times 2430 = 780,760 \text{ "}$$
$$2,340,010 \text{ "}$$

[1] See this journal, April, 1906.

If this carbon could have been entirely burned to CO^2 there would have been generated

$$513.8 \times 8100 = 4,161,780 \text{ Calories}$$

Showing that only 56 per cent of the calorific power of the carbon was developed in the furnace, the other 44 per cent exists as potential calorific power in the waste gases, and part of it is really put back into the furnace as sensible heat in the hot blast.

There is a little doubt as to how to consider the CH^4 in the gases; that is, whether the heat of its formation should be reckoned in as developed in the furnace. This would be (C, H^4) = 22,250, or 1,854 Calories per kg. of carbon contained therein. Its presence in the gas probably results largely from the distillation of the fuel at a high temperature, and the heat required to disunite the CH^4 from the solid fuel is probably as great as is represented by its heat of formation from carbon and hydrogen. The item is, therefore, a doubtful one, and as far as we know, we may be coming about as near to the truth by omitting it altogether as by counting it in. If we wished to add it in the illustration just given the calculation would be

Volume of CH^4 = 2575.6 × 0.005 = 12.88 m.
Weight of C = 12.88 × 0.54 = 6.9 kg.
Heat of formation = 6.9 × 1,854 = 12,793 Cal.

It should be emphasized that in this calculated heat of oxidation of carbon in the furnace, no account has been taken whatever of *where* in the furnace this heat is generated. Above all, the mistake should not be made of supposing that the 780,760 Calories produced by formation of CO represents the heat generation at the region of the tuyeres; nothing could be further from the truth. A great deal of carbon is burnt to CO at the tuyeres, and some above the tuyeres, but a goodly proportion of this CO oxidizes by abstracting oxygen from the charge and becomes CO^2. It would not be incorrect, however, to divide the heat of oxidation of carbon in the furnace into two parts, viz.: to assume all the carbon as first forming CO, and part of this CO afterwards forming CO^2, corresponding to the amount of the latter formed in the furnace. If this were done, we would have

513.8 kg. C to CO = 513.8 × 2430 = 1,248,535 Cal.
449.2 kg. CO to CO^2 = 449.2 × 2430 = 1,091,475 "
2,340,010 "

This analysis gives us the information that of the total heat generated by the oxidation of carbon in the furnace, somewhere about one-half is generated by its burning to CO, and the other half by the further oxidation of CO to CO^2, and we also know that the larger part of the former takes place at the tuyeres, and all of the latter takes place during the reduction of the charges in the upper part of the furnace.

If we know, however, or have calculated the amount of the blast received by the furnace, or, more properly speaking, the amount of oxygen in the blast, then the heat generated by oxidation of carbon at the tuyeres becomes known. In the previous illustration, taken from Problem 51, we can also take from the same problem the weight of oxygen in the blast, 557.7 kilos. This would burn 557.7 × 0.75 = 418.3 kg. of carbon to CO at the tuyeres, generating there

$$418.3 \times 2430 = 1,016,469 \text{ Calories,}$$

or 44 per cent of all the heat generated by oxidation of carbon in the furnace, leaving 1,323,540 Calories as generated above the tuyeres by the agency of the oxygen of the charges. These figures tell us just where and how the principal items of heat were generated in this particular furnace, and the similar calculation may be made for any blast furnace for which we have the necessary data.

(3) *Sensible heat in the hot blast.* To calculate this item we need to know the weight or volume of the different constituents of the blast and their temperature. The question at once arises, as to what base line of temperature shall be chosen. It is most convenient to choose 0° C., since that is

not over 15° from the average temperature in the largest iron producing countries. However, any other prevailing temperature may be taken as the base line, involving merely a little more calculation, since our specific heats are reckoned from 0° C. The temperature ought, moreover, to be taken as near to the tuyeres as possible, to properly take into account the effect of cooling of the blast in the bustle and feeder pipes, from radiation and expansion. The blast consists of air proper and moisture, the former with a mean specific heat between 0° and t° C of $0.303 + 0.000027t$, in kilogram calories per cubic meter, or in ounce calories per cubic foot, the latter with a similar mean specific heat of $0.34 + 0.00015t$. Since the moisture at times amounts to as much as 5 per cent of the blast, it should be calculated separately.

Illustration: With the outside air at 30° C., and saturated with moisture (raining), calculate the heat carried into a blast furnace by blast carrying in 1859.1 kilos. of nitrogen, the temperature of the heated blast being 600° C. Barometer 720 millimeters of mercury. Temperature base line 0° C.

Solution: One cubic meter of the moist blast, as taken into the blowing cylinders, carries all the moisture it can hold, the tension of which is therefore 31.5 millimeters. The tension of the air proper present is therefore $720 - 31.5 = 688.5$ millimeters, and each cubic meter of moist air carries

$$\frac{31.5}{720} = 0.0438 \text{ cubic meter of moisture, and}$$

$$\frac{688.5}{720} = 0.9562 \text{ cubic meter of air proper.}$$

Whatever the temperature of the blast, the moisture and air proper will be in this same proportion whenever its temperature is over 30° C. If the temperature were 0° C. the moisture would be mostly condensed, but for the purposes of calculating the heat brought in we may assume the moist air to be at 0° C., with its moisture uncondensed. That volume of blast which would be 1 cubic meter at 0° and 760 mm. pressure, would, therefore, bring in, at 600° C., the following quantity of heat:

H^2O 0.0438 ×	$[0.34 + 0.00015 (600)]$	× 600 = 11.3 Calories
Air 0.9562 ×	$[0.303 + 0.000027 (600)]$	× 600 = 179.7 "
	Total	191.0 "

Since the nitrogen present in this is

$$0.9562 \times 1.293 \times \frac{10}{13} = 0.9511 \text{ kg.}$$

the heat brought in per 1859.1 kg. of nitrogen is

$$191.0 \times \frac{1859.1}{0.9511} = 373,344 \text{ Calories}$$

An amount equal to over one-third of all the heat generated by combustion of carbon at the tuyeres.

(4) *Heat of formation of pig iron from its constituents.* The pig iron contains several per cent, some 5 to 10 altogether, of carbon, silicon, manganese, phosphorus, sulphur and other elements. The energy of their combination with the iron is a somewhat indefinite quantity, and in no case can be considerable. Berthelot states the energy of combination of carbon with iron as $(Fe^3, C) = 8,460$, which would be 705 Calories per kilogram of carbon, and another investigator (Ponthiere) states the heat of combination of phosphorus with iron to be zero. In the present state of uncertainty it is hardly allowable to add in any other than the heat of combination of the carbon in the iron, and leave out that of the other elements.

(5) *Heat of formation of the slag from its constituent oxides.* Here we touch upon a quantity of more than insignificant proportions, yet which is not yet quantitatively known with satisfactory accuracy. The main constituents of the slag are SiO^2, Al^2O^3, CaO, MgO and CaS, which are provided by clay, limestone and iron sulphide. If we allow, on the other side of the balance sheet, for the heat necessary to de-hydrate clay, drive carbonic acid off carbonates, and break up iron sulphide and enough CaO to furnish Ca for CaS, we are then entitled, on the other hand, to place in the heat evolution column the heat of combination of aluminium silicate with lime and magnesia, the heat of formation of CaS and its heat of solution in the silicate slag. The heat of formation of CaS is 94,300 Calories, or 2,947 Calories per kilogram of sulphur; its heat of combination with a silicate slag is unknown. The heat of combination of lime with aluminium silicate has been determined only for the proportions 3CaO to $Al^2Si^2O^7$, that is, for 168 parts of CaO uniting with 222 parts of aluminium silicate. This has been determined in Le Chatelier's laboratory as $(3CaO, Al^2Si^2O^7) = 33,500$ Calories, which is 200 Calories per unit of CaO combining, or 150 calories per unit weight of $Al^2O^3 + SiO^2$. The calculation would be made on the basis of the amount of lime (plus lime equivalent of magnesia present), if it were present in a smaller ratio than 168 to 222 of silica and alumina, and on the basis of the silica and alumina, if their ratio to the summated lime were less than 222 to 168. It is probable that in the near future these quantities will be known more accurately.

One item of heat received by the furnace has not been mentioned, because of its usual absence, viz.: heat in hot charges. Very rarely roasted ore comes hot to the furnace, in which case its sensible heat must be counted in, else the thermal sheet of the furnace will be that much out of balance.

Heat Absorption and Disbursement.

The items on this side of the balance sheet are:

(1) Sensible heat in wase gases, including water vapor only as vapor.
(2) Sensible heat in outflowing slag.
(3) Sensible heat in outflowing pig iron.
(4) Heat conducted to the ground.
(5) Heat conducted and radiated to the air.
(6) Heat abstracted by cooling water, tuyeres, etc.
(7) Heat for de-hydrating the charges.
(8) Heat for vaporizing water from charges.
(9) Heat absorbed by decomposition of carbonates.
(10) Heat absorbed in reduction of iron oxides.
(11) Heat absorbed in reduction of other metallic oxides.
(12) Heat absorbed by decomposition of moisture of the blast.

(1) *Sensible heat in waste gases.* The amount of these gases is known only from the carbon contained in unit volume, by analysis, and the known weight of carbon entering and leaving the furnace. If there is much fine coke carried over by the blast, allowance must be made for the carbon in it, because this would not be represented in the gas analysis. The analysis of completely dried gas is that usually obtained, because if the gas is measured without drying, an uncertain amount of moisture is condensed, and, therefore, it is usual to dry before measuring and analyzing. The amount of moisture in the gases is either assumed as that driven off from the charges, as shown by the balance sheet, or else is determined directly by drawing the gases through a calcium chloride tube or other dessicating apparatus. Several tests should be made to get a fair average, because much more will be in the gases immediately after charging than immediately before. Dust must be excluded from the drying tube by filtering the gases through dry asbestos. The average temperature of the gases should be known over a considerable period, a thermo couple in the down-comer gives this most accurately and more uniformly than if inserted above the stock line in the furnace.

The weight of moisture per unit volume of dry gas is then converted into volume at standard conditions by dividing by 0.81 (1 cubic meter = 0.81 kg; 1 cubic foot = 0.81 ounce avoirdupois). The sensible heat of the gases is then calculated, using 0° C. as the base line, and the proper mean specific heats

of the gases per unit of volume. The water vapor will here be considered simply as a gas, and its sensible heat above water vapor at 0° only calculated. This leaves the latent heat of vaporization of this water to be considered as a separate item (606.5 calories), that is, as heat absorbed by reactions in the furnace, thus putting it on exactly the same footing as the CO^2 in the gases which has been expelled from carbonates in the furnace. By so proceeding much uncertainty as to the heat in the water vapor is avoided.

If the amount of the dust is considerable its quantity should be ascertained, and the heat in it also calculated and added in to the heat in the moist gases. Its specific heat may be approximated as so much carbon, iron oxide and silica, the proportions of each of these present being known.

(2) *Sensible heat in outflowing slag.* The weight of slag produced is seldom taken directly, but can be reckoned up with all needful accuracy from the balance sheet of materials entering and leaving. Its temperature and specific heat, solid and liquid, melting point and latent heat of fusion, are unfortunately almost always unknown factors. The one datum which is needful, however, is the total heat in a unit weight of liquid slag as it flows from the furnace, and this is not a difficult quantity to obtain. A rough calorimeter with a reliable thermometer and containing a carefully weighed quantity of water, may be easily constructed. Some liquid slag is run directly into it, and by observing the rise of temperature and afterwards filtering out, drying and weighing the granulated slag, a satisfactory determination can be arrived at. This is corrected to 0° C. by using an approximate specific heat, say 0.20, for the range of final calorimeter temperature to zero. In this connection it is important to note that the calorimetric determinations of Akerman on blast-furnace slags, give the heat in the *just-melted* slag whereas slags flowing out of a furnace are considerably, some 200° to 500° C., above their melting point, and therefore contain some 50 to 150 Calories more heat than that given by Akerman for a slag of similar composition. Since Akerman's values run from 350 to 400 Calories, the actual heat in the outflowing slag may be between 400 and 550 Calories. Akerman himself states that an average of twenty-seven Swedish furnaces gave 530 Calories as the actual heat in unit weight of outflowing slag, and Bell uses 550 in most of his calculations on Cleveland (England) furnaces.

(3) *Heat in outflowing pig iron.* The heat in *just-melted* pig iron is evidently too small a quantity to use in this connection. The heat in the outflowing pig iron at 200° to 500° above its melting point will be 50 to 100 Calories greater. The former quantity is about 245 calories; the latter will be 300 to 350. Bell takes 330 for Cleveland furnaces; Akerman states 250 to 325 for Swedish furnaces. We may conclude, then, to use 300 Calories for a coke furnace running cool, and up to 350 Calories for a very hot furnace.

(4) *Heat conducted to the ground.* This is a very uncertain quantity. It varies with the kind of ground, and is more nearly a constant per day than per unit of pig iron produced. It is, therefore, expressed per unit of iron produced, larger for small furnaces run slowly than for large furnaces run fast. It is less when running rich ores and greater with poor ores, other things being equal. As nearly as can be assumed we would put this item as lying between 60 and 200 Calories per unit of pig iron made. Bell uses 150 for some Cleveland furnaces, but it is certainly less than 100 in some charcoal furnaces using pure ores and fuel, and consequently with a small heat requirement.

(5) *Heat conducted and radiated by the air.* This item is likewise more nearly a constant quantity per day for a given furnace, and is therefore less per unit of iron produced the faster the furnace is run. It may vary between 60 and 250 Calories per unit of pig iron, the former in furnaces of low heat requirement per unit of iron produced, the latter in those of high heat requirement. If the amount were calculated it would [illegible] the temperature of the outside shell, that of the air, the velocity of the wind, and the total outside surface, in order to calculate by the principles of heat radiation and conduction, the amount radiated per day. No one has done this yet for any one furnace, and in most cases [illegible] using assumed figures and [illegible] iron, averaging 100 to 150 for charcoal furnaces of low heat requirement, 200 to 450 for Cleveland furnaces (Bell), and 300 to 500 for large, modern furnaces with thin walls and great height.

(6) *Heat abstracted by cooling water.* In the old-fashioned heavy masonry, cold blast furnace, this item was zero. With the advent of hot blast, the water needed for cooling the tuyeres entered as a heat abstracting factor. It is greater the harder a furnace is blown, but does not increase proportionately with the output. The heat lost by tuyere-cooling water may be 50 to 100 Calories per unit of pig iron made. That for cooling of bosh plates and the outside of the crucible in modern furnaces, may vary all the way up to 200 Calories. These two items are very large in a modern furnace, but are necessary expenditures of heat energy in order to preserve the lines of the furnace during fast running. For any particular furnace they may be determined with all needful accuracy by measuring the amount of water pumped or used for these purposes and its temperature before and after using.

(7) and (8) *Drying and [illegible]* [illegible] into the furnace as moisture and as combined water of the charges. To convert the moisture, such as is evaporated by a current of moderately warm air, into vapor requires 606.5 Calories per unit of water. This allows merely for its vaporization in the furnace, and not for any sensible heat which it may carry out of the furnace at the temperature of the waste gases. This latter item is properly considered in with the sensible heat of the waste gases. The common practice of saying that it takes 637 Calories to evaporate the moisture of the charges is wrong, because this amount would convert water at 0° to vapor at 100°, and, therefore, would include part of what is properly the sensible heat of the waste gases. On the other hand, it is equally wrong to say that this heat of vaporization should be counted in as sensible heat in the hot gases; it would be just as logical or, rather, equally illogical, to count the latent heat of vaporization of CO^2 as sensible heat in the gases.

To drive off water of hydration from hydrated minerals in the charge requires an additional amount of chemically-absorbed heat. As far as is known, this is small for the water driven off hydrated iron oxides, so small as to be a negligible quantity and safely left out, but if it comes from clay the large amount of 611 calories is absorbed in merely separating it from its chemical combination [illegible]. Al_2SiO_5 [illegible] Calories, which would require 611 × 257 = 157.0 Calories to put into the state of vapor each unit weight of water entering the furnace chemically combined in clay. This does not concern the ordinary moisture in most clay expelled at a low heat, but only the combined water in the clay. Where [illegible] occurs in the iron blast furnace, however, and [illegible] its amount explains some of the difficulties met in working clayey charges, particularly since a large part of this chemically combined water is expelled only at a red heat, and, therefore, [illegible] the lower zones of the furnace.

(9) *Decomposition of carbonates.* Raw limestone, carbonate, as the usual flux of the blast furnace, and its carbon [illegible] is evolved at temperatures between 700° and 800° C. Whether some of this is subsequently [illegible] with carbon and reduced to CO, is immaterial to the balance sheet, because more than enough CO [illegible] from the [illegible] to represent the CO^2 of the flux, and we charge the furnace only with the formation of the CO and CO^2 [illegible] found in the gases, less the CO from fluxes. Full charges [illegible]

the heat absorbed also in the assumed reaction $CO^2 + C = 2CO$, but this is an error, because it is doubtful how much of the CO^2 is thus decomposed, and the question, in its last analysis, is one of heat *distribution* in the furnace, and does not concern the totals of heat absorbed or evolved. We can, therefore, omit the item of decomposition of this CO^2 (as likewise, and for analogous reasons, the heat evolved in carbon deposition in the upper part of the furnace — $2CO = C + CO^2$), and need consider only the heat required to expel the CO^2 from carbonates. This is:

(CaO, CO^2)	= 45,150 Calories	= 1,026 Calories	per kg.	CO^2
(MgO, CO^2)	= 29,300 "	= 666 "	"	
(MnO, CO^2)	= 22,200 "	= 500 "	"	
(FeO, CO^2)	= 24,900 "	= 566 "	"	
(ZnO, CO^2)	= 15,500 "	= 352 "	"	

By using the above figures in connection with the known composition of ore and fluxes, the heat required to decompose carbonates can be correctly calculated.

(10) *Reduction of iron oxides.* The heats of formation of the various oxides of iron are:

(Fe, O)	= 65,700 Calories	= 1,173 Calories	per kg. iron
(Fe^2, O^3)	= 270,800 "	= 1,671 "	"
(Fe^2, O^4)	= 195,600 "	= 1,746 "	"

And, therefore, just these quantities of heat are required per unit weight of iron reduced from these compounds. If the ore is a carbonate the heat absorbed in driving of CO^2 from $FeCO^3$ can be first allowed for, and then the heat required for reduction of the FeO calculated on the weight of the reduced iron. If some FeO goes into the slag it will be as FeO, and if the ore was Fe^2O^3, or Fe^3O^4, the reduction of unit weight of iron from the state of Fe^2O^3, or Fe^3O^4, to the state of FeO, absorbs respectively 573 or 498 Calories, as may be readily deduced from the heats of formation of the three oxides concerned. If FeS is present its heat of formation is

(Fe, S) = 24.000 Calories = 429 Calories per kg. Fe.

If the iron is charged partly as silicate, such as mill or tap cinder, an additional amount of heat will be required for reduction, equal to that needed to separate the iron oxides from their combination with silica. The heat of formation of the bi-silicate slag only has been determined:

(FeO, SiO^2) = 8,900 Calories = 148 Calories per kg. SiO^2.

And since the cinders concerned contain relatively more iron than this, we can best make allowance for the heat required to set free the silica. It is necessary, therefore, to take the amount of silica in the iron cinder charged, and allow, as necessary for its decomposition into FeO and SiO^2, 148 Calories for each unit weight of SiO^2 contained.

(11) *Reduction of non-ferrous oxides.* Silicon is usually present in pig iron, its reduction from silica requiring:

(Si, O^2) = 180,000 Calories = 6,413 Calories per kg. Si.

There is a little doubt about this (Berthelot's) figure; more recent determinations, not yet published, point rather to 196,000 and 7,000 Calories respectively.

Manganese is often present in the ores as MnO, Mn^3O^4, or MnO^2, and going partly into the slag as MnO. The heat absorbed in reduction to manganese is:

(Mn, O)	90,000 Calories	= 1,053 Calories	per kg. Mn
(Mn^3, O^4)	328,000 "	= 1,088 "	"
(Mn, O^2)	125,300 "	= 2,278 "	"

For the MnO produced and going into the slag, the reduction from Mn^3O^4, or MnO^2, per unit weight of contained manganese, absorbs 3.5 or 0.25 Calorie respectively.

Sulphur generally comes from the reduction of FeS, requiring 667 Calories per kg. of sulphur; but care must be taken not to allow for this heat twice, since if reckoned once under the head of iron reduction it must not be reckoned on the balance sheet a second time under sulphur. One reduction of FeS absorbs both corrections.

Phosphorus may be reduced in large quantity in making basic iron. It probably comes mostly from calcium phosphate, in which case we must reckon on not only the heat of oxidation of phosphoric oxide but also its heat of combination with lime:

(P^2, O^5)	= 365,300 Cal.	= 5,892 Cal.	per kg. of P.
$(3CaO, P^2O^5)$	= 159,400 "	= 2,410 "	" contained.

Making a total heat requirement of 8,302 Calories to separate unit weight of phosphorus from phosphate of lime, and leave free lime. In a furnace making a pig iron with several per cent of phosphorus, this item becomes quite large.

Calcium occurs in the slag as CaS, its reduction from lime requiring

(Ca, O) = 130,500 Calories = 3,263 Calories per kg. Ca.

Other elements than the above rarely occur in pig iron in notable quantity. If rare ones occur, their heat of reduction can be sought in thermochemical tables. Those of tungsten, titanium, molybdenum and chromium are, however, not at present known.

(12) *Decomposition of moisture in the blast.* This is to be counted as vapor of water, the heat required to decompose, which is:

(H^2, O) vapor	= 58,060 Cal.	= 3,226 Cal.	per kg. H^2O.
		= 29,030 "	" H^2.

It is not correct to allow here for the sensible heat in this water vapor coming in with the hot blast, because that heat should go on the other side of the balance sheet as heat delivered to the furnace. Neither is it correct to subtract the heat of combination of the oxygen of this moisture with carbon to form CO at the tuyeres; because, although that combination actually does take place, yet the heat thereby evolved properly belongs also on the other side of the balance sheet, and is there properly taken care of as part of the heat of oxidation of carbon in the furnace.

A New Apparatus to Determine the Melting Points of Slags.

By Woolsey McA. Johnson.

A slag is essentially a complex silicate of several bases. Most prominent among these latter is calcium oxide, and in lead or copper blast-furnace work ferrous oxide. The several silicates usually crystallize out in the process of cooling, so that slag has not a sharp melting point in most instances.

An accurate knowledge of the "running temperature" of the slag is, of course, of great importance in any work in which it is necessary to produce a liquid slag under given metallurgical conditions, as in lead furnace practice, or to prevent the formation of slags, as in the zinc retort. The literature of the subject does not offer such data as were satisfactory, for in all cases the point determined is the uncertain "softening point." Such results are not directly translatable into actual practice, unless subject to great corrections from the basis of practical knowledge.

While working some years ago on a small "nickel brick cupola" of the Orford Copper Co., I had the opportunity, at the end of the short weekly campaign, just as the cupola furnace was going out of blast, of observing the process of slag formation on the coke through several of the small holes which were then appearing in the sides of the furnace. With the knowledge of this phenomenon in mind, I devised the following piece of apparatus to give results as directly applicable to practical work as was possible. The first execution of the work was done at the instance of Mr. Fred. W. Gordon, chief consulting engineer of the Lungwitz Reduction Co., at Warren, N. H., in the summer of 1905. A discussion of the characteristics of proper slag for this interesting metallurgical operation discloses certain conditions entirely and distinctly novel. And for this purpose my design proved quite serviceable and neat.

A round 4-inch bar of Acheson graphite was sawed off by a

carpenter's saw in 3½ inch lengths, a narrow slot, ⅝ inch wide and 1½ inches high, was then cut in bottom of this. In top a hole 1 inch in diameter was drilled with machine drill to a depth of 1 inch. This was continued to by a 7/32 inch hole to the narrow slot.

A horizontal 1-inch hole was drilled above line of lower slot to receive a Le Chatelier pyrometer to a point as near as possible to the 7/32-inch "dripping hole." Small proportions of

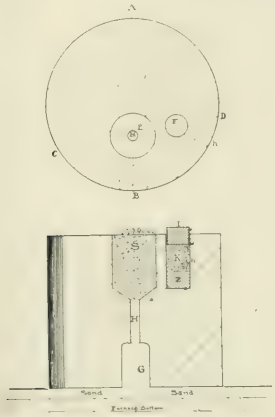


FIG. 1.—TESTING SLAGS.

the formed-slag, or slag-making materials, were placed in upper hole. The entire apparatus was placed in a special oil-fired experimental reverberatory furnace with a sand layer over hearth. The pyrometer was inserted through the hole in one side of the furnace.

The observing hole was on the opposite side of the furnace, and a view could be had of the slag directly through the slot when the slag reached the "dripping point," and dropped or ran through, depending on its viscosity.

The fact that the pyrometer couple was so close to the slag

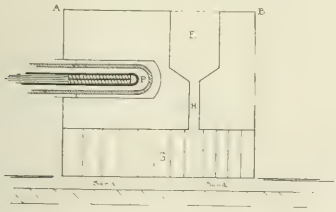


FIG. 2.—TESTING SLAGS.

at this critical temperature increased the accuracy of results. Even though slag is a poor conductor of heat, Acheson graphite is so good a one that the temperature gradient is small.

To ensure a check on the pyrometer we bored a ½ inch hole in one side, put 5 grams of zinc in this hole, covered with charcoal, and each time checked 1 pyrometer to the boiling point of zinc, about 918° C. at the altitude of Warren. The characteristic snaky flame of zinc vapor came through small 7/32-inch horizontal on side. The ½-inch hole was plugged with a whittled graphite plug.

The apparatus was also made much larger, and by allowing slag to drip through granulated coke column, 3 inches in diameter 4 inches high, we measured the rate of reduction of reducible oxides, FeO and ZnO. I also designed another appa-

ratus resembling in appearance the [illegible] I propose to measure rate of flow of slags through a small apperture at different temperatures.

Apparatus of other material than Acheson graphite is to be considered. Fire clay would flux with slag, silica, and iron would oxidize, and iron oxide so formed would flux with any slag.

Results on some 150 typical slags were very gratifying and determinations checked often to 5° C., although not sufficient in number, because of necessary rapidity of work. The work discloses not only corroboration of practical knowledge but also new and unlooked for quantitative data on this vast subject.

As it will be impossible for me to continue this work on the scale necessary, I will gladly give any advice on the subject to anyone with proper laboratory facilities. It certainly deals microcosmically with the operations of the blast furnace.

Vertical Arc Furnaces for the Laboratory.

BY SAMUEL A. TUCKER.

Graphite as manufactured by the Acheson process is a material which lends itself admirably to the construction of electric furnaces, and this is due to three causes: the ease with which it may be machined or shaped to the purpose required, its high electrical conductivity, and its freedom from impurities. It is, therefore, a most suitable material for the construction of electric furnace parts which are required to withstand a high temperature and in which the addition of carbon is of slight importance.

The vertical arc or pot furnaces, as used for some time past in this laboratory, are constructed after the pattern of the

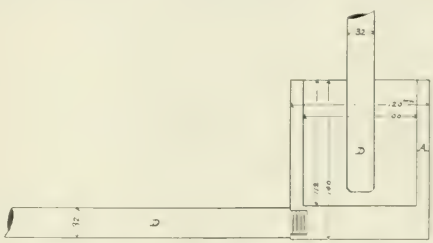


FIG. 1.—VERTICAL ARC FURNACE—ELEVATION.

Willson furnace, as shown in Fig. 1. *A* is the crucible, with the dimensions as given, *D* and *D'* are the electrodes, the material being of Acheson graphite. The horizontal electrode *D* is united to the crucible at *J*, the crucible being bored out and tapped for this purpose to admit the threaded end of *D*. The vertical electrode *D* is held by means of an appropriate clamp and means by which it may be raised and lowered at will.

The crucible *A* is turned from a solid electrode 5 inches in diameter, and this operation can be accomplished in the lathe in about 20 minutes, a cut of 3-16 inch being taken at a speed of 230 feet per minute with a feed of about 4 inches per minute. These figures are, of course, dependent to a certain extent on the operator, but the job need not take over 20 to 25 minutes, including the chucking of the work.

The crucible is surrounded with a heat insulating material and brick retaining walls, as in Fig. 2, in order to protect the graphite from oxidation and prevent loss of heat by radiation from the walls. The choice of material for this lining is a question of heat conserving and refractory qualities, and it has been found that slaked lime or powdery [illegible] as obtained from the barrel after it has stood about the laboratory for a time, gives satisfactory results.

The electrodes, which are the regular 1¼-inch diameter, will take 300 amps. for some time without undue heating, and for short runs the current can be raised considerably above this figure.

In using this furnace the arc is struck on the bottom of the crucible, and the charge fed in and subjected to the heat action of the arc from the charge to electrode *D*.

If the desired product is in the molten condition, alloys, etc., the vertical electrode is raised out of the way, the electrode holder, as in *I*, Fig. 2, disconnected, and the crucible lifted with a pair of tongs together with the horizontal electrode *D'*,

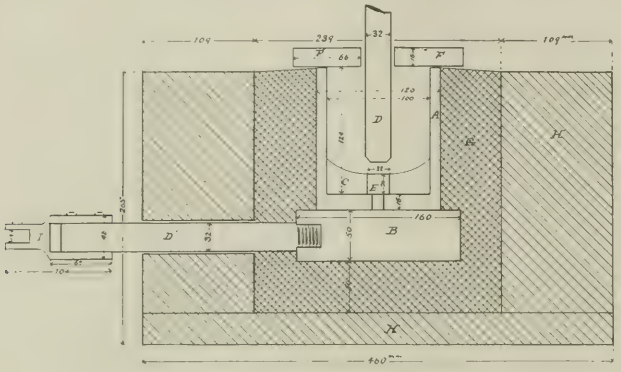


FIG. 2.—VERTICAL ARC FURNACE. ELEVATION.

and the contents poured out into a mould. In case the product cannot be poured, carbides, etc., it is taken out by the chisel from the crucible after it has cooled, and it is remarkable what an amount of hard treatment a crucible of this kind will stand.

The chief objection to this type of furnace is that the charge is brought in direct contact with the arc, and the temperature at this point is consequently raised far above that which is necessary to bring about the desired reaction. The consequence is that volatilization takes place at this point to such an extent that the yield is reduced very materially. To obviate this the duration of the run has to be shortened, and the bulk of the charge is not raised to a high enough temperature, particularly if it is placed at some distance from the sphere of the arc. So we then have a case of too high a temperature at one place and not enough at the other.

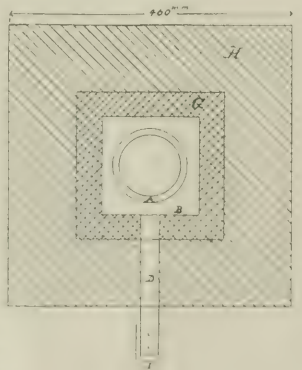


FIG. 3.—VERTICAL ARC FURNACE. PLAN.

A furnace of the Moissan type overcomes these difficulties to a great extent, as the temperature is maintained by the radiant heat from the arc, but furnaces of this type are somewhat difficult to construct, lack durability, and the charge is unevenly heated, that at the top of the containing crucible or opening being subjected to the highest temperature, while that at the bottom being comparatively far removed from the heated zone may not be raised to the required temperature.

Figs. 2 and 3 show a plan and elevation of an arc furnace designed to obviate these difficulties. As before, *A* is the crucible, *D* and *D'* being the electrodes, but D' instead of being connected directly to the crucible as in Fig. 1, is connected by the screw joint with a graphite brick or plate *B*. The crucible is thus in electrical connection with this electrode *D'* by means of surface contact with *B*, and this has been found amply sufficient, and has the advantage that the connection *D'* and *B* is a fixture, one of these answering for any changes in the crucible which may become necessary. Moreover, if the crucible is to be lifted out for pouring the contents, the operation is easier with a plain crucible without the attached horizontal electrode.

The crucible itself differs from the preceding one, in that at the bottom is placed a graphite plug *E*, raised about 20 mm. above the bottom. The arc is then struck from the vetical electrode *D* to this plug, and continues in this way throughout the run. Thus the charge *C* is heated by the radiant heat from the arc, and is not subjected to its direct action as was the case in Fig. 1.

The result is that the charge after a time becomes evenly heated to the required temperature, giving a better product of higher yield. A temperature of 2,000° C. may be maintained at the charge space, using a current of about 350 amps. at 35 volts drop across the furnace. This figure was determined by an optical pyrometer sighting through a graphite tube, the end of which was placed in the position occupied by the charge.

The crucible is protected by lime *G*, supported by the brick retaining walls *H*, the mouth being covered by any suitable material. For this purpose a carborundum wheel broken in half will be found suitable, and should the temperature be so high that this is melted, graphite plates can be substituted.

Thus the furnace consists essentially of a Moissan furnace of graphite, arranged vertically, with the advantages of greater ease of construction, and in which the charge is brought nearer to the source of heat.

Electrochemical Laboratory,
Department of Chemistry,
Columbia University.

American Society for Testing Materials.

The annual convention of the American Society for Testing Materials was held from June 21 to 23 at Atlantic City, N. J. The Iron and Steel section held three sessions, in which a large amount of business was transacted and numerous excellent papers were presented. Concise abstracts of all of them will be given in our Synopsis in our next issue.

The important subject of specifications for steel rails was again thoroughly discussed, but no progress can be recorded, since the whole matter was again referred to the committee A for further consideration. The trouble is that the rail manufacturers and the railway engineers within the committee can not reach an agreement on several important points.

The report of committee G on uniform speed in commercial tests recommends that speeds of 6 inches per minute for boiler and bar specimens (8-inch testing length), and 3 inches per minute for forging specimens (2 inch testing length), be considered maxima; below these values speed has no influence on ultimate strength and elongation. The determination of elastic limit should be made at a low speed, say ½ to ¼ inch per minute.

The heat treatment of steel and the corrosion of iron and steel were also the subject of extended discussion. These will be dealt with in our next issue.

Electric Smelting of Iron Ore.

Dr. Eugene Haanel's official "preliminary report on the experiments made at Sault Ste. Marie, Ont., under the Canadian Government auspices, on the smelting of Canadian iron ores by the electrothermic process," has just been made public. Since the chief results of these tests have already been covered in an illustrated article in our April issue (page 124), we abstract from Dr. Haanel's report only those parts which contain new information.

Construction of Furnace.

The electric furnace was designed by Dr. Paul L. T. Héroult. It is shown in the adjoining diagram.

The furnace consisted of an iron casing bolted to a bottom plate of cast iron, 48 inches in diameter. The casing was made in two cylindrical sections, to facilitate repairs. To render the inductance as small as possible, the lines of magnetic force in the iron case were prevented from closing by the replacement of a vertical strip of 10 inches width of the casing by a copper plate. Carbon paste was rammed into the lower part of the furnace up to the bottom of the crucible. The lining consisted of common fire-brick, which from the bottom of the crucible up for a distance a little above the slag level was covered with carbon paste to a thickness of a few inches. The crucible, therefore, consisted entirely of carbon.

The lining of the furnace was given the shape of a double cone set base to base. Changes in the dimensions of the interior were made from time to time, as indicated by experience, but for the majority of the experiments they were as follows:

	Inches.
Diameter of bottom of crucible	24
Height of lower cone	11
Height of upper cone	33
Diameter of joint base of the two cones	32
Diameter at top of furnace	30

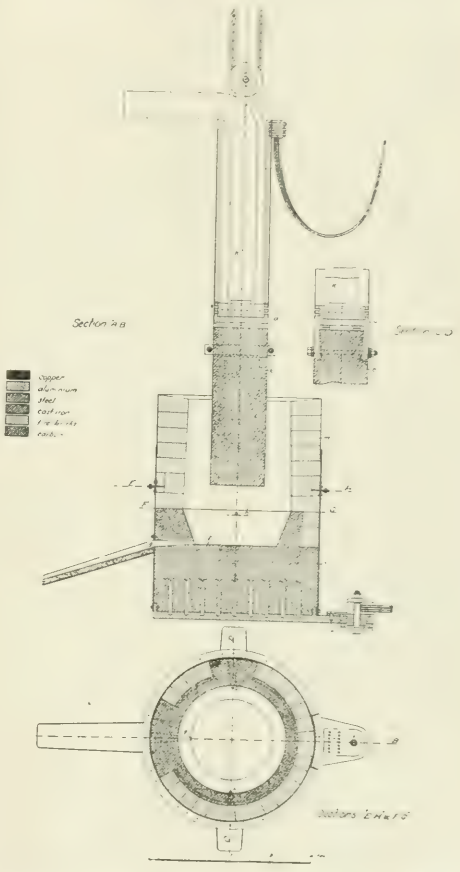
The electrodes, manufactured by the Héroult process (which is kept secret) and imported from Sweden, were prisms of square cross-section, 16 x 16 inches by 6 feet long. The contact with the cables carrying the electric current to the electrode consisted of a steel shoe riveted to four copper plates, which ended in a support for a pulley. The electrode with its contact was supported by a chain passing under the pulley, one end of the chain being fastened to the wall, the other end passing over a winch operated by a worm and worm-wheel. This formed a convenient arrangement for regulating the electrode by hand.

The electrical energy was furnished by one phase of a three-phase, 400-kw., 30-cycle, 2,400-volt alternator. Current was supplied at 2,200 volts to a 225-kw. oil-cooled transformer, which lowered the voltage to 50. The transformer was placed in a separate room in the furnace building, close to the furnace. From the transformer the current was led to the bottom plate contact of the furnace, and to the electrode contact by conductors, consisting each of thirty aluminium cables, 5/8 inch in diameter.

Experiments.

A number of experiments required to be made to adjust the capacity of the crucible of the furnace to the energy available, and to determine the shape to be given to the interior of the furnace, to insure easy passage of the charge into the reducing and melting zones. After this attempts were made to utilize the calorific energy of the carbon monoxide resulting from the reduction of the ore, which in all experiments so far recorded had been wasted. To accomplish this, air under pressure was introduced into the furnace about 12 inches below the upper level of the charge. The carbon of the charge in the form of coke dust, was mixed with fire-clay, and briquetted to prevent it from being consumed by the air blast. It was argued that by thus utilizing the carbon monoxide in preheating the charge and partially reducing the ore, the output would be materially increased.

It was found, however, that the great heat evolved by the combustion of the carbon monoxide caused the charge to become sticky and to hang. Nor could this be remedied by stoking, the space between the walls of the furnace and the electrode being too narrow. Moreover, the electrode, although it had been protected with asbestos and iron sheeting, was found after the experiment to have been badly overheated. The furnace was not at all adapted for these experiments, and further attempts in this direction were abandoned. The experiments, however, showed that with a different construction furnace, in which the electrode is isolated from the charge, the output might be greatly increased by the introduction of an air blast.



Sections of Héroult Electric Furnace for Iron Ore Reduction

Smelting of Magnetite.

Magnetite is the chief ore, and is to some extent a conductor of electricity. It was expected that considerable difficulty

[1] See the patent of Héroult, on page 152 of our April issue, and the editorial discussion on page 165 of our May issue. It will be seen, however, that the procedure at the Soo was different from the process as patented, since in the latter the carbon is introduced through the hollow electrode separately from the ore.—Editor.

would be experienced in the smelting of magnetite on account of its conductivity. It was thought that with the furnace in use, in which the electrode was immersed in the charge, the current would disseminate itself laterally from the sides of the electrode through the charge, preventing the current at the reducing and fusion zone from attaining such density as would be required for the high temperature necessary for reduction and fusion. With charcoal as a reducing agent no difficulty was experienced in this respect, nor was the inductance of the furnace increased by the presence of magnetite.

Charcoal can be cheaply made from mill refuse and other sources of wood supply useless for other purposes. Charcoal and peat-coke constitute home products in the provinces of Ontario and Quebec, while coal-coke must be imported. No difficulty whatever was experienced with charcoal as reducing agent, without being briquetted with the ore. "In fact, so admirably adapted was charcoal, when crushed to pass a ¾-inch ring, as a reducing agent in the electric furnace, that coke and briquettes of coke with clay were abandoned, and all the experiments with magnetite and roasted pyrrhotite were made with charcoal." Some of the charcoal available was of very poor quality, some of it being little better than charred wood, containing only about 56 per cent of fixed carbon. ("This and the fact that a considerable quantity of the charcoal was consumed on top of the furnace, account for the large quantity of charcoal used per ton of product. A modification of the furnace, protecting the upper layer of the charge from the atmosphere, and the use of charcoal properly carbonized, would decrease considerably the amount of charcoal which was actually used in the experiments, and consequently reduce the cost of production.")

The following is a record of one run ("No. 14" in the report), in which magnetite from the Blairton mine was treated with charcoal as reducing agent and limestone and sand as flux:

Analysis of raw materials:

Blairton ore—0.60 per cent SiO_2, 60.74 Fe_2O_3, 17.18 FeO (55.85 Fe), 1.48 Al_2O_3, 2.84 CaO, 5.50 MgO, 0.037 P_2O_5 (0.016 P), 0.57 S. 5.053 CO_2, and undetermined.

Charcoal—14.06 per cent moisture, 28.08 volatile matter, 55.90 fixed carbon, 2.54 ash.

Limestone—1.71 per cent SiO_2, 0.81 Fe_2O_3 + Al_2O_3, 92.85 $CaCO_3$, 4.40 $MgCO_3$, 0.004 P, 0.052 S.

The sand used was common furnace sand, of which no analysis was made.

Composition of charge—400 pounds ore, 125 pounds charcoal, 25 pounds limestone, 6 pounds sand.

Analysis of products:

Iron produced ("cast No. 80," grey iron)—Total C 3.73 per cent, Si 3.53, S 0.042, P 0.034.

Slag produced—33.80 per cent SiO_2, 10.20 Al_2O_3, 21.79 CaO, 30.50 MgO, 2.05 S, 0.25 Fe.

Ratio of weight of slag to that of iron = 0.41.
Length of run, 65 hours 30 minutes.
Mean volts on furnace, 36.03.
Mean amperes, 4987.
Power factor, 0.919. hence
Watts 165,125, or electric horse-power, 221.34.
Pig iron produced, 11,989 pounds, hence
Output of pig iron per 1,000 e. h. p. days = 0.92 tons, or
E. h. p. year per ton of pig = 0.276

Reduction of Sulphur and Production of Ferro-Nickel Pig.

Another important result of the experiments was the fact that ores of high sulphur content, not containing manganese, can be made into pig iron containing only a few thousandths of a per cent of sulphur.

In the following run ("No. 18" of the report) roasted pyrrhotite was treated with charcoal as reducing agent and limestone as flux:

Analysis of raw material:

Roasted pyrrhotite—10.96 per cent SiO_2, 3.31 Al_2O_3, 65.43 Fe_2O_3 (45.80 metallic Fe), 3.92 CaO, 3.53 MgO, 1.56 S, 0.016 P, 0.41 Cu, 2.23 Ni.

Charcoal and limestone same as in previous runs.

The limestone in the charge was decreased from 120 pounds when starting to 50 pounds. The composition then being 400 pounds ore, 110 pounds charcoal, 50 pounds limestone.

Analysis of products:

Iron Produced.	*Cast No.* 125.	*Cast No.* 130.	*Cast No.* 133.
Total C.....	3.23 %	3.38 %	2.50 %
Si ...	4.90[2]	4.50[2]	6.32[3]
S	0.007	0.006	0.007
P ...	0.062	0.037	0.042
Cu ..	0.86	0.87	0.71
Ni ..	3.70	4.12	4.00

Slag produced—16.44 per cent SiO_2, 13.86 Al_2O_3, 53.25 CaO, 8.80 MgO, 5.28 S, 0.65 Fe, traces of Cu and Ni.

Ratio of weight of slag to that of iron = 0.69.
Lenth of run, 50 hours 20 minutes.
Mean volts on furnace, 36.05.
Mean amperes, 5,000.
Power factor, 0.919. hence
Watts 165,649, or e. h. p. = 222.05.
Pig iron produced (ferro-nickel) 7,336 pounds, hence
Output of ferro-nickel pig per 1,000 e. h. p. days = 7.038 tons.
E. h. p. year per ton of pig = 0.389.

On account of the enhanced selling value of the ferro-nickel pig, the furnace has been taken over by the Lake Superior Power Co. for use in commercial operation.[3]

Smelting of Titaniferous Iron Ore.

In the following run ("No. 19" of the report) titaniferous iron ore was treated with charcoal as reducing agent and limestone and fluorspar as flux:

Analysis of raw material:

Titaniferous iron ore—7.12 per cent SiO_2, 30.30 Fe_2O_3, 28.78 FeO (43.59 Fe), 7.00 Al_2O_3, 1.00 CaO, 4.14 MgO, 0.064 P_2O_5 (0.028 P), 0.04 S, 17.82 TiO_2, and 2.50 Cr_2O_3 (1.42 Cr).

Charcoal and limestone as in previous runs.

Composition of charge—400 pounds ore, 100 pounds charcoal, 50 pounds limestone, and 50 pounds fluorspar.

Analysis of Products:

Pig Iron Produced.	*Cast No.* 136.	*Cast No.* 137.
Total C.....	... %	3.50 %
Si	4.50	2.80
S	0.007	0.091
P	0.143	0.060
Ti (appr.).	1.00	1.30

Slag produced—7.00 SiO_2, 28.50 Al_2O_3, 14.23 CaO, 2.93 MgO, 38.92 TiO_2, 1.13 Fe, 0.90 S.

On account of the furnace being in a very bad condition, the lining being eaten away by the limey slag used in the previous run, the run had to be stopped and no figures as to output could be obtained.

The slag was very fluid, and likely the fluorspar in the charge could have been reduced considerably or omitted altogether.

The iron obtained in cast No. 136 was probably mixed with

[2] "By increasing the limestone of the charge the silicon content of the ferro-nickel pig recently produced has been depressed to 2 per cent."

[3] It is reported from a reliable source, although not authoritatively confirmed, that the process pays its way in commercial operation, but would be made far more remunerative by increasing the capacity, since the cost of attendance is comparatively high for the size of furnace (which was originally intended for the experimental work only). With an increase of capacity of furnace, the cost of attendance would not be materially raised.—Editor.

some iron from the previous charge, when ore with high phosphorus content was used.

"This experiment made with a titaniferous iron ore containing 17.82 per cent of titanic acid permits the conclusion that titaniferous iron ores up to perhaps 5 per cent titanic acid can be successfully treated by the electric process."

General Remarks.

The ores treated, with the exception of the hematite and the roasted pyrrhotite, contained a high percentage of magnesia, producing a very infusible slag. When the furnace had been running for some time this infusible material formed a scale around the crucible, the electric energy available not being sufficient to keep it in a molten condition. The crucible and lower part of the furnace were, therefore, partially filled up, preventing easy access of the charge to the reducing and melting zone. This slower feeding left the charcoal on top of the furnace exposed to the air a longer time, thus increasing the amount of charcoal required and decreasing the output. With a greater current than was available and consequent higher temperature, the formation of the scale would have been prevented, and the output correspondingly increased.

The electric installation at the disposal of the experimenters was far from ideal for electric smelting experiments. Aside from the drop of voltage, due to the frequent slipping of the belt connecting motor and generator, it was impossible to increase the current beyond 5,000 amps. at from 35 to 40 volts.

This inelasticity of the system prevented the determination of the most suitable current and voltage for a given charge in the furnace.

Consumption of Electrode.

For the production of 42,711 pounds of pig iron 384 pounds of electrode were consumed, the same electrode having been in commission for thirteen days.

The consumption of electrode per ton (2,000 pounds) of pig iron was, therefore, 17.98 pounds.

During the time this electrode was in commission the material in the furnace was melted down several times, exposing the red-hot electrode to the oxidizing atmosphere. The consumption of electrode was found to be greater for white iron than for grey iron, and since the 42,711 pounds of pig iron produced included several casts of white iron, the consumption of electrode was also on that account greater than it would have been had only grey iron been produced.

Modification of Experimental Furnace for Commercial Production of Pig Iron.

Probably the largest unit which can at present be constructed on the model of the experimental furnace will not exceed 1,500 hp. The construction of the experimental furnace to fit it for the production of pig iron on a commercial scale will require to be modified in the following important particulars:

(1) The top of the furnace requires to be modified to permit of the application of labor-saving machinery for charging.

(2) Provision requires to be made for the collection and utilization of the carbon monoxide produced by the reduction of the ore; this involves also the protection of the charcoal of the charge from combustion on top of the furnace.

The greater capacity insuring less loss of heat by radiation and the modification of the furnace to permit of the utilization of the carbon monoxide, will materially increase the output beyond that ascertained by the experimental furnace. The experiments indicated that under *normal* conditions about 11.5 tons were produced by an expenditure of 1,000 e. h. p. days.[4]

[4] This figure is the same as was given by us on page 126 of our April issue, on the authority of Dr. Héroult. It may also be expressed in the form that 1 ton (2,000 lbs.) of pig iron produced requires 83 hp-days. In the commission report of two years ago (our vol. II., page 484) Mr. Harbord gave the figure 128 hp-days per ton, of course, on the basis of experiments made in a cruder and more experimental furnace. It is, therefore, not unreasonable to assume that under similar conditions with a properly constructed plant the output per 1,000 e. h. p. day would certainly reach 12 tons. The latter figure has been adopted in calculating cost of production per ton of pig.

The protection of the charcoal of the charge from combustion on top of the furnace will materially decrease the amount of charcoal necessary for reduction and consequently lessen the cost of this item. This saving has, however, not been taken into account in the estimate of cost.

Estimate for 10,000-hp. Plant.

The following estimate for a 10,000-hp. plant, producing 120 tons of pig iron per day of 24 hours, is given in the report on the authority of Dr. Paul L. T. Héroult:

Furnaces, contacts, overhead work	$24,500
Bins, chutes, elevators	14,000
Crushers	4,000
Hoists and regulators	10,500
Instruments	1,400
Cables for conductors	8,400
Building	10,500
Mixer and casting machine	10,000
Traveling crane and tracks	5,000
Ladles	1,500
Slag trucks	3,000
Ore bins	3,000
Repair shop	5,000
	$100,800
Charcoal plant	500,000
Power plant (assuming cost of developing 1 e. h. p. = $50.00)	50,000
	$650,800
Electrode plant	6,000
Unforeseen expenditure	43,200
	$700,000

Dr. Héroult emphasizes that the figure of 83 hp-days per ton of pig iron is a safe figure and represents the upper limit of electrical energy consumption in a properly constructed commercial electric furnace. If the carbon monoxide is burned to dioxide within the furnace itself, the energy consumption would be considerably reduced. The theoretical thermochemical data on this point may be found in our analysis of Dr. Héroult's corresponding patent, on page 152 of our April issue. According to Héroult, the production of 1 kw of iron requires 200 grams of carbon and 1 electric hp-hour (an allowance being made in this estimate for losses by radiation and for heat escaping with the gases). This may, for the present, be considered as the lower (theoretical) limit of fuel and energy consumption. One e. hp-hour per kg iron is equivalent to 38 hp-days per ton of 2,000 lbs. of iron.—Editor.

[5] Concerning this point, which is of most fundamental importance, reference may be made to the very conservative figures of Dr. Louis Bell on the cost of power generation for electrochemical works, our vol. II., page 301. The cost of developing hydraulic power varies enormously according to local conditions. In any region where electrochemical work is likely to be carried on, hydraulic rights have to be purchased at prices varying from, say $5 to $25 per horse-power, and available throughout the year. Then comes the work of development, building dam and canal, blasting and dredging, building roads and retaining walls and power house and setting wheels. The total is most uncertain—almost anything from $25 or so per horse-power to $100 or more. The latter figure would be too large for serious consideration in connection with electrochemical work, and the smaller amounts a mere combination of good fortune and low prices. The total cost of the electric generator and transformer plant, given by Dr. Bell as running "from $15 to $25 per kilowatt capacity in plant reaching the case limit only in large work under very favorable conditions," a figure of $15 to $25 per kilowatt are equivalent to $20.25 to $33.75 per horse-power. Hence, if there is no transmission line, the total cost of power, according to Dr. Bell, will vary between about $36 to $109 per horse-power. Note that Dr. Héroult's figure of $50 is within these limits. As an article in our vol. II., page 121, Mr. Louis Simon considers that Dr. Bell's figures are too high, and that for an electrochemical power plant the cost of developing one electric horse-power may not be more than $10.—Editor.

Amortization	5%	
Depreciation	5%	15% on $700,000 = $105,000.
Interest	5%	

On a production of 43,200 tons per year of 360 days per ton of pig iron..... $2.43

Cost of Production Per Ton Pig Iron.

Ore (55% metallic iron) at $1.50 per ton.....	$2.70
Charcoal, ½ ton, at $6.00 per ton.....	3.00
Electric energy, amortization, etc.....	2.43
Labor	1.00
Limestone	0.20
18 pounds of electrode at 2 cents per pound.....	0.36
General expenses	1.00
Total	$10.69

Dr Haanel's report is designated as a "preliminary" one, so that a report, giving in detail all records of all tests made, is presumably to be issued later on. However, the present preliminary report appears to contain everything that is essential. Undoubtedly the results of the tests at the Soo have furnished us with important figures, on the basis of which it may be safely estimated in any special case, whether or not electric smelting of iron ore can be economically carried out in a given locality.

* * * *

Since the above report marks in a certain way the end of what may be considered as the experimental epoch of the electrometallurgy of iron and steel, a concise summary of what has or has not been accomplished seems now in order. We will try to classify somewhat the vast amount of information which has been formerly published in our columns on this subject. In the following references to articles published in this journal the Roman figure represents the volume, the Arabic figure the page.

Manufacture of ferro-alloys.—For making high percentage ferro-alloys the electric furnace now reigns supreme, especially in the manufacture of those ferros in which a highly refractory oxide (like that of titanium) is to be reduced. For this problem the electric furnace is superior to the blast furnace simply on account of the higher temperature available. The chief technical problem is to keep the carbon content down. See the summary of Scholl, II., 349, 395, 449; processes of Rossi, I., 523, III., 322, 362; Héroult, I., 467, III., 155; Gin, I., 131, II., 149, 324; Sjostedt, I., 353, II., 153, 188, 207, IV., 128; de Alzugaray, II., 430; Stewart and Kuegelgen, II., 468, IV., 32; Price, III., 170, 171; Vielhomme, III., 362; other notes and articles on ferro-alloys, I., 105, 583; II., 65, 122, 145, 199, 318; III., 79, 84, 120, 133, 168, 210, 226, 254, 423, 448, 459; IV., 195, 247

Manufacture of high-grade steel.—The electric furnace seems to be destined to gradually replace the crucible steel process. The advantages of the electric furnace are lower cost of operation, due to larger units and consequently smaller labor charge per ton of output and greater durability of the electric furnace. The cost of electric power is a comparatively small item in this case, and becomes almost insignificant if fluid steel is supplied from an open-hearth furnace or from a Bessemer converter to an electric furnace for refining. It is likely that steel which has been refined in the electric furnace at a small cost, may be used in future to a much greater extent than is now possible with the high price of crucible steel. In this case electric steel will cover a broader field than is now covered by crucible steel. An impartial and authoritative statement of the situation in 1904 was given in the report of the Canadian Commission, visiting Europe (II., 479). Dr Haanel being chairman. Of the numerous processes which have been proposed, tried and tested, at least two are now in successful operation in this country; the Héroult process in a 50-ton per day electric furnace at the plant of the Halcomb Steel Co., in Syracuse (III., 338, IV., 164) and the Colby induction furnace at the Tacony works of Henry Diston & Sons, near Philadelphia (capacity 200 pounds of steel). See concerning process of Héroult, I., 63, 287, 449, II., 408, 481, III., 156, 337, IV., 31; Keller (similar to Héroult), I., 162, 420, IV., 167; Colby induction furnace, III., 134, 299, 341, 373; Kjellin induction furnace, I., 70, 141, 283, 376, 526, 576, II., 479, III., 294, 395, 432, 481; Schneider induction furnace, II., 283, 323, III., 156; Gin, II., 20, III.; 372, IV., 167; Girod, II., 309, III., 434; Stassano, III., 391, IV., 167; Minet-Neuburger, II., 369; Gerard, III., 111. Concerning the subject in general see also I., 462; II., 475; III., 311.

Reduction of iron from ore.—The above report of Dr. Haanel and our preliminary notes on the Soo experiments (III., 447; IV., 84, 124, 164) contain the best information now available on the question as to the special conditions which would render an electric furnace equal or superior to the blast furnace. See also the report of Goldschmidt on the Stassano furnace, I., 247, and the report of the Canadian Commission, visiting Europe, II., 483; also I., 277, 336, 397; III., 53. Furnace designs: Héroult's, I., 467 and above report; Keller, I., 420, II., 483, IV., 167; Harmet, I., 422, 587, 588, II., 110, 503; Stassano, I., 247, 461, III., 391; Conley, I., 426, II., 424; Lodyguine, III., 178; Gerard, III., 111; Contardo, I., 63, II., 110.

Various special applications of the electric furnace in iron and steel metallurgy.—Steel mixer, Héroult, IV., 30. Use of electric heat in combination with open-hearth furnace for taking off the peak of the load, Richards, II., 178. Fritting together of iron concentrates from magnetic separators, Ruthenburg, I., 58, 84, 202, 482, 494, II., 315, 486, III., 410, 481, IV., 238. Treatment of New Zealand iron sands, Galbraith III., 112, 344, 346. Treatment of black sands of Pacific coast, Day, III., 445, IV., 4. General articles on electrometallurgy of iron and steel, Goldschmidt, I., 461; Bennie, II., 307; Neumann, II., 488; Canadian Commission report, II., 475, 479.

The Lines of Current in Storage Batteries—An Experimental Study.

By M. U. Schoop.

While our knowledge of the distribution of the lines of current in metallic conductors and of the distribution of the lines of force in a magnetic field is relatively perfect, we are much less informed on the distribution of the lines of current in electrolytes and on the true current-density at different points of an electrode surface. This is indicated by the lack of literature on this subject.

As an interesting contribution to this subject, I may first mention an investigation by A. Tribe,[1] who placed small longitudinal intermediate electrodes at different points of the electrolyte. The deposit obtained on each intermediate electrode was taken as an indication of the electric conditions at that point. Since they act as bipolar electrodes, polarization currents must result between the two sides of the electrode so that this method is not suitable to give an exact picture of the "electrolytic field." J. Stark[2] studied in great detail the path which the current takes if a small electrode is placed between the main electrodes, whereby the original metal forming the auxiliary electrode produces local currents with the electrolytic deposit.

Of great interest is, finally, an investigation of Dr K Norden. The difference in current density at different points of an electrode was rendered by him directly visible by the qualitative change of the deposit from a mixture of solutions of copper sulphate and zinc sulphate upon an intermediate electrode. On the supposition that the intermediate electrode is perfectly parallel to the main electrodes and fills the cross-section of the electrolyte as perfectly as possible, the differences

[1] Phil. Mag., 1881, page 446.
[2] Wiedmann's Annalen, 1898, page 245.
[3] Paper read before the annual convention of the German Electrochemical Society, in Zurich, 1900. See Zeit. f. Elektrochemie, 1900

in the current density at different points of the electrode are indicated directly by the color of the deposit, since the color changes with the proportion of the copper to the zinc in the deposit.

In the following I will describe a method which permits to study, with relatively simple means, the distribution of the lines of current in electrolytes with sufficient exactness for all practical purposes. I have described this method in a paper read on May 18, 1906, before the French Physical Society in Paris.

I.

If we pass a current through an electrolyte between two electrodes, the lines of current will not be held together so as to have the shortest possible length. On the contrary, the lines of current will always fill the whole space filled with the electrolyte. The relation which is valid for metallic wires, namely, $R = c \frac{l}{q}$ where R is the resistance, l the length, q the cross-section and c the resistivity of the wire, can be used directly for electrolytes only under the supposition that there is no straying of lines of force. For instance, in the special case in which the electrolyte is enclosed in a cylindrical tube, which is closed at both ends by electrode surfaces, having uniform conductivity all over their area, the lines of current may be assumed to pass uniformly in straight lines through the electrolyte from one end to the other.

In all other cases there is a straying of lines of current whereby the length of the lines of current from one electrode to the other electrode is increased, while the cross-section (or the number of ions transporting the current) is also increased. The increase in the length of the lines of current raises the resistance, while the increase of cross-section diminishes the resistance. As may be easily shown by an experiment, the latter effect is predominant, so that the total resistance is diminished. It is, therefore, possible to state the law in this form: The ions which transmit the electricity do not choose the geometrically shortest way, but the electrically shortest way (that is, the path of least total electrical resistance).

It is true that the influence of the electrolytic "shunt current" on the distribution of the lines of current and on the current density on the electrode surfaces must be smaller the

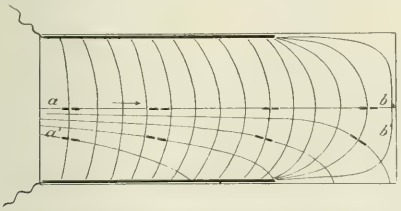


FIG. 1.—TEST OF STRAYING OF LINES OF CURRENT.

nearer the electrodes are together. One might, therefore, think that this effect is insignificant in storage batteries, when the distances between plates are, as usual, between 3 and 15 mm. This assumption, however, would be wrong, as we shall see later on.

In the determination of the distribution of lines of current in storage batteries, two different problems should be distinguished; namely, *first*, what is the distribution of the current in the electrolyte under the assumptions that there is no drop of voltage in each electrode itself (which are, therefore, assumed to be of infinitely high conductivity), and that the conductivity of the electrolyte is uniform throughout? *Second*, what is the current density at different points of the electrode surface, and how is the distribution of the lines of current affected by the voltage drop in the electrodes, by the non-uniform conductivity of the electrolyte and by the straying of the lines of current (*i. e.*, by the electrolytic "shunt current")?

The first problem may be easily solved by the following simple experiment: In a long vessel of celluloid, two bright sheets of lead are placed, as indicated in Fig. 1. The influence of non-uniform acid density and concentration is eliminated in this arrangement. The sheets are placed directly on the walls of the vessel, but it is preferable to provide their inside surface (which is attached to the wall) with a coating of insulating paint. I found celluloid paint quite useful, made by dissolving pieces of celluloid in amyl acetate.

From a very thin, spongy lead plate two small rectangular pieces are then cut out and combined in form of the little instrument shown in Fig. 2, which is called the "analyzer." Since the distribution of the lines of current in the electrolyte should not be changed by the analyzer, the dimensions of the two rectangular electrodes should be chosen as small as possible, and the little glass tubes containing the two wires which lead to the two electrodes should also be small. These two wires are connected with a Weston millivoltmeter or a suitable sensitive galvanometer.

The analyzer is then placed in the electrolyte and turned around its axle of symmetry until the voltmeter no longer gives any deflection. The two electrodes of the analyzer are then on an equipotential surface, and the lines of current are perpendicular to it. If it is thought that it is not possible to turn the analyzer with sufficient exactness by hand, a frame may be used along which the analyzer may be displaced.

FIG. 2. ANALYZER

The millivoltmeter gives zero deflection if the analyzer is displaced, for instance, along the line *a b* in the center of the vessel, Fig. 1, the analyzer being moved as indicated by the arrow. In this case the analyzer electrodes are parallel to the two main electrodes and are at equal distance from both.

If the same experiment is repeated, but starting from a point nearer to one main electrode than the other, then the analyzer must be turned more to one side, the further it is passed along the vessel. In this way the equipotential curves shown in Fig. 1 are obtained. By plotting in the diagram curves which are at every point perpendicular on the equipotential curves one gets the lines of current which are also shown in Fig. 1.

If one wants to avoid the formation of gas, which always occurs with lead plates, the latter may be replaced by fresh accumulator plates, but it is to be remarked that in this case a symmetrical distribution of current over the electrode surfaces (*i. e.*, equal current density at all points) is no longer guaranteed, since the conductivity of the active material, especially with spongy lead plates, may not be at all uniform all over the plate.

The lines of current, shown in Fig. 3, were obtained with two bright lead plates. This figure is only intended to indicate the course of the lines of current and of the equipotential lines, and not to give quantitative information on the distribution of the current in the electrolyte or the distribution of the current density over the electrodes.

The distortion of the lines of current, which is the greater the nearer one gets to the right-hand end of the vessel, is clearly shown. If an insulating wall would be inserted from *b* to *b'* all lines of current would be straight lines perpendicular to the two electrodes. If this insulating wall would be inserted in the manner indicated by C in Fig. 3, another distortion of the lines of current would result on account of the electrolytic shunt circuit being cut up.

One of the first experiments which I made in order to get an exact picture of the lines of current in electrolytes was as follows: A horizontal tray, such as used for photographic purposes, was filled with hot paraffine to a height of 4 mm. When this layer of paraffine had solidified dilute sulphuric acid was poured onto it. As analyzer I used in this case two plati-

num wires in form of a fork at the end of a glass tube, connected with the terminals of a telephone. In this case alternating current was, of course, used, or a pulsating current produced by superposing alternating current on the direct current. The analyzer was revolved until the sound in the telephone became a minimum. Then the two platinum points were pressed into the paraffine. In this way dots were obtained in the paraffine, giving directly the equipotential curves.

If one wants to make the experiment in a somewhat more exact way, a tray of hardwood with perpendicular walls should be used, as shown in Fig. 4, two strong nails being symmetrically arranged at both ends and connected to the terminals of the current. By means of an insulating thread around one nail and around the analyzer, the latter is conducted around the nail in a circle.

This experiment is evidently analogous to the classical fundamental experiment of Kirchhoff on the distribution of the lines of current on metallic surfaces (Poggendorf's *Annalen*, Vol. 64, p. 497, 1845). I might mention, however, that I was led to this possibility of determining the distribution of the lines of current in electrolytes by a personal observation, which induced me to study the subject in greater detail for several months.

This observation was made by me while I experimented on alternating-current electrolysis. While putting two fingers into the electrolyte, I found the physiological effect to be more or less evident, or to disappear altogether according to the

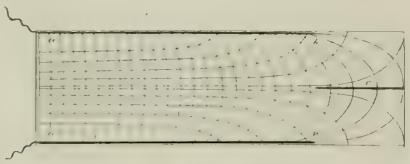


FIG. 3.—LINES OF CURRENT AND EQUIPOTENTIAL CURVES.

position of the fingers in the electrolyte. This position determined, of course, the voltage difference between the two fingers.

If an electrolyte of poor conductivity is used with a strong current, the index finger and the middle finger may be used as a good automatic indicator, since the fingers have a natural tendency to get as quickly as possible into the equipotential line (if they do not jump quickly out of the electrolyte altogether). Little fishes also assume immediately a position along an equipotential surface, so that no current can pass through them. Since fishes are very sensible toward electrical effects a very small voltage should be used. With a 110-volt supply they die immediately.

Against the use of electrode plates the objection may be made that the original distribution of the lines of current can be changed by the introduction of the electrode plates into the electrolyte, and further that polarization may occur. But this objection is no longer valid for the use of the alternating current and telephone, since the dimensions of the analyzer with the platinum wires may be made so small that an effect on the distribution of the lines of current is rendered practically impossible. An influence of polarization phenomena on the result is also practically excluded.

With the telephone it can be conclusively proven that everywhere where there is electrolyte there are also lines of current. For this experiment a high beaker may be placed between two electrodes in a larger vessel, the experiment being arranged as shown in Fig. 5. This apparatus can be easily put up with simple means, and is also quite suitable for some other experiments on the distribution of lines of current in the electrolyte.

On a brass rod *a* two U formed glass tubes are fastened, through which the two wires *c c* pass, which are connected to the electrodes of the analyzer. By means of these wires the analyzer may be raised or lowered. The glass tube of the analyzer, containing the two wires, should be filled with paraffine, and the two terminals should consist of two small plates instead of points. By means of this apparatus it can be easily shown that as long as the analyzer plates are parallel to the main electrodes, hence perpendicular to the lines of current, it is impossible to stop the noise in the telephone,

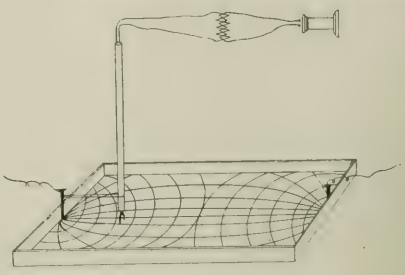


FIG. 4.—SIMPLE ARRANGEMENT OF TEST.

although, of course, the noise gets less and less the more the analyzer is lowered to the bottom of the beaker.

II.

Of equal theoretical and practical importance is the question, how the straying of the lines of current in an electrolyte influences the distribution of the current on the surfaces of the electrodes; *i. e.*, the current density at different points of the electrodes. That there is such an effect we have already seen, the current density increasing at the points of the electrode surfaces near the "electrolytic shunt." For the present we will not consider how the current density on the electrodes is influenced by the manner in which the current enters or leaves the electrodes, or by any lack of uniformity of conductivity over the electrodes or within the electrolyte.

A number of phenomena, well-known from practice, may first be mentioned. With soluble electrodes, with or without stirring, the anode is not uniformly dissolved at all places, but is dissolved more strongly in its lower half, so that when a large part of the electrode has already been dissolved the upper part is often almost unchanged. The same happens even if care is taken to have a good, uniform current supply and efficient circulation.

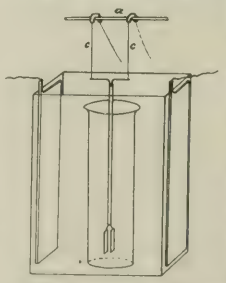


FIG. 5.—PROOF THAT THE LINES OF CURRENT FILL THE WHOLE ELECTROLYTE.

This phenomenon is in agreement with the observation that the loss of capacity in spongy lead plates of storage batteries which have been in use for some time, is greater in the lower parts of the plates than in the upper ones. I once determined the capacity of a spongy lead plate, 210 × 210 × 6 mm., of a stationary storage battery which had been used for three years, and which showed only one half its original capacity, so that the negatives of this battery (which had been made for a prominent German storage battery factory) had to be replaced. When the plates were carefully examined it was seen that the contraction of the spongy lead was most evident near the lower rim of the plate. I then cut two exactly equal pieces from the lower and upper portions of the plate and determined their capacity separately. I was surprised to find that the upper piece had still 85 per cent of its original capacity, but the lower piece only 40 per

cent. It is probable that the negatives of other makers will show a similar behavior, especially as most stationary types of plates are not of square form (as was the case with the plate which I tested), but are considerably longer than they are broad.

Another observation in connection with this point is that in case of the reversal of a negative plate the brown color always appears first at the bottom. In this experiment it is, of course, necessary to make the cross-section of the electrolyte equal to the surface of the electrode, so that a straying of the lines of current is excluded.

It might be said that the behavior of these negatives is due to the better conductivity of the acid at the bottom than at the top, and that the current density must be higher at the lower parts of the plates than at the upper ones, on account of the smaller resistance of the electrolyte at the bottom.

One has to consider, however, that the difference of concentration of the free acid between the plates at the bottom and top of the cell establishes itself very slowly, and that is very small even with high cells, so that the increase of conductivity, due to the higher concentration at the bottom, is practically negligible, in view of the very flat maximum of the conductivity curve of sulphuric acid. The concentration is rendered again uniform, moreover, by the circulation, due to gasing during charge and due to vibration in automobile batteries.

In the following experiment, the arrangement of which is very similar to that of Fig. 1, the influence of different concentrations and conductivity of the electrolyte is eliminated. In a lengthy celluloid box two bright sheets were inserted, as shown in Fig. 6; the positive one being a copper plate of 1 mm.

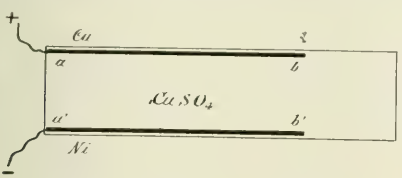


FIG. 6.—ARRANGEMENT OF EXPERIMENT.

and the negative one a nickel plate of ½ mm. thickness. The surfaces with which they were placed against the walls were provided with a coating of insulating paint. Copper sulphate solution was used as electrolyte.

After a current of 5 amps. had been passed through the cell for 24 hours, copper being deposited upon the nickel sheet, the paint was removed and strips about 3 mm. broad were cut off from the four ends of the plates, since small growths like warts had appeared at the rims of the nickel plate. After that strips, 5 mm. broad, were cut off at *a* and *a'* and at *b* and *b'*, and the changes of weight were determined. It was found that the copper strip *b* was lighter by 1.98 gram than the copper strip *a*, and that the nickel strip *b'* was heavier by 1.67 gram than the nickel strip *a'*. If the "electrolytic shunt" had not influenced the current density at the electrodes it is evident that the opposite effect should have been observed, on account of the voltage drop in the electrodes, the current entering at *a*.

The fact that storage battery plates behave differently at the top than at the bottom does not seem to have found much attention in storage battery practice. Very little has apparently been published about it in literature. All I could find in this respect was a note in the book of Dr. P. Schoop, "Die Sekundär Elemente," 2d part, p. 34. It is said there to be peculiar that manufacturers pay so little attention to this matter. "To convince one's self of the defective distribution of current over the surface of plates, for instance, of an older cell of the well-known firm of Pollak & Co., it suffices to observe it carefully during charge in a glass vessel. When gas begins to be set free towards the end of the charge, the gasing first starts only at the upper half of the plates, and only later on the lower half also begins to give off gas when the development of gas has already become strong, whereby the potential in the upper parts has increased. The investigation of English accumulators by Prof. Ayrton, in London, confirms this observation. He found that the active material in the lower half of the plate was always harder than that in the upper parts, on account of a larger content of lead sulphate."

Whoever has had to do in practice with the formation of Plante plates knows how difficult it is to get a homogeneous layer of lead peroxide, especially on large plates, and how easily the phenomena occur which are known in storage battery practice as buckling, etc. These phenomena may be caused by various circumstances, as we will see later on, but it is certain that they are mostly due to the non-uniformity in the current density over the electrodes, due to the "electrolytic shunt." It needs no further proof that a plate which operates with higher current density at the bottom than at the top undergoes larger changes of volume at the bottom, and that a plate which shows differences of change of volume at different points must have a tendency to mechanical distortion and buckling. (*To be concluded.*)

Notes on Electrochemistry and Metallurgy in Great Britain.

(*From Our Special Correspondent.*)

BLAST FURNACE GAS AND POWER DISTRIBUTION.

Concerning the "waste gases" from blast furnaces, the last Royal Commission on Coal Supplies in the United Kingdom reported that these were the only possible untapped source of fuel. It is, therefore, a matter of considerable interest to be able to record that Parliamentary sanction is now being sought to incorporate a company for the purpose of generating electricity and power gas in Cumberland. Lands for generating stations are scheduled at Maryport, Workington and Cleator Moor. The capital of the company will be £450,000, in £5 shares, with borrowing powers to the extent of £150,000.

The objects of the company are to supply electrical energy to authorized undertakers and persons requiring motive power supply, and current to be used for lighting the premises where power is supplied. The Broad of Trade will be referred to in the event of any authorized consumer withholding consent to the company supplying any customer in that area. Authorized undertakers will be required to enter into a seven years' contract, whilst, as in the other power bills, in the event of any dispute as to price between the company and a power user, the Board of Trade will be asked to appoint an arbitrator. Power gas consumers will be asked to enter into a contract for five years. Interest out of capital during construction will be paid at the rate of 3 per cent per annum to the extent of £20,000. Substantial commencement to be made with works within three years, and powers under the bill cease after seven years if a supply is not then given.

The charges for electric supply will be at the rate of 10s. per electrical horse-power, which the company is required to give, and a sliding scale from 3d. to ¾d. per unit, according to quantity. For gas the charge will be 10s. per indicated horse-power, with a sliding scale from 3d. to ¾d. per gas unit.

The only opponents to the scheme were the Whitehaven Corporation, the Workington Urban District Council and the West Cumberland Tramways Co. An agreement has been come to with the Whitehaven Corporation, however, whereby the power company acquires the existing municipal electricity undertaking in that town, the purchase price being £25,000, the capital expended upon it. This purchase is to be completed, in instalments, by March 31, 1910, or sooner if the power company desires. After the completion of the agreement, the company, which will then become the distributors in Whitehaven, is not to charge any higher prices than are now in force. If the purchase is not completed within the period stated in the agreement, all rights of the power company in Whitehaven are to cease.

In giving evidence on behalf of the bill, one of the promotors pointed out that it was proposed to erect three generating stations and to use the waste gases from the blast furnaces in the area, in order to drive gas engines for producing electrical energy. Agreements had been made with the owners of three of the greatest iron works for the supply of such gas for the purposes of the bill. It was estimated that 60,000 hp. were running to waste from the blast furnaces, but half this quantity would be sufficient for the purposes of the bill. It was also intended to supply power gas from the same sources. This was a very suitable gas for compression, and there was no difficulty in its distribution. The Home Office had reported upon the fact that there was no restriction as to the amount of carbon monoxide in the gas, and recommended that a provision limiting this to 14 per cent or such higher percentage as might be approved by the Home Secretary in the interests of the public health. There would be a greater percentage than 14 per cent, but the promoters were willing to place themselves in the hands either of the Home Office or the Board of Trade as to the percentage in the bill.

One can only hope that if Parliamentary sanction is secured the undertaking will be more successful than is so frequently the case with pioneer undertakings.

The Finances of Electrolytic Alkali and Bleach.

The annual report of the directors of the Castner-Kellner Alkali Co., Ltd., for the year ending March 31 last, states that the net profit (after allowances for keeping up works, plant and machinery in a state of efficiency) is £68,847.7.3, added to £12,797.16.11 from 1905, leaving an available balance of £62,548.5.10, and the directors recommend that £30,000 be appropriated to depreciation reserve, and that a dividend at the rate of 8 per cent be paid for six months to March 31 (making 6 per cent for the year), leaving £14,548.5.10 to be carried forward. The directors have increased a part of the company's plant, and have obtained suitable premises at Wallsend-on-Tyne, where the necessary electric power will be supplied on advantageous terms.

"Unsolved Problems in Metallurgy."

Taking the title of this paragraph as his subject, Mr. R. A. Hadfield delivered, on May 3 last, the fourteenth "James Forrest Lecture" at the Institution of Civil Engineers. For a day or so after the lecture I felt a little wonderment at the choice of the title, and questioned its appositeness to the lengthy subject matter which had interested a large audience. Point by point, Mr. Hadfield had reviewed the whole field of the metallurgy of iron, and described the latest achievements in each sphere. Led away perhaps by the wonder and magnitude of these achievements, I failed at the outset to grasp the unwritten moral of the whole lecture, and that was that finality was disclosed in none of the directions mentioned, and that in its ultimate sense all the many and complex problems were in a certain sense unsolved.

Unsolved, that is to say, because limits to improvements are unknown in regard to each of these, and because the explanations and theories of the physicist are only hypotheses and explanations based on present knowledge, and can take little account of future discoveries.

Mr. Hadfield's review naturally touched upon the influence of alloys, heat treatment, pyrometry, the future of electric furnaces, methods of testing and their value, micrography and so forth.

At the conclusion there was an adjournment to the reading room downstairs, where impact tests and analyses and pyrometers were shown in operation. The casual non-metallurgical engineer probably observed the microstructure of iron and steel for the first time under a microscope. The electrical engineer handled Heusler alloys, and then listened to brother civil engineers (whose early training had not embraced modern physics as concerned with magnetism), asking curious questions about an Ewing hysteresis tester.

All watched various liquid air experiments as affecting the magnetic properties and brittleness of certain alloys. When the audience had thinned down an opportunity presented itself of personally asking Mr. Hadfield whether liquid air treatment offered any immediate commercial possibilities. The reply was: "To-day, or this week, perhaps not. As to next year, one cannot say. All precedent is against prophecy in a negative sense. What is a scientific curiosity one year very often, by some sudden demand or improvement in a parallel field, becomes everyday practice next year. There are so many cases which might be cited as examples. Therefore, while to-day no demand or useful service arises, matters may be very different next year."

Market Quotations During May.

The chief feature of the month was the amazing price reached by tin. Commencing on May 1 at a shade over £183, the price mounted from day to day, and under the influences of a heavy home demand and a decline in the visible supply, reached £215 for cash and £205 for three months' delivery by May 14. The rising wave then receded almost as suddenly as it had risen, and fell to £185 within a week. A slight further fall took place, but this was followed by a recovery; finally tin closed at £185 per ton. At this price considerable attention is being given to Cornish mining properties, and several mines have been reopened.

Copper commenced by falling from £85 to £83.5 per ton. The closing price on May 30 was £85.17.6 cash. The present indications seem to point to a continuance of prices above the price of £80 per ton. As regards iron, Cleveland warrants have remained fairly steady, demand and supply balancing fairly closely. There was a rise from 50s. to 51s. in the middle of the month, with subsequent lower quotations at 49s. 11d. Lead has advanced from £16.2.6 to £16.7.6. Zinc sheets have risen to £32. Copper sulphate is fetching £25.15.

In the chemical trades very little changes have to be recorded. Ammonia sulphate is 2s. 6d easier at £12.2.6 per ton. The coal tar derivatives are fractionally cheaper, but the alkaline poducts are unchanged. Shellac has closed at £9.11 per cwt., and I hear of a virtual cornering in the supply.

The Van't Hoff-Raoult Formula.

The Van't Hoff-Raoult formula which is at the bottom of the whole modern theory of solutions, is the subject of a most interesting critical essay of Dr. Wilder D. Bancroft in the May issue of the *Journal of Physical Chemistry*. This paper, which should be studied carefully by all interested in the theory, brings out some very important points as summed up by the author as follows: The osmotic pressure varies with the heat of dilution. The abnormal molecular weights for sodium in mercury, sulphuric acid in water, resorcinol in alcohol, cupric chloride in water, and alcohol in benzene are due wholly or in part to the heats of dilution. The abnormal molecular weights for sodium chloride in water are not due to the heat of dilution. The molecular weights at infinite dilution are correct, but all others are in error to a greater or lesser extent. It is probable that a quantitative theory of concentrated solutions can be worked out if we make corrections for the heat of dilution.

This matter is of extreme interest in connection with the persistent and able fight of Dr. J. Kahlenberg against the "orthodox" electrolytic dissociation theory. As Dr. Bancroft points out, Kahlenberg has worked with finite solutions and has found that one must consider the specific nature of solvent and solute. For finite solutions the heat of dilution is a factor and varies with the specific nature of solvent and solute just as Kahlenberg has found. But for infinitely dilute solutions for which the orthodox electrolytic dissociation theory has been mathematically formulated the heat of dilution becomes negligible, and the results are independent of the nature of solvent and solute within certain limits.

An Electric Furnace for Heating Crucibles.

BY OLIVER P. WATTS, PH. D.

Accounts of the important types of the electric furnace used commercially are of frequent occurrence in the technical and scientific journals, but published data concerning the construction and operation of laboratory furnaces that have proved successful for particular purposes is as yet not abundant. It is the purpose of this paper to describe a type of experimental furnace used in melting on a considerable scale moderately refractory metals, such as iron and chromium.

In an investigation upon the production and properties of electrolytic iron and iron alloys, now being pursued at the University of Wisconsin, under a grant from the Carnegie Institution, it became necessary to secure a means of melting iron with the introduction of a minimum of impurity. From the high temperature required, some form of electric furnace seemed most suitable, and after several months of experimenting with both arc and resistance types, a very simple form of resistance furnace was adopted, and has now been in successful use for several months.

It consists of a rectangular brick box, into which the crucibles are set, crushed carbon shoveled in and packed around them, and the current turned on until the desired temperature is attained. The outside dimensions are: Length, 54 inches; width, 28 inches; height, 26 inches. The height includes a cellular base consisting of two courses of fire-brick set on edge.

As the furnace rests upon a cement floor laid on the ground, this construction affords ample protection, but if the furnace were set upon a table, in any part of which wood were used, it would be necessary to place a solid layer of brick between the two parts of the cellular base, to check radiation to the table.

The inside dimensions are: Length, 40 inches; width, 12½ inches; depth, 9 inches. The capacity of the furnace is 27 crucibles, each 3 inches in diameter and 7 inches high.

The material used in its construction is fire-brick, with a lining of a single layer of magnesite brick. This lining is necessary, as fire-brick is attacked by hot carbon, and is also melted at the temperature usually attained in this furnace. No cement was used in laying the brick, and none is necessary, but it would be advisable where a permanent furnace is desired, to put iron straps around the outside to prevent spread-

FIG. 1.—VIEW OF ELECTRIC CRUCIBLE FURNACE FOR MELTING PURE IRON.

ing of the side walls under the influence of repeated expansion and contraction. This will be done when it becomes necessary to rebuild this furnace.

The method of heating is similar to that of the Acheson graphite furnace, in which the articles to be heated are bedded in a granular resistor with their lengths at right angles to the heating current, but with the difference that for the present this case the crucibles—are placed in a vertical instead of a horizontal position.

Various crucibles were tried, the most satisfactory being that bored out in a lathe from a cylinder of Acheson graphite, and lined with magnesia. The crucibles to [illegible] in transverse rows of three, those of each row being [illegible] and set away from the sides of the furnace [illegible] leaves a space of 1½ inches between the [illegible] may be reduced to ¾ inch and still give good results.

In charging the furnace a layer of the resistor is put in to the depth of an inch, the crucibles set in the [illegible] and each one covered by a disc of magnesia, and above this a section of graphite 2½ inches thick—the latter to prevent contact of the granular resistor with the [illegible] of carbon have a most unpleasant way of eating holes in mag-

FIG. 2.—STARTING DEVICE.

nesia linings. More of the resistor is now added, and when at a depth of 3 inches it is tamped down at the sides and between the rows of crucibles.

Particular care is taken to tamp the resistor well where it makes contact with the terminals. These consist of bars of graphite 14 × 1 × 3 inches, to carry the current through the furnace wall, and a short bar of the same material set on edge inside the furnace and mortised to the first bar. The remainder of the resistor is added to a depth of ½ inch or more above the tops of the crucibles, a cover of magnesia brick, or better, of thin slabs of magnesia cement, is laid on, and above this a layer of fire-brick. At high temperatures magnesia brick conducts heat much better than fire-brick, hence the value of a layer of the latter as a top to the furnace. Several sheets of asbestos paper laid on top diminish the heat loss. Fig. 1 shows the furnace in operation.

Any furnace with a granular carbon resistor requires for convenient and economical control a variation in voltage of at least 100 per cent. This furnace is operated from a direct-current dynamo, giving from 95 to 125 volts, but because of interference with other work, it must often be operated at the constant voltage of 110.

For this reason a special device is used in starting the furnace. This is shown in Fig. 2. A slab of graphite [illegible] thick and 6 inches high, extends across the middle of the furnace, and serves as an auxiliary terminal. Electrical connection is made with this by a bar of graphite 3 × 1 inches in sections, as shown in the illustration.

At the start the two halves of the resistor are connected in parallel; when, through heating the resistance has diminished sufficiently, they are connected in series and the graphite bar, which carried current into the middle of the furnace, is removed. The slab of graphite, of course, remains in place.

A typical run with this furnace is as follows:

Time. Min.	*Amps.*	*Volts.*	*Kw.*	*Resistance Ohms.*
0	150	125	18.7	0.83
20	300	120	30	0.40
40	400	120	48	0.30
47	600	120	72	0.20
63	600	110	66	0.18
80	630	105	69	0.16
85	500	76	38	0.15
100	550	74	40.7	0.13
Current interrupted for 7 minutes in changing from parallel to series.				
107	200	120	24	0.60
130	250	120	30	0.48
170	300	116	34.8	0.38
220	400	120	48	0.30
250	450	120	54	0.26
265	500	117	58.5	0.23
280	500	111	55.5	0.22
310	510	110	56.1	0.21
340	500	105	52	0.21

Time, 5 hours 33 minutes; kw-hours, 248; average kw., 45; average watts per cubic inch, 10; ohms per inch cube at end of run, 0.538. The heat was sufficient to melt the purest iron obtainable.

The resistor consisted of a mixture of coke, carbon electrodes and graphite. These materials were run through a crusher, and all pieces larger than 5-mesh were removed. At the outset the finest material was also rejected, but later, contrary to orthodox practice, all except the coarse material was retained. To raise the resistance, carborundum was added, with good results.

By thus mixing materials of different resistivities, it is possible to adapt the resistor to any reasonable voltage and form of furnace that may be desired. In this case it is advisable, though not absolutely necessary, to screen the different materials to a fairly uniform size.

Furnaces of this kind have been built for crucibles of 2, 3 and 4 inches diameter, and varying in capacity from two to eight crucibles set in a single row.

The data of a run with one of these smaller furnaces follows:

Inside dimensions, 33 x 4½ x 6 inches.

Capacity 8 crucibles, 3 inches in diameter and 5½ inches high.

Time. Min.	*Amps.*	*Volts.*	*Kw.*	*Resistance in Ohms.*
0	150	107	16	0.71
20	200	110	22	0.55
21	300	110	33	0.37
24	350	110	38.5	0.31*
27	400	110	44	0.27*
35	370	107	40	0.29*
40	430	107	46	0.25
50	480	107	51.4	0.22
65	380	107	40.7	0.28
72	480	80	38.4	0.17
90	440	83	36.5	0.19
110	0	0		

Kw-hours, 66.6; average energy, 33.5 kw.; average watts per cubic inch, 37.6; ohms per inch cube at end of run, 0.173.

This furnace was designed to be operated from a 110-volt circuit, and, as shown above, it was not necessary to throw any resistance into the circuit except for the last 38 minutes. This result was secured by the great length in proportion to cross-section of the resistor, and by the liberal addition of carborundum.

In order that the furnace might start at 110 volts, a charn

* Starting rods are removed.

of three carbon rods, ¼ x ⅜-inch in section, was buried in the resistor, and withdrawn only when the latter had become heated sufficiently to take the desired amount of energy. Such use of small carbon rods for regulating the current in granular carbon resistors has been found very convenient.

In operating the large furnace the crucibles were allowed to cool in the furnace, but in some of the smaller furnaces they were not completely buried in the resistor, and were withdrawn for pouring the metal. When the crucibles are removed, the resistor falls into the holes, so that freshly charged crucibles cannot well be set in. As an intermittent furnace has served the purpose, no particular endeavors have been made to render it suitable for continuous operation.

To prevent the removal and introduction of crucibles while the furnace is in operation, FitzGerald (Electrochemical and Metallurgical Industry, February, 1905) has recommended the construction of the resistor of silico-carbides. The writer has tried no experiments for this purpose, but would suggest an outer container of Acheson graphite bedded permanently in a carbon resistor, the crucibles setting loosely in this container. As graphite is such an excellent conductor of heat at high temperatures the added thickness of this material between the resistor and the charge within the crucibles would have but a slight effect upon the operation of the furnace, the only serious consideration being the extra heat needed to bring this additional material to the working temperature of the furnace.

Crucibles of Acheson graphite, 4 inches in diameter, are as readily heated in these furnaces as those of 3 inches diameter, but with the increase in size there is a greater proportion of failures of the magnesia linings.

In the paper by FitzGerald, already referred to, concerning the use of crucibles of Acheson graphite in a granular carbon resistor, he says: "Here we have, unfortunately, a material that is an excellent conductor, for its resistivity is only 0.0008, so that it short circuits the current at the part of the resistor where it lies. In order to get the most satisfactory heating of crucibles of this sort it is necessary to use currents of great amperage, so that a heating effect is obtained by the passage of the current through the walls of the crucible."

That satisfactory heating of crucibles of Acheson graphite can be secured without depending upon the passage of the current through the walls of the crucibles, as the source of heat is conclusively proved by the operation of the furnace described above. In the largest furnace, the area of graphite in crucibles and covers cut by a transverse plane through the furnace varies from 9 to 13 square inches. Five hundred amperes is sufficient to heat this furnace, yet a current 25 per cent greater than this fails to heat to redness in the air a cylinder of Acheson graphite 1½ inches in diameter, having a cross-section of only 1¾ square inches.

It is apparent that the heating effect must be sought elsewhere than in the passage of the current through the crucible walls, and must be mainly due to its passage through those portions of the granular resistor which lie between the rows of crucibles, aided more or less by a high contact resistance between resistor and crucibles.

The high electrical conductivity of Acheson graphite at elevated temperatures indicates a correspondingly great thermal conductivity, and this latter property no doubt contributes greatly to the successful operation of the furnace, the crucibles acting as equalizers of temperature as well as conductors of heat from the resistor to the crucible linings.

At first thought the burying of a crucible in a carbon resistor might seem a most unpromising method of melting metals that one desires to keep free from carbon, but experiment shows that iron and many alloys may be so melted and contain only 0.01 to 0.05 of 1 per cent carbon.

It has been found that the addition of aluminium, magnesium, or calcium, metals which are capable of reducing carbon monoxid at high temperature, increases the carbon content to several tenths of a per cent. If added in large quantity they would probably introduce considerable amounts

of carbon by such reducing action. A similar reaction will probably occur in the arc furnace.

Among the advantages of this resistance furnace for crucibles may be mentioned:

I. Low cost and simplicity of construction.

II. The possibility of designing the furnace to heat any desired number of crucibles simultaneously.

III. By adapting the size of furnace to the amount of energy available, any desired temperature may be attained up to that of the destruction of the container for the resistor. Where linings must be used in the crucibles, their melting or vaporizing fixes the maximum temperature at which the furnace may be successfully used.

IV. The crucibles are buried in the source of heat, and hence are heated more rapidly, effectively, and probably more uniformly than in some other furnaces; *e. g.*, in the arc furnace, where the source of heat is on one side only.

V. For heating a half dozen crucibles at once it has proved more economical of energy than the arc furnace.

Some of its disadvantages are:

I. The need of a variable voltage for the most satisfactory operation.

II. It is, so far as developed by the writer, an intermittent furnace, and hence wasteful of energy.

III. It contains a large mass of material other than the crucibles and charges, viz.: the resistor, which must be heated to the maximum temperature attained.

Laboratory of Applied Electrochemistry, University of Wisconsin.

Faraday Society.

The twenty-first ordinary meeting of the Faraday Society was held on June 12, 1906. Mr. W. Murray Morrison being in the chair.

Zinc Deposition and Rotating Cathodes.

A paper on *The Electrolytic Deposition of Zinc, Using Rotating Electrodes*, by Dr. T. Slater Price and Mr. G. H. B. Judge, was communicated by Dr. Mollwo Perkin.

An improved form of apparatus for the electrolytic deposition of metals using a rotating cathode, is described. The ordinary beaker is replaced by a tap funnel of about 100 c.c. capacity, so that the electrolyte can be run off at the end of the experiment, thus obviating the use of a siphon. Use is made of a bicycle hub for rotating the cathode.

In the deposition of zinc from a pure solution of the sulphate, no extraneous electrolyte being added with a current of 2 amps. (cylindrical) cathode of gauze, 22 mm. in diameter and 40 mm. long), and voltage 4.5-5.5, the results are uniformly low. The error increases with higher currents, and is due to the heating effect of the current. On keeping the electrolyte cool by means of an ice jacket good results are obtained, even in the presence of free sulpuhric acid, provided that the concentration of the latter does not exceed a certain amount (about one-sixth normal).

It is further shown that zinc can be quantitatively deposited from solutions of zinc sulphate, containing 0.2274 grm zinc in the presence of 2–3 grms. sodium sulphate. The deposit is better than that obtained by the usual method, using sodium acetate and acetic acid; it is not necessary to cool the electrolyte.

An improvement of the method of Kolloch and Smith (*Journ. Amer. Chem. Soc.*, 1905, **27**, 1,255), using a rotating anode and mercury cathode is also described. It is found necessary not to have present more than two drops (40 drops = 3 c.c.) of free sulphuric acid in order to obtain quantitative results in the estimation of zinc.

A paper by Dr. F. Mollwo Perkin described *A Simple Form of Rotating Cathode for Electrochemical Analysis.*

The cathode consists of a spiral of platinum wire or, better, iridio platinum wire. Nickel wire may be employed in place of platinum, and the author recommends its employment in the case of the more expensive metal. Attention is drawn to the insolubility of platinum anodes, with heavy current densities, gold being dissolved in a cyanide solution in 35 minutes.

Electrolysis of Thiocyanate Solutions.

Mr. S. Binning and Dr. F. Mollwo Perkin presented a paper on *The Electrolysis of Solutions of Thiocyanates in Pyridine and in Acetone.*

On oxidation of thiocyanates with chlorine, persulphates, etc., a yellow coloring matter, canarine, is obtained. Electrolysis of aqueous acidified solutions of thiocyanates an apparently similar product, which was originally described in 1884 by Lurdow, is obtained. The authors consider that the substance is not identical with the canarine obtained by chemical means, because it shows certain reactions not given by the oxidation product. They find that in alkaline solutions no coloring matter is produced, but that cyanides are apparently formed. A difficulty met with in the electrolysis was the formation of the coloring matter on the anode surface, which caused a great resistance to the passage of the current. The difficulty was got over by using a lead cathode on which bristles had been fixed. This was rapidly rotated within a cylindrical platinum anode, the bristles cleaning the anode surface as the cathode rotated. With a current of 5 amps. (10 amps. per square decimeter) the e. m. f. was 5 volts.

Anhydrous ammonium thiocyanate readily dissolves in dry acetone and pyridine, in either case there being no difficulty in obtaining a 20 per cent solution. These solutions readily conduct the electric current. In the case of the acetone solution with a current of 1.5 amps. (3 amps. per square decimeter) the e. m. f. required is 6.7 volts. With pyridine for the same current it is about 9 volts. In the case of the acetone solution a yellow amorphous powder separates out. With the pyridine solution the liquid becomes orange red, but no precipitate is produced. On pouring this orange solution into a large excess of water an orange precipitate falls out. These two substances are certainly not identical with the canarine obtained by oxidation; at present the authors are unable to say whether the product obtained from acetone is the same as that produced from pyridine. It is found that when a lead anode is employed in the pyridine solution it goes into solution; lead thiocyanate is produced and metallic lead is deposited at the cathode.

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

Electric Furnaces.

Regulation of a Group of Electrodes in an Electric Furnace. W. K. Gibboney, 821,930, May 29. (Assigned to the Pittsburg Reduction Co.) Application filed Nov. 14, 1904.

The object is to regulate independently and automatically a group of electrodes in an electric furnace, so as to maintain constant and uniform conditions. The inventor provides each electrode with a regulating electromagnetic device which is operated automatically by a current derived from the current passing through the electrode, and he opposes the action of this derived current by an independent electromagnetic force. By varying the strength of this opposing electromagnetic force by appropriate means, the current supplied to the electrode can be varied as desired. The electrical details of the arrangement are described and diagrammatically shown.

Combined Non-Regenerative Reverberatory Furnace and Electric Furnace.—E. A. Touceda, 823,560 and 823,561, June 19. Application filed Jan. 28, 1905.

In non-regenerative reverberatory furnaces as used, for instance, for preparing iron for malleable castings, a very large

percentage of the heat units produced escapes through the stack and is wasted. To utilize them the inventor provides a steam boiler between the hearth chamber and the stack. This boiler works in connection with a steam engine, driving a dynamo. The electric current thus generated is supplied to two electrodes in the hearth chamber.

Compound of Carbon and Silicon for Incandescent Lamp Filaments.—W. G. Clark, 821,017, May 22. Application filed Jan. 24, 1900.

A non-porous and non-crystalline compound of carbon and silicon, claimed to be suitable for incandescent lamp filaments and harder and more durable than carbon filaments, is produced by decomposing a carbonaceous gas in the presence of a volatile silicon compound; for instance, by passing an electric current through a carbon filament in the presence of a mixture of hydrocarbon gas and the vapor of silicon tetrachloride. The compound is different from carborundum; it is formed at a temperature lower than the fusing point of carborundum, and is a conductor of electricity when cold. Very little exact information is given. The inventor is uncertain whether it is a chemical compound or an intimate mechanical mixture.

Carbon Holder for Electric Furnaces.—G. O. Seward, 824,153, June 26. Application filed May 3, 1904. (Assigned to Willson Aluminum Co.)

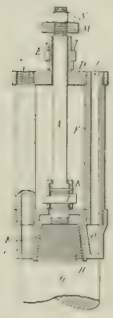


FIG. 1.—CARBON HOLDER.

Mechanical details of construction of the carbon holders fro mwhich the electrodes are suspended and through which the connection of the electrode with the line is made. The carbon holder comprises a suspending rod A (Fig 1) which is connected to the electrode G by a contact member B, and a connecting device for carrying the current to the electrode. The connecting device comprises a ring C, surrounding the contact member B, a guide D surrounding the suspending rod A, and having an upward projection to which is connected the line cable E, and hollow members F, connecting the ring C with the guide D. The electrode G is supported within the contact member B. The latter is in the form of a metallic bushing or cap or head formed with tapered screw-threads, corresponding with similar screw-threads H formed on the top of the electrode. B is connected with A by the rod J screwed into the top of B and fitting within a ring K, carried on the lower end of the suspending rod.

Titanium-Steel. A. J. Rossi, 822,305, June 5. Application filed May 20, 1902.

The inventor whose pioneer work for titanium is well recognized patents the following electric furnace process for making steel containing between 1 and 5 per cent of titanium and a desired amount of carbon. He feeds into the furnace an ascertained quantity of molten cast iron having an ascertained content of carbon, and then, without the aid of an air blast, he introduces into the molten cast iron a predetermined quantity of granulated oxides of titanium, calculated to be sufficient to decarburize the iron to the required extent. By regulating the current, he then increases the temperature to 3,200° or 3,500° F., for the reduction of the oxide of iron and also of the titanic acid by the carbon of the pig iron. This results in the decarburizing of the iron while the freed titanium is simultaneously alloyed with the iron. (The peculiar feature of the process is that the inventor introduces the titanium into the steel not in form of ferro-titanium, the employment of a ferro-alloy has the advantage that no particularly high temperature is required in the steel furnace itself, while Mr. Rossi must carry his whole steel bath to the reduction temperature of titanium oxide. Though his process has other advantages.)

ELECTROLYTIC PROCESSES.

Electrolytic Water Purifier.—A. E. Dieterich, 823,671, June 19. Application filed April 10, 1906.

A drum casing contains a double spiral screw of aluminium serving as electrodes. They form a spiral passage from one end of the casing to the other. While the water passes through it is subjected to electrolysis. The screw is rotated at times, in order to eject the precipitate from the apparatus without removing the electrodes.

Recovering Precious Metals from Water.—C. E. Holland, 818,174, April 17. Application filed June 5, 1905.

The object is to recover the precious metals from running streams in the vicinity of gold and silver mines. The inventor estimates that in the majority of cases stamping mills, combined with amalgamating and concentrating systems, lose from 20 to 50 per cent of their gold in this manner, this amount going to waste in the streams. The inventor conducts the stream of water through an electrolytic tank with aluminium or iron electrodes. Thereby the organic and inorganic matter in suspension or solution is coagulated or precipitated, the coagulations and precipitated matter containing practically all the gold and silver. This product is conducted from the tank to drying tables, tanks, pans, retorts or similar appliances, where it is subjected to a process of evaporation, decantation or filtration, the gold and silver remaining in powdered or granulated form in the residue. The precious metals are then recovered by treatment with a cyanide solution or in some other way.

Electrolytic Production of Oxygen and Hydrogen.—K. J. Vareille, 823,650, June 19. Application filed Nov. 15, 1905.

The object is to have the electrodes of opposite polarity near together, in order to decrease resistance and energy consumption, and to prevent at the same time the oxygen and hydrogen from mixing. The construction is shown in Fig. 2, where 13 are the electrodes connected to the rods 14, with the terminals 16 (positive) and 17 (negative) at the top. The electrodes 13 are of corrugated sheet iron. They are separated from each other by means of troughs 20 provided with insulators 18. Through the funnels 22 and pipes 23 the cell is filled with electrolyte. The oxygen gas is collected in a receiver in the center, the hydrogen gas in two receivers at the left and right.

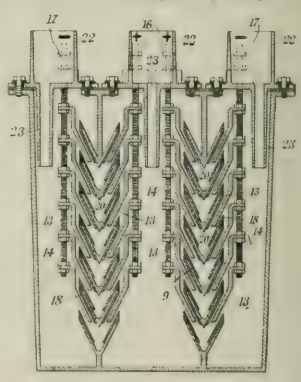


FIG. 2.—PRODUCTION OF OXYGEN AND HYDROGEN.

BATTERIES.

Alkaline Storage Battery.—T. A. Edison, 821,623, 821,624, 821,625, 821,626, 821,627, 821,628, May 29. Applications filed at various dates between Nov. 2, 1904, and May 20, 1905.

821,623 refers to a battery-filling apparatus so deviced that a positive notification is given when the electrolyte reaches the proper level. For this reason an electrode is provided at the proper height, which closes a circuit when the electrolyte rises to the predetermined level. 821,624 refers to a modification of his former construction of the gas separator, in which he used a perforated diaphragm or gauze in the top of the separator casing, through which the gases pass in a diffused and attenuated condition, in order to prevent the possibility of an ex-

plosion within the receptacle. Edison now provides the separator casing with a cover or auxiliary valve, which is normally closed to prevent the admission of any dust or dirt, but which will open by the accumulation of pressure beneath it to provide for the escape of the gases into the atmosphere. 821,625 refers to the removal of acid radicals, carbonates, organic matter and other soluble impurities from the active materials. These impurities are due to impurities in the chemicals or water used and are highly obnoxious. To remove them, Edison subjects the active masses to a treatment in a hot caustic solution with the concurrent generation of large quantities of hydrogen gas in situ within the active masses, the heating being effected, preferably electrically, by passing a discharging current through the active masses. The last three patents refer to three different methods of making cobalt films or flakes for admixture with the particle of active mass of nickel hydroxide. In 821,626 Edison deposits by electrolysis a film of zinc on a copper cathode, and then a thin film of cobalt on the zinc. By treatment with dilute acid the zinc is dissolved and the insoluble cobalt film thereby separated. The process of 821,627 consists in subliming cobalt chloride and deposit it in form of scale like or crystalline flakes, then reducing the chloride flakes to the hydroxide state, and finally reducing the hydroxide to the metallic state. In 821,628, Edison fuses a sulphur compound (sesquisulphide) of cobalt and cools the fused mass, which results in the crystallization of the metallic sulphur compound in thin scales or [illegible] screening, non-scaly crystals are eliminated. Finally, the scales are roasted and oxidized and then reduced to the metallic state.

Storage Battery.—G. A. Ford, 824,348, June 26. Application filed July 20.

Details of construction, the character of which is best indicated by claim 9: "in a storage battery, the combination of a supporting pan or tray of non-conducting material, an electrode plate within said pan having a central tapering web, with vanes between it and the edges of said plate, said vanes lying at an angle to the plane of said plate, and an auxiliary electrode plate having a tapering web coextensive with said other tapering web and provided with fins between said vanes."

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

ELECTROCHEMISTRY.

Electric Copper Smelting.—In *The Mexican Investor* of March 17, A. Bosch gives an interesting comparison of the relative cost of electric smelting and ordinary blast-furnace smelting of silicious gold-silver ores, with copper as a collector for the values. The comparison is made for a 400-ton plant on the basis of cost of coke of $12.00 per ton and $4.50 for the electric horse-power-year. Under these circumstances the author finds that the electric process allows a saving in fuel cost of about 43 per cent of the cost of the blast-furnace process. For a 400-ton plant, about 27,500-hp. water-power must be developed. The author recommends to develop about 10,000 hp. more selling electric power in the neighborhood. The total net profit of the combined power plant from the sale of electric energy at $80.00 per horse-power-year would then be about 7.9 per cent interest on the invested capital. While this might be considered too low, it should not be forgotten that the chief business is the smelting process, and that the distribution of electric power to mining regions by the smelting company would stimulate further mining, aiding indirectly to the prosperity of the smelter. As the final result of the comparison, the "total smelting cost" per ton of material treated is given as $2.48 with coke-fuel and $1.46 with electric current, representing a saving of $1.02. The pre-eminent advantage in electric copper smelting is the greater rapidity of fusing the charge, due to a higher temperature at the smelting zone. This results in a larger output and also permits to change the ordinary silicious slag into a higher silica-lime slag, without endangering the necessary fluidity of the slag. In a note attention is called to the availability of the great falls of Juanacatlan, near Guadalajara for such electric smelting developments. "The industrial interests of a country around a great waterfall are far more important than the profits brought in by tourists to simply see the waterfall. There is no comparison to be made between these interests." Mr. Bosch adds that "to see a mass of water falling is a very childish pleasure, and those people advocating the preservation of great waterfalls should compare the labor conditions in coal centers and those around hydraulic power centers."

Disinfection by Hypochlorites in England.—Concerning the new disinfecting plant of the Metropolitan Borough of Poplar, the following information is given in the London *Electrician* of June 1. The apparatus which deals with 185 gallons per day of 8 hours, "takes up a surprisingly small amount of space." The electrolyzers are arranged in four tiers or terraces. The electrolyte, which consists of 20 parts of a saturated solution of magnesium chloride with 100 parts of a saturated solution of sodium chloride, flows into the first set of cells at the top of the terrace at the rate of 3⅛ pints per minute, and here the solution attains a strength of about 1 gram of available chlorine per liter. It then flows over into the second cell, where the amount of available chlorine is about doubled, and so on to the third and fourth, when it leaves the apparatus, carrying about 4 grams of active chlorine per liter. Each cell contains zinc cathodes and platinum anodes (consisting of thin platinum wire wound over flat slate surfaces). Great difficulty was first experienced in getting a stable solution, it being found that within a few hours the oxidizing properties were very much reduced. This difficulty has now been gotten over by adding small quantities of magnesia to the electrolyte as it leaves the electrolyzing cells, thus neutralizing the free hypochlorous acid. By this treatment the solution when kept in closed carboys or bottles keeps its strength for weeks and even months. The four rows of cells contain forty electrodes and are connected in series, which are worked with a current of 15 amps. at 240 volts. A certain amount of the current energy is dissipated in heat, the solution entering the apparatus at 60° F. and leaving it at about 90° F. It is not possible to obtain solutions of much greater strength than 4 grams per liter, without using some external method of cooling. As, however, the solution is always diluted before being used for disinfecting purposes, there is no necessity to increase the amount of active chlorine in it. From time to time it is necessary to clean the electrodes which become coated with magnesium oxychloride. The formation of this oxychloride, although to a certain extent a disadvantage, is [illegible] because it helps to prevent the cathodic reduction of the hypochlorite. When preparing sodium hypochlorite it is [illegible] advantageous to add small quantities of a magnesium or calcium salt in order to minimize reduction. The Poplar Borough Council distribute the disinfectant fluid in bottles, and use it for watering the streets. "It most certainly has very powerful deodorizing properties, the extremely offensive odor from a bucket of very rotten fish was almost immediately suppressed

by pouring on about 1 pint of the fluid. Whereas, 1 pint of carbolic fluid costs 17 cents, the hypochlorite fluid (which concentration?) with electric current at 3 cents per kw-hour, only costs 1 cent." A fuller description with illustrations may be found in London *Electrical Review* of June 8.

Electric Annealing Furnace.—In treating metals, especially in hardening tool steels, it is most important for various purposes to heat them uniformly for a certain time at a distinct temperature. Any lack in uniformity of heating metallic pieces (especially with pieces of irregular cross-sections) results in flaws. In *Elekt. Bahnen* of May 14, a note is published on an electric furnace of Koerting Bros., of Berlin, in which the metal pieces to be heated are placed in a fused salt bath, the alternating current for heating being introduced through two cast iron electrodes from the secondary of a transformer; by changing the number of secondary windings the temperature may be easily regulated. For a furnace of "medium size" it is stated that per c c of bath about 0.25 watts are required for a temperature of 750° C., 0.6 watts at 850°, 1.4 watts at 1,000°, and 3 watts at 1,300°. No further technical details are given.

Formation of Ozone.—To produce ozone by electric discharges of air it is necessary to employ the silent discharge, the formation of arcs being fatal. It has been suggested that the formation of ozone is ultimately due to the ultra-violet light of the brush discharge. In connection with this point some notes by G. A. Vosmaer (whose process of making ozone was described in our Vol. II., p. 511) in the London *Electrician* of June 8, are of interest. He found that ozonized air is a worse conductor of electricity than ordinary air. Since it seemed probable that a certain amount of ionization is produced by the brush discharge, the ozone generated was tested for its property of discharging a charged electroscope, but with entirely negative results. Against the hypothesis of formation of ozone being due not to the electrification, but to the ultra-violet light he mentions the fact that if one discharger is working to its full extent, its luminosity (and output in ozone) is greatly diminished by the proximity of another, so much so that if the second discharger is placed at a distance of less than an inch, both are nearly extinguished, and if there is a third the middle one entirely stops working. This is a very remarkable fact, and is one of a great many unexpected phenomena which take place in connection with the production of ozone and upon which the author is still conducting some researches. The author experimented with a 20-kw plant of his own design, in which the dischargers consisted of one sharp edged and one flat pole connected to a 100-period 10,000-volt transformer, special arrangements being adopted to prevent sparking across.

Flour Belaching with the Aid of Electric Discharges. In the London *Electrician* of June 8 the process of the Alsop Flour Process Company is described. A current of air is passed through a chamber containing a long, high-voltage continuous-current arc, and this air is then used for treating flour in an agitator. The result of this treatment is that the flour emerges from the agitator with a considerably whiter color. Two patterns of apparatus are employed. In the form shown in Fig. 1 a double-acting pump draws the air from the inlets H H through the chambers A A, which contain the arc, whence it passes through the pipes C C to the cylinder B of the pump. The pump piston rod G is driven by a pulley on the shaft F, and is connected through links to the rods E E, to which the upper electrodes of the arc are attached. On removing the covers, attached to the chamber A by means of the insulated handles, the arc is visible, and the inside of the chamber can be got at for cleaning and for replacing the electrodes. Current is led in by the wires D D D. The electrodes between which the arc is struck are usually of mild steel or iron, and are insulated from the walls of the chamber. They are provided with taper ends, which fit taper sockets, and, whilst being correctly held, are easy of replacement. The suction stroke of the pump starts when the electrodes are together, and thus the air flows through the chamber as the arc is being lengthened by the raising of the upper electrode, until the arc is extinguished. During the pressure part of the stroke, the upper electrode descends again. As the pump is double acting there is a continuous stream of air, first through the arc chamber on one side and then through a similar chamber on the other side. A simple clutch arrangement similar to an arc lamp clutch allows for the burning away of the electrodes. The stroke imparted to the rocking lever is slightly longer than the maximum distance between the electrodes, and when the latter come together the spring-controlled clutch releases the rod E for the remainder of the stroke. A generator voltage of 400 is employed, but, to increase the length of the arc, an inductance of considerable size is included in the circuit. The length of arc employed is from 2 inches to 5 inches, according to the size of the pump. The apparatus of the type shown by Fig. 1 is made in three sizes: 4-inch piston diameter by 4-inch stroke, 7 inches x 7 inches and 10 inches x 10 inches, and the speed is 50 r. p. m. With the largest sized pump, thirty sacks of flour can be treated per hour. There is in all cases a small resistance in series with the arc, which provides for an adjustment of current in addition to that obtained by altering the inductance in the circuit. The current employed ranges from 1½ to 12 amps. in the largest size according to the output of flour, or the bleach required. The second pattern described is designed for smaller outputs.

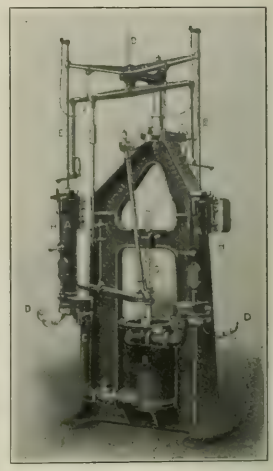


FIG. 1.—APPARATUS FOR PASSING DISCHARGES THROUGH AIR FOR BLEACHING FLOUR.

An American patent of the same company was analyzed on page 240 of our June issue. As to the rationale of this process it is to be remarked that in this case arc discharges are employed so that probably compounds of nitrogen and oxygen are formed in the air and the bleaching effect is due to these compounds. Essentially different from this method are those processes (proposed by others) in which ozone is used as bleaching agent and is produced by the silent electric discharge through air. S. Letham (our June issue, p. 240) claims that the best results are obtained by treating the air successively first by the silent electric discharge and then by an arc discharge. Methods of this kind for whitening flour are reported to be in commercial use to some extent in the Middle West and South, but no exact information is available.

Generation of Hydrogen from Calcium Hydride.—For preparing hydrogen gas (for filling balloons and other purposes) quite a number of different methods are in use or have been proposed. Among the latest ones is the use of electrolytic iron, suggested by Burgess, as noticed in our June issue, page 224. In the meanwhile an observation of Moissan has led in France to the industrial preparation of calcium hydride. It was shown by Moissan that metallic calcium absorbs hydrogen when heated, forming the colorless hydride CaH_2. This is now made industrially and sold under the name of hydrolyte. Some notes concerning it are given in a paper by Jaubert in *Comptes Rendus*, Vol. 142, p. 788, abstracted in *Am. Jour. of Science*, June. Calcium hydride is well adapted for generating

hydrogen, since it reacts with water according to the equation: $CaH_2 + 2H_2O = Ca(OH)_2 + 2H_2$. The reaction is thus quite analogous to the generation of acetylene from calcium carbide. One kilogram of the substance thus liberates a cubic meter of pure hydrogen. Metallic calcium is manufactured by the electrolysis of the fused chloride. The electrical energy required for producing 100 kilograms of the metal in 24 hours is about 20 volts and 7,500 amps., or 150 kilowatts. The hydride is prepared by heating the metal to a high temperature in horizontal retorts in which hydrogen circulates. The gas is slowly absorbed, and after several hours all the calcium is transformed into the hydride. The commercial product occurs in white or gray, irregular porous pieces, having considerable hardness. It is insoluble in the common organic solvents, but is instantly decomposed by cold water, similarly to calcium carbide. The product is about 90 per cent pure, the impurities being chiefly oxide and nitride.

Iron and Steel.

Iron and Steel Institute.

The annual spring meeting was held in London, May 10 to 12, President R. M. Hadfield in the chair. The Bessemer Gold Medal for 1906 was awarded to Floris Osmond, of Paris, and the Carnegie Gold Medal to L. Guillet, also of Paris, thus suitably recognizing the splendid work of the new French school. A special silver medal was awarded to W. Rosenhain, of Birmingham, and Carnegie scholarship grants to H. C. Boynton and E. Hess, in the United States. The following abstracts of the papers presented at this meeting we have prepared from the special number of the *Ironmonger* of May 12:

Compression of Steel Ingots in the Mould.—A. J. Capron, of Sheffield, described a process of fluid compression wherein the moulds are divided into two parts vertically, with a loose packing in the joint, and are compressed by horizontally applied pressure. Several moulds may be placed tandem, that is, in series, and the same pressure then acts on all. In this case all the moulds must be filled at the same time, either from a trough or a bottom gate. Within a few minutes after casting the ingots shrink from the mould, the loose packing pieces are removed and the pressure applied until the ingots are solid. Groups of small ingots of only 80 pounds weight each are treated with great success. The power required is comparatively small, 30 hp. being sufficient for an 8,000-ton press dealing with a cast of 60 tons of steel. In the discussion, Mr. D. V. Pirie claimed that the Hermé process of fluid compression is the best so far devised; with it, ingots of 25 to 30 tons can be cast absolutely solid from top to bottom without a blow-hole the size of a pin-head.

Cooling of Cast Iron.—Prof. Thomas Turner, of Birmingham, discussed the volume and temperature changes taking place during the solidification and cooling of cast iron. In his experiments a bar was cast in a sand mould in the shape of a T, which kept one end fixed while the other was free to contract or expand. A thermo-couple indicated the temperature. A wire nail is inserted into the free end, so that as the metal flows around it it becomes embedded, and its free end is in contact with an extensometer multiplying the movement forty times. Copper, aluminium, antimony lead, tin and zinc give a uniform contraction curve, without any arrests or kinks. White, washed American pig iron, containing only iron and carbon, gave a contraction and cooling curve having one marked arrest, at 665°, evidently coincident with the formation of pearlite. Gray hematite iron containing 3.47 silicon and 0.55 manganese showed a slight expansion at 1,135° just after setting, and an equally strong expansion at 695°. Phosphoric grey iron from Islip, Northampton, carrying 3.98 silicon, 0.50 manganese and 1.25 phosphorus, showed a very strong expansion at 1,080° just after setting, another small expansion at 900°, not observed in the other samples, and a very strong expansion at 730°, evidently corresponding to the arrests at 695° and 665° in the purer samples. In conclusion, the author pointed out that the tests can be readily [illegible] a foundryman [illegible] ing the casting qualities of [illegible]

Chilling of Cast Iron.—E. Adamson reported tests on the influence of silicon, phosphorus, etc., on chill casting. The conclusions drawn are that the depth of chill is primarily dependent upon the percentage of [illegible] temperature of casting. Total carbon and combined carbon are the definite factors, graphitic carbon rather a negative factor. Silicon reduces the amount of chill, but the amount of this reduction to be expected cannot be deduced from [illegible] of silicon alone, as found by analysis. Phosphorus reduces the true chill, but it rather increases the interlacing effect between the white and grey portions, and therefore is not altogether harmful. Manganese increases the depth of chill, and below 1.17 per cent does not interfere materially with the mechanical properties. Aluminium decreases the depth of chill.

Manganese-Iron Alloys.—Prof. Arnold and F. K. Knowles contributed a preliminary note on alloys of these two metals containing only 0.1 to 0.2 per cent of carbon and silicon; 13 to 20 per cent manganese alloys worked like tool-steel; 20 to 35 per cent behaved like hard tungsten steels. The general opinion expressed in the discussion was that the paper was incomplete and prematurely written.

Thin Steel Sheets.—Mr. E. F. Law discussed "Brittleness and Blisters in Thin Steel Sheets." Blistered sheets are usually, but not invariably, brittle; on the other hand, many brittle sheets show no signs of blisters. Heat treatment does not remove the brittleness, which is, moreover, not due to improper heat treatment in the first case. Microscopical examination showed that brittle and blistered sheets were invariably impure from segregation of sulphur phosphorus and apparently also iron oxide. Manganese sulphide and manganese carbide, rich in phosphorus, tend to roll out into thin lines and sheets, so that in thin, black plates their influence in making lines of weakness is far from negligible. The final conclusions were that oxidized steel is more liable to give blistered sheets, and that large ingots which segregate in cooling tend to give brittle sheets, because of the segregation of the impurities.

Copper in Steel.—F. H. Wigham reported results of numerous tests of the properties of steel containing copper. Difficulty was found in getting the copper alloyed; one method used was to put the copper into the steel as it ran into the moulds, and to add likewise up to 1 per cent of aluminium to facilitate the alloying. This method is open to several grave objections, such as the lack of time for alloying and the simultaneous effect of the aluminium. The results of the tests were that 0.6 per cent of copper is the limit allowable in steels containing 0.5 per cent or more of carbon, while 0.25 per cent of copper is no disadvantage at all in the manufacture of wire. Low carbon steel with up to 0.5 per cent of copper welded without difficulty and drew into wire perfectly.

Rolled Steel Wheels and Tires.—P. Eyerman, of Benoit, Wis., described the manufacture of solid rolled car wheels and tires, such as has been long known, and was described in America by Mr. Vauclain in the *Journal* of the Franklin Institute for February, 1905.

Chain Making Machinery.—E. Lelong described a machine which takes in the heated bar of iron and makes the links either interlocked or separate, making 100 to 120 links of 25 mm. chain per hour. It is claimed that the machine-made chain is 20 to 25 per cent stronger than chain made by hand from the same bar material.

Fracture of Steel.—C. O. Bannister, of London, discussed at length "The Relation Between Fracture and Micro-Structure of Steel Test Pieces." He distinguishes five types of fractures of test pieces: cup, laminated, irregular, crystalline and slip or oblique fractures, mechanical, chemical and microscopic data all very complete, accompanied by photographs and microphotographs are then given for six specimens of material

breaking with each kind of fracture. Cup fractures occur only in breaking circular bars of the best steels. Laminated fractures are associated with steels showing "ghost lines," which may often give excellent mechanical tests and satisfactory chemical analyses, but yet are not good steels because of their laminated nature, as shown up in the fracture. Irregular fractures are generally associated with inferior steels, and are always associated with slag lines or well-developed "ghost lines." These steels consist of irregular patches of pearlite, more or less surrounded by ferrite, along which the specimen has a tendency to break. The extended investigation of steels with crystalline or slip fractures is not yet completed. The frequent occurrence of manganese sulphide and silicate in commercial steels fully justifies their being classed as important constituents, and kept in view in examining the steel.

Copper.

Melting Point.—M. P. Dajean, in *Revue de Metallurgie* for May, redetermines the melting point of copper by the very simple method of using a Le Chatelier couple, and finding by comparison that "best selected" copper melts at 20° higher than pure gold. "Admitting" (as the French like to say) that gold melts at 1,065°, then copper melts at 1,085°, since 1065 + 20 = 1085, "as nobody can deny." When the copper contains Cu^2O, its melting point falls to a eutectic carrying about 5.0 per cent CU^2O, and having a melting point 25° below that of pure copper. The alloys of copper with aluminium show a regular reduction of the melting point at the rate of approximately 5 for each per cent of aluminium added. Dajean recommends using copper for standardizing thermo-couples, allowing it to oxidize freely, so as to get the eutectic melting point. It appears to us preferable to keep the copper deoxidized by means of charcoal, and thus avoid any uncertainty as to whether the eutectic has been reached or not.

Copper and Sulphur.—E. Heyn and O. Bauer describes in *Metallurgie* for Feb. 8, an extremely interesting and important research on the fusing points and eutectics of copper and cuprous sulphide. With pure copper melting at 1,085°, and cuprous sulphide at 1,127°, the diagram was completed for all admixtures of the two, 3.8 per cent of cuprous sulphide, representing 0.76 per cent of dissolved sulphur, forms a eutectic melting at 1,067°. This eutectic showed itself on the cooling curve from 1 up to 95 per cent Cu^2S. Above 3.8 per cent an upper melting poin: above the eutectic rises rapidly, up to 9 per cent Cu^2S, melting at 1,102°. From this point up to 85 per cent Cu^2S the melt consists of two inimiscible layers, one the eutectic of Cu^2S saturated with Cu, and melting at 1,102°, the other, the eutectic of Cu, saturated with Cu^2S, melting at 1,067°. Above 85 per cent of Cu^2S a new melting point appears, running up steeply to that of pure Cu^2S at 1,127°. The microscope showed clearly characteristic appearances for these eutectics and their component parts; hydrofluoric acid was found the best etching solution, since it turns cuprous oxide black, but leaves cuprous sulphide blue.

Special Brasses.—The indefatigable Guillet adorns *Revue de Metallurgie* for May with a 46-page treatise on special kinds of brass. He has studied carefully the copper-zinc, or ordinary brasses, and he then determines by careful comparative micrographic study how many per cent of zinc are replaced by each per cent of an added foreign element, such as aluminium, lead, phosphorus, etc. This ratio he calls the equivalence of the element added. The studies were made on brasses containing 59 to 67 per cent of copper. The results showed that 1 per cent of aluminium replaces 6 per cent of zinc, that the equivalence of aluminium may be called 6, and that a large percentage of aluminium about 10, is necessary to give any special characteristic to the alloy not possessed by ordinary brasses. Manganese has an equivalent coefficient of 0.5. Iron, up to 1.66 per cent, has an equivalent coefficient of 0.9. Tin has the coefficient of 1. Lead may go into solution up to 0.03 per cent, and then has a coefficient of about 1. Silicon dissolves to the extent of 1.4 per cent in brass of 63 per cent copper, and 2 per cent in brass with 90 per cent copper; its coefficient within these limits is 10. Magnesium dissolves only to the extent of 0.2 to 0.3 per cent, and has a coefficient of 2. The presence of more magnesium is extremely injurious to the mechanical properties of the brass. Antimony acts quite similarly to magnesium.

Nickel.

Nickel-Silicon Alloys.—Guertler and Tamman (*Z. für anorg. Chemie*, 1906, **49**, 93; *J. Soc. Chemical Industry*, May 15) have studied these alloys, and by the melting points have proved the existence of Ni^3Si, Ni^2Si, Ni^3Si^2, NiSi and $NiSi^2$. The softest alloys are stated to be those with 40 to 50 per cent of silicon. Those with up to 10 per cent of nickel can be worked easily when cold, but not at a red heat; they are softer when quenched than if cooled slowly from a high temperature.

Nickel-Antimony Alloys.—K. Lossew (*Z. für anorg. Chemie*, 1906, **49**, 58; *J. Soc. Chemical Industry*, May 15) found from the melting points only two compounds of these two metals, NiSb and Ni^2Sb^2. Starting from antimony, melting at 630°, the first eutectic point is at 611°, with between 2 and 3 per cent of nickel; it then rises to the first maximum, at 1,158°, corresponding to NiSb, with 32.83 per cent of nickel. From 32.83 to 54.97 per cent nickel mixed crystals of NiSb and Ni^2Sb^2 are produced; at 54.97 is a maximum of 1.170°. From Ni^2Sb^2 to 66 per cent nickel the melting point falls, the second eutectic, at 66 per cent, melting at 1,100°. From here on the melting point rises to that of pure nickel. In these alloys the compound Ni^4Sb forms at 677° C. if it is allowed to cool slowly in air; if quenched, this does not occur.

Slags.

Reaction Velocity and Viscosity.—C. Doelter, in the *Zeitschrift für Elektrochemie* of June 15 (this year), answers, as if by mental telegraphy, the query of Prof. Howe, contained in our correspondence column, page 258. Doelter divides silicate slags into two classes: such as possess when melted small viscosity and have a sharply defined melting point, and such as have greater viscosity and show no sharply defined melting point. The first class are simply constituted silicates, such as $CaSiO^3$, Mg^2SiO^4; the second are complex, such as $NaAlSiO^4$, $Na^2AlSi^2O^6$, $K^2Al^2Si^4O^{12}$. The latter can be considered as split up into their component simple silicates when melted, or into aluminates and silicates. These two classes are distinct, but there are transition silicates between them. The second class show while cooling an upper and a lower melting point, or point of arrest on the cooling curve, also a long period of solidification, and very low velocity of melting or crystallizing; the amount of undercooling observed is much larger, and they are much more inclined to form glassy slag than the first. The velocity of reactions in these slags is very small, because of their viscosity. In the first class, the speed of melting and crystallizing are very much greater, and chemical equilibrium intends to establish itself much quicker.

Of great importance is the difference between the two classes in respect to dissociation. The less viscous, simple silicates may in fact be dissociated, being partly resolved into their constituent oxides, aside from the question of content of free ions; as a consequence of decreased viscosity, the electrical conductivity of these slags increases rapidly as their temperature is raised. In every case, in these slags, ionization equilibrium is reached rapidly, and reaction velocity has a much higher value than in slags of the second class, and equilibrium is more rapidly reached.

In slags of the second class, thermolytic dissociation undoubtedly takes place; they are bodies representing a binary system. When melted they fall into their component silicates, and by long maintenance in the viscous condition other complexes than the original ones can be formed. Measurements of crystallization velocity and solution velocity in these complex silicates show only one-two hundredth the velocity found in the simple, less viscous slags.

There is, therefore, no doubt that such silicates of the second class are strongly resolved when melted, and that while cooling the former equilibrium is only very slowly restored, thus frequently causing the formation of a glass. Many such silicates, as Albite and Orthoclase, will on this account never be obtained crystallized from the melted state, unless additions of other bodies be made to accelerate the reaction velocity; small amounts of tungstic acid, vanadic acid or molybdic acid are in this respect active accelerators, and thus induce crystallization.

It has long been known to mineralogists that the simple silicates separate out first, and the complex silicates later, and that the quantitative relations relative to the eutectic mixture are not here the determining factors—as they are in alloys. The reasons for this are given by the low reaction velocity in complex melts.

The relations of silicates are therefore governed by the general reaction velocity, melting velocity, crystallization velocity, solution velocity, the velocity with which equilibrium is reached in the melted slag, and the particular reaction velocity of two reacting substances. These velocities are all, in their turn, dependent upon the *viscosity* of the slag, the reaction velocities being considerably greater with the thinner and least viscous varieties of slag.

RECENT METALLURGICAL PATENTS.

Lead.

Smelting Lead-Sulphide Ores.—Mr. A. G. Betts, the indefatigable worker in lead metallurgy, has recently patented (821,330. May 22) a process of smelting lead ore which does not require the admixture of silicious ores and in which the losses of lead are reduced. The smelting charge consists of 65 parts lead sulphide, 20 siderite or limonite, or the equivalent amount of other iron oxides, 20 sodium carbonate, which may be in part replaced by the equivalent amount of caustic soda, and 12 to 15 parts fine coal or coke. In general the mixture will contain some silica, blende, pyrite, limestone or other gangue of the lead-sulphide ore. The principal reactions which take place when the mixture is heated are:

$$Na_2CO_3 + 3C + PbS = Pb + Na_2S + 3CO.$$
$$Fe_2O_3 + 3C + 2PbS = 2Pb + 2FeS + 3CO.$$

The mixture is preferably smelted in an ordinary reverberatory smelting furnace. The fusion takes place at a moderate temperature, yielding nearly all the lead of the ore as metal and an easily-fusible iron-sodium matte. Loss by volatilization of lead scarcely takes place at all when the charge is covered with a little coal, or the flame in the furnace is reducing. The mixture can be smelted in a blast furnace, which should be run so that there is an excess of carbon monoxide in the escaping gases. Success is attained by proportioning the furnace charge so that the matte produced contains considerable proportions of both iron sulphate and sodium sulphide, for the reason that the mixed sulphides melt at a comparatively low temperature and the mixture does not hold in igneous solution appreciable quantities of metallic iron or lead, which is the case with iron sulphide alone. Sodium sulphide alone is difficulty fusible. The matte, leaving out of consideration silica, zinc, etc., best approximates in composition the formula Na_2S . FeS. As iron is a cheaper agent than soda, it is usually more economical to use more iron and less sodium than this formula requires Iron ores containing manganese are equally as good as pure iron ores, since manganese acts analogously to iron. When sodium sulphate or sodium sulphide are available at a low price, the inventor may replace sodium carbonate as desulphurizer with iron oxide, and add enough sodium sulphate or sulphide to reduce the percentage of iron sulphide in the matte to a desirable figure. When sodium sulphate is used the following reaction occurs:

$$Na_2SO_4 + 4C = Na_2S + 4CO$$

Pyrite mixed with the [illegible] sulphur and ferrous sulphide, which latter forms part of the matte. Lead oxide, if present, is in part reduced by carbon and takes no part in the formation of matte. Lead sulphate, if present, [illegible] allowing enough carbon to reduce it, [illegible] the lead sulphide produced should be figured as so much galena.

Furnace Construction.

Tilting Furnace.—J. C. Cromwell (823,669, June 19, assigned to Garrett-Cromwell Engineering Co.) patents the tilting furnace shown in Fig. 1. It is [illegible] of the open-hearth type, with gas [illegible] from the regenerators, and is mounted upon a framework of girders C. The girder frame has at each end supporting brackets C^1, in which are journaled rollers C^2, adapted to ride upon the curved trackway D. The curvature of this trackway is that of a circle, having its center about the gas and air ports BB. The segmental rack D^1 serves for the tilting operation, an electric motor being used with suitable gearing. In the illustration the driving connection is shown as that of a worm-shaft E^1 and wheel E^2, transmitting power from the motor E to gears E^3E^3, which in turn transmit the power to cog E^4, meshing with the segmental rack D^1. This reduction device is stated to be very effective, the furnace being under absolutely perfect and positive control in all positions. This furnace may obviously be rotated about a given axis, [illegible]

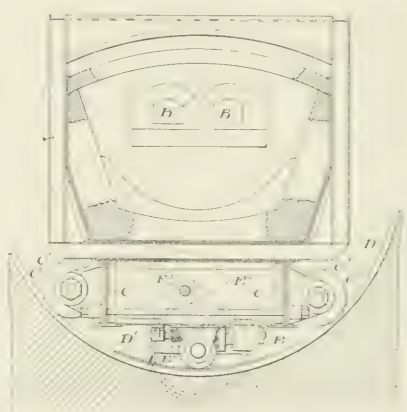


FIG. 1.—TILTING FURNACE.

line passing through the centers of curvature of the [illegible] supporting rails, and it is so designed that the shifting of the load upon tilting will not disturb the state of static equilibrium, the center of gravity being close to the axis of rotation. During rotation the fuel ports are not exposed to cold air, and combustion may be continued at all positions of the furnace. No matter how the furnace may expand under the influence of heat, the supporting frame upon which it is mounted always receives the thrusts along exactly the same lines so that the original design will be effective and the [illegible] under the changed conditions.

Alloys.

Alloy for Electric Resistance. [illegible] June 26) patents an alloy which is intended as a substitute for German silver for use as electric resistance. The disadvantage of German silver is that it [illegible] service it crystallizes and breaks. [illegible]

of zinc. Driver, therefore, uses no zinc. His alloy consists of 75 parts copper, 20 nickel and 5 manganese. It is said to have "a high and practically constant resistance," and to be permanent and stable in its physical and electrical properties.

Nickel Alloy as Substitute for Platinum.—C. W. Birmingham (824,018, June 26) patents the following alloy, which is said to be fusible "only at a very high temperature," and "may be used as a substitute for platinum, especially as employed in connection with electrical appliances." It consists of 16½ ounces Troy silver, 4½ pounds nickel, ½ ounce bismuth, and 53 pennyweights gold. The alloy is said to be "highly ductive and far more durable than platinum."

Copper Alloy.—G. Chandoir, Jr. (820,954, May 22), patents an alloy which has higher tensile strength and greater ductibility than copper, bronze or brass at high temperatures. He adds from ¾ to 3½ per cent of cadmium to the copper or bronze or brass. "By this addition the tensile strength of copper at the temperature of 500° C. is increased to 30.3 kg. per sq. mm. The increase of the ductibility is especially conspicuous in the alloys richer in cadmium. An alloy containing 92 per cent of copper, 4½ of tin, and 3½ of cadmium has shown especially good results.

Contact Process for Sulphuric Acid.

Rejuvenation of the Catalyser.—In an article in our Vol. II., p. 347, in which a concise summary was given of the principal engineering and industrial points which brought about the success of the contact process for the manufacture of sulphuric acid, attention was called to the absolute necessity of keeping the contact substance of finely-divided platinum free from impurities, especially arsenic. This means that an actual chemical elimination of everything, except sulphur dioxide, oxygen and nitrogen, is necessary before the gas is admitted to the contact compartment. How this problem has been solved in commercial practice was described in the article referred to. A recent patent of I. Kitsee (821,042, May 22) refers to the same point, but he does not try further improvements of the purification of the gases, but rather purification of the finely-divided platinum, which forms the catalyser, when "poisoned" by arsenic and other impurities.

For this purpose he makes the finely-divided platinum the anode in an electrolytic cell with sulphuric acid as electrolyte. The principle of this method is that platinum is not dissolved, while arsenic, mercury, phosphorus, antimony and metals of the same class, known as "contact poisons," are readily dissolved and oxidized. To regenerate a contact substance which has lost its catalytic action on account of impurities, "it is only necessary to send a current density through the apparatus equal to 1 amp. for each ounce of contact mass, and the time to purify the same entirely varies from 1 to 3 hours." . . . "It is obvious that this contact mass can be made the electrode of an electrolytic apparatus during the time of the production of the sulphuric acid, and it is further obvious that in such cases the destructive influence of the contact poison is entirely obviated."

American Electrochemical Society.

At the June meeting of the board of directors the following gentlemen were elected members of the American Electrochemical Society: A. R. Johnson, Madison, Wis.; W. J. Rodgers, Philadelphia, Pa.; F. A. Stamps, Niagara Falls, N. Y., and H. I. Metzbow, New York City. At the next meeting the names of the following two gentlemen will come up for election: R. A. Hadfield, Sheffield, England, R. C. Stover, Norwich, N. Y.

The two vacancies in the board of directors, due to the election of Vice-President C. Hering to the presidency, and to the resignation of Manager C. J. Reed, were filled by the board electing Mr. Alfred H. Cowles, of Cleveland, a vice-president, and Dr. Henry Noel Potter, of New York City, a manager.

The constitution of the board is now as follows: President, Carl Hering (till 1907); junior past presidents, H. S. Carhart (till 1907), W. D. Bancroft (till 1908); vice-presidents, E. F. Roeber, L. Kahlenberg, A. H. Cowles (till 1907), and S. P. Sadtler, J. W. Richards, Samuel Reber (till 1908); managers, W. H. Walker, C. E. Acker, E. Weston (till 1907), C. A. Doremus, C. P. Townsend, W. R. Whitney (till 1908), E. G. Acheson, C. F. Burgess, H. N. Potter (till 1909); secretary, S. S. Sadtler (till 1907); treasurer, P. G. Salom (till 1907).

The executive committee consists of Mr. Carl Hering, chairman; Mr. S. S. Sadtler, secretary, and Messrs. W. D. Bancroft, H. N. Potter, J. W. Richards, S. P. Sadtler and P. G. Salom.

The tenth general meeting of the Society will be held in New York City during October, 1906.

BOOK REVIEW.

Industrial Furnace and Methods of Control. By Emilio Damour. Translated from the French, with additions, by A. L. J. Queneau, B. S. (Paris), E. M., A. M. (Columbia), Consulting Engineer with the New Jersey Zinc Co. With preface by Henry M. Howe, LL. D. New York: *Engineering and Mining Journal*. 317 pages, 88 illustrations, 24 tables. Price, $4.00.

The ground covered by Prof. Damour's book is the discussion of the calorific powers of fuels, temperatures of combustion, heat balance sheets of various kinds of furnaces, chimneys, theory of generators, losses of heat by radiation and conduction. The translator has added excellent chapters on pyrometers, gas analysis, burning of powdered fuel, conversion tables, and a bibliography; and Dr. M. N. Bolles has contributed chapters on calorimetry and the analysis of fuels.

The work abounds in meritorious features, the discussions being often most illuminating. It also abounds in shortcomings, part apparently existing in the original and part to be laid to the translator. The commendable features are particularly the complete tabulation of the data throughout the book, the liberal use of curves, so as to exhibit the results of calculation graphically, the clear and numerous illustrations, the very readable and easily-understood style of writing, the fine typography, good paper and attractive binding.

The shortcomings are in some cases such as to need more extended allusion. One great drawback is that all heat calculations throughout the work are made per gram of material burned; grams are well enough for laboratory use, but it seems ridiculous, and has practical disadvantages also, to cast up a balance sheet of a 50-ton open-hearth furnace, for instance, per gram of fuel burned or of steel melted. All gas calculations are based upon first transposing the actual volumes of gases concerned (measured in liters instead of cubic meters), into molecular volumes, by dividing the real volumes by 22.32. This is an unfortunate policy, for no one except an expert chemist ever has a concrete idea of what molecular volumes represent, whereas, everybody knows what actual volumes are. Thus, to gain a theoretical advantage, the authors have sacrificed the only method of presentation which can be helpful and comprehensible to practical men.

Both Prof. Damour's and the translator's fundamental chemical definitions are sometimes faulty and misleading, atoms and molecules being confounded (as on page 8 and in Table I). Sensible heat and latent heat are used almost interchangeably throughout, as if they were synonyms. In discussing combustion, it is stated that no external work is done when it takes place under constant pressure, and this mistake is continued through the discussion

Prof. Howe states that the translator's mastery of "our puzzling English rouses our applause," but we cannot join with the professor's enthusiasm. One illustration may illustrate the point: "Axiom—The total quantity of air necessary for the fuel combustion, escaping through the stack as water vapor, carbon dioxide and nitrogen, is always the same for a given weight of the burned fuel."

In spite of these defects, there is so much valuable material in the book that it should be a desirable addition to every metallurgist's library.

The Electrical Nature of Matter and Radio-activity. By Harry C. Jones. New York: D. Van Nostrand Co. 212 pages. Price, $2.00 net.

This is a collection of a series of articles published in the *Electrical Review*, carefully revised and brought up to date. The aim has been to present the important facts and conclusions in this field, as far as possible in non-mathematical language and in semi-popular style, but with strict scientific accuracy. The bulk of the work is an exceedingly clear and readable presentation of the whole field of radio-activity, numerous references being given to the original articles, so that any particular branch of the subject may be followed in detail outside the limits of this treatise. Prof. Jones has accomplished a very useful and very important work, in thus making accessible to the general reader the most recent advances in electrical and chemical science.

Notes on Electrochemistry. By F. G. Wiechmann, Ph. D. New York: McGraw Publishing Co. 145 pages. Price, $2.00

The six chapters of this elementary text-book deal with general principles (general principles of science, matter and energy, forms of energy, measurement of physical phenomena); electrical energy (theories of electricity, radio-activity, intensity factor and capacity factor of electrical energy, properties of electricity, Ohm's law); electrochemistry (evolution of electrochemistry, electrolysis, electromotive force, dissociation voltage); electrolytic dissociation (ion theory, conductivity, migration of ions); electro-analysis (electro-analysis, reversible reactions and cells, sources of current, current measurement, current regulation, electrodes, current density, records, resumé); electrotechnology (a very concise summary of the industrial applications of electrochemistry).

Differing from larger standard books on theory and applications of electrochemistry, "the present notes aim merely to offer a general survey of the subject, to serve as an introduction to its study and to aid in the securing of a proper understanding and appreciation of the work along individual lines." The strongest point of the book is the thorough way in which fundamental conceptions and principles are worked out. Without going deeply into details of electrochemical theories, the author succeeds in giving a good survey of the most important principles, the latest advances in physical science being taken into consideration. The author's individuality is often exhibited in a strong manner. It is probably the first time that a text-book of this kind has been written by a chemical engineer, not by a professor. The book is free from the rehash from larger works, so often found in elementary text-books.

The last chapter of electrochemistry is interesting in its concise recapitulation of the most important industrial processes using electrochemical methods. The classification employed in this chapter is novel. The author distinguishes primarily between direct-action processes and indirect-action processes, the latter group comprising all processes in which the electrical energy suffers partial or entire transformation into some other form of energy before being applied to the work required, thus including electric furnace processes and electrolysis of fused salts. The weak point of this classification is that processes like the production of caustic soda and chlorine by the mercury-cathode process of Castner on one hand, and by the usual diaphragm process of Acker on the other hand, are separated from each other, although they belong together [illegible]. Of course, [illegible] point. This chapter contains, on not more than forty-five pages, a vast amount of detailed information, and if we consider the swift changes in the industry it is easily understood that some of the information given in the book is not fully up to date (*f. i.*, fixation of atmospheric nitrogen, p. 128).

One small and purely formal point calls for comment. Throughout the book the word cation is spelled with the K. We know the author is not the only one spelling the word in that fashion; Faraday did so and others are doing so, although it is unquestionably wrong. Cathode is derived from the Greek words cata and hodos, cation from cata and ion, not hion.

A special feature of the book is that as introductory note to each chapter a bibliography is given of important treatises bearing upon the subject matter discussed in the chapter.

The book should be found useful by those who wish to acquire a working knowledge of the first and most general principles which underlie electrochemical science and industry.

Handbuch der Angewandten Physikalischen Chemie. Edited by Prof. G. Bredig. Leipzig: Johann Ambrosius Barth.

This new pretentious series of German monographs intends to cover the whole field of *applied* physical chemistry. In other words, the purpose is to make the principles of physical chemistry accessible to the chemical engineer—to give a review of the theory, not for the theorist's sake, but for the benefit of the engineer.

This is an excellent programme. It becomes more evident from day to day that in the chemical industries the day of the pure analyst is past, and that chemical and metallurgical engineering must necessarily advance as applied physical chemistry. If the editor of the work and his contributors are able to keep up the standard set in the first volume, reviewed below, the serial will become a monumental and classical work.

Vol. I.: Elektrochemie wässeriger Lösungen. By Prof. Fritz Foerster. 507 pages. Price (in New York), $6.75 in paper cover, and $7.00 bound.

This is undoubtedly the best book now available on the electrolysis of aqueous solutions. Prof. Foerster is probably best known from the extended research work which, with the coöperation of his assistants and students, has been carried out by him on the electrolysis of chlorides, bromides and iodides. In this work, extending over several years, the author's method showed two general traits: first, an aim to supplement electrochemical research by parallel chemical research (as Prof. Bancroft would say, he endeavored to consider the chemistry of the electrochemistry of any special problem), and, second, an aim to develop the theory for the sake of its practical applications. The numerous papers of the author on this special subject have provided a safe and sane theoretical foundation for a uniform theory of electrolysis of chlorides, bromides and iodides, and also the way he gives, and will give, valuable hints to the engineer. Incidentally, we may mention that in studying the behavior of carbon anodes in chloride solutions, Prof. Foerster was one of the first to recognize the value of Acheson graphite, and to make Germany familiar with the work of E. G. Acheson.

The whole volume before us, dealing with the broad field of the electrolysis of solutions in water, manifests a broad spirit. [illegible] chapters of a purely introductory character. One hundred of the book [illegible] deal with electrical energy and the current in general, Faraday's law and its practical applications, theoretical consequences of Faraday's law, and [illegible] voltage, [illegible] electrolytic [illegible] secondary [illegible] general theory of electrolysis.

The other six chapters deal with industrial electrolysis and fill two-thirds of the book. First comes a chapter on the electrolytic production of hydrogen and oxygen gases, then two chapters on special electrochemistry of metals and on application of electrochemistry of metals respectively. The last three chapters (200 pages) which are probably the most important ones, since the information contained in them is not to be found elsewhere in such convenient and complete form, are devoted to electrolytic reduction, special electrochemistry of halogens and electrolytic oxidation.

The book is fully up to date; it is written from a very broad point of view, and at the same time it is accurate in details. Its careful study is highly to be recommended to anybody who has to do with the electrolysis of aqueous solutions.

Vol. II.: Physikalisch-Chemische Mineralogie. By Prof. C. Doelter. 272 pages, with 66 illustrations. Price (in New York), in paper cover, $4.00, bound, $4.35.

This treatise is more tentative than finished, but mineralogists and geologists should find it suggestive in many ways. The author recognizes that methods of physical chemistry applied to mineralogy will be found most useful by indicating the direction in which to proceed in experimental research. This is only natural, since it is largely almost virgin soil that is covered in the book.

Autogenous Welding of Metals by the Oxy-Acetylene Blowpipe and a New Method of Generating Oxygen.

By Andre Beltzer.

For a long time manufacturers, especially in Europe, have been endeavoring to devise means by which the autogenous process of welding could be so cheapened and simplified as to bring this highly effective method into general industrial everyday use.

The following processes have up to the present been used, although to a limited extent: First, welding by electricity; and, second, the Goldschmidt aluminothermic process.

Welding by the electric arc cannot be considered as generally practicable on account of the power required, the special installation necessitated, the danger and inconvenience in handling the electrodes, the difficulty in the regulation of the temperature, and the condition of the atmosphere created by the electric arc by which the texture of the metals is unfavorably affected; iron, for instance, becomes hard and brittle and unfit for working, while copper takes on a porous, spongy condition, etc.

The aluminothermic process is also complicated, involving the building of moulds around the pieces to be welded. The crucibles and moulds used in this process are expensive, and suffer greatly from deterioration.

The application of autogenous welding by the above processes are limited, since they cannot be adapted to complicated assembling and the welding of thin sheets.

The blowpipe is the only simple and practicable solution for the process of autogenous welding, on account of the facility with which it can be handled. The gas which can be most advantageously used in combination with oxygen is acetylene, for the following reasons:

1. The temperature of the oxy-acetylene flame nearly equals that of the electric arc, reaching 3,500° C. (That of the oxy-hydrogen flame is about 2,000° C.) The calorific power of the gas is 13,368 calories per cubic meter (not including the latent heat given out by the condensation of combustion products which is not utilized). The oxidation heat of hydrogen is only 3,860 calories per cubic meter. It can be readily seen that the oxy-acetylene flame is applicable where the heat furnished by the oxy-hydrogen flame would be insufficient, for instance, in the welding of iron of 15 mm. thickness.

2. Whilst the oxy-hydrogen flame is difficult to regulate, the oxy-acetylene flame can be perfectly regulated by the most inexperienced operator, on account of its brightness. It is in fact of the greatest importance in metallurgy that the chemical nature of the flame be perfectly known.

3. Calcium carbide, from which acetylene gas is generated by simple contact with water, is cheap and portable.

Generation of Oxygen.

The greatest factor militating against the development of autogenous welding by the blowpipe has been, up to a short time ago, the high commercial price of oxygen, but owing to the advent of steel receivers, in which the gas is contained under high pressure, the oxygen industry has grown considerably, though still greatly handicapped by the difficulty and cost of transportation. The receivers are necessarily very bulky, being fifteen to twenty times heavier than the contained oxygen, and 150 times the weight of hydrogen. Railway companies charge their highest tariff (same as explosives), added to which is the rent of receivers and the cost of repairs (renewing of valves, painting, etc.).

It has, however, been finally demonstrated that chemically pure oxygen can now be obtained at a reasonable cost below the present price of oxygen in tanks, by the employment of a newly discovered product, called "epurite" (oxygen powder).

Epurite is a substance containing oxygen in a latent form, susceptible of easy liberation on contact with water, analogous to the action produced by water on calcium carbide.

Oxygen so generated is chemically pure, every kilogram of epurite giving about 55 liters of gas at an ordinary temperature of 15° to 20° C. and at 760 mm. barometric pressure. By reason of the low cost and the facility of manufacture, epurite brings oxygen within the reach of all industries, eliminating all danger of explosion arising from the use of tanks or tubes of compressed oxygen, and, being absolutely harmless, enjoys the lowest freight rate accorded chemical products by railway companies.

Welding.

Fig. 1 shows diagrammatically the general arrangement of the apparatus.

One of the oxygen generators *A* is charged with water and epurite. In the receptacle *C* is a solution of sulphate of iron, which is allowed to flow into the generator to act as a catalytic agent for the generation of oxygen. The oxygen liberated passes into the gasometer *D*, and is compressed to 10 atmospheres by the compressor *E* in the tank *F*. From the tank the gas passes by a tube through the pressure regulator *G* and valve *H* to the blow-pipe *K*, where the oxygen should arrive at a pressure of 150 cm. water.

The acetylene apparatus *NM* is arranged so as to give the gas at the above pressure. This pressure of 150 cm. water is calculated so that the exit speed of the gas will counteract the possible back burning of the mixture before reaching the end of the blowpipe.

The blowpipe is provided with metallic gauzes in order to prevent the flame throwing back; the valve of the acetylene tube (fixed to the blowpipe) is at first turned on full, the pressure regulator being adjusted to about a one-half atmosphere; the flow of oxygen is controlled by a valve *H*, so that there is only one inner cone in the flame which will have only slight fluctuations. The flame now is neutral and ready for use.

A whiter color of the flame and the division of the inner cone in two are indications that there is an excess of acetylene gas, and that the flame is carburating, the molten metal emitting sparks like stars (formation of cast iron). When the flame is oxydizing (shown by the violet tint of the flame) the metal boils and is very bright. For proper welding (steel sheets, for instance,) the joint should be bright. The carburating flame gives a gray porous and non-resistant welding. This flame, together with an oxydizing flame, gives a brittle

welding, and is, moreover, very rarely used. Twenty different sizes of nozzles can be used on the same blowpipe in welding of all thicknesses, from 1 mm. to 30 mm. (0.6 mm. for sheets 1 mm. thick, and 4 mm. for sheets 30 mm. in thickness).

During the process of welding the apex of the cone must

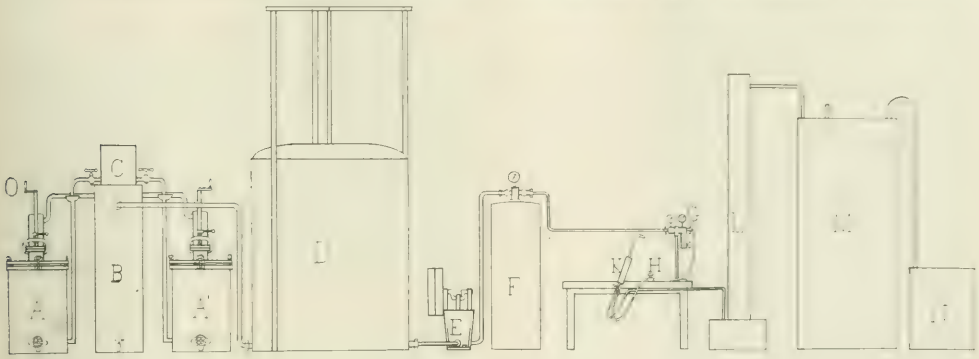


FIG. 1.—ARRANGEMENT OF PLANT.

(A to H, oxygen apparatus; I. to N, acetylene apparatus. A and A', oxygen generators; B, [illegible]; C, [illegible]; D, gasometer; E, compressor; F, pressure reservoir; G, pressure regulator; H, cock; K, hosepipe; L, [illegible]; M, [illegible] sure; N, acetylene generator.)

be from 2 mm. to 3 mm. distant from the object to be welded. The two edges (previously dressed) are fused, and simultaneously lined and slightly overloaded by the fusing of a rod of the same metal held in the flame.

In this manner iron, steel, copper, brass, cast iron, etc., can be effectively welded.

For thick metals or plates it is necessary to bevel the edges, which can be readily done by many mechanical methods, such as planing, slotting or grinding, and it is well to anneal the sheets or plates, thereby increasing the resistance and quality of the welding.

For brass it is necessary to fill up the interstices of the two sheets to be welded with borax moistened with water, otherwise the volatilized zinc would be deposited on the welded part as oxide of zinc and spoil the welding.

Cast iron which cannot be self-welded must be brazed with copper.

The Blowpipe.

The volumes of oxygen and acetylene used are practically the same. The acetylene is, therefore, not completely burned in the blowpipe according to the reaction:

$$(1)\ldots 2C_2H_2 + 5O_2 = 4CO_2 + 2H_2O$$
$$4\ \text{Vol.} \quad 10\ \text{Vol}$$

But it is incompletely burned according to the reaction:

$$(2)\ldots C_2H_2 + O_2 = 2CO + H_2$$
$$2\ \text{Vol.} \quad 2\ \text{Vol.}$$

This is easily understood when we consider that at a temperature of 3,500° C. the water and carbonic acid formed by the reaction (1) are completely dissociated and cannot exist.

To this last fact the success of the oxy-acetylene flame as a welding agent is chiefly due. To bring about the proper conditions for autogenous welding two metals it is necessary to bring them to their melting point without oxidizing or carburating. The hydrogen and carbon oxide generated in the oxy acetylene flame form a reducing atmosphere and permit the metal to be autogenously welded. These two gases protect the molten metal, and are then burned entirely into carbonic acid and water by the surrounding air. With hydrogen a large excess of gas is necessary, five to six volumes (instead of two theoretically), in order to resist the oxidizing action of the water which is immediately formed in the flame.

In actual use, on account of trials necessary for regulation,

The following table shows the calibres of blowpipe necessary to use for various thicknesses of sheets, etc.:

Calibres.	*Thickness of Sheets.*	*Number of Welded Meters Per Hour.*
0.8 cm.	1 mm.	4.00 meters
1.0 "	2 "	2.00 "
1.2 "	3 "	1.50 "
1.4 "	4 "	1.25 "
1.8 "	5 "	1.00 "
2.0 "	10 "	0.50 "
2.5 "	15 "	0.35 "

Column three shows the amount of work an operator of average ability should accomplish.

It is an established fact that in general the cost of welding by the epurite process is about 50 per cent of the cost of ordi-

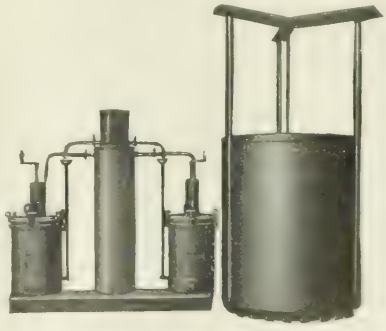


FIG. 2.—EPURITE APPARATUS.

A, V, E, D, [illegible].

nary riveting on work up to and including 7 mm. [illegible] cost of welding would be the same as riveting on sheets 8 to 9 mm. in thickness, but more advantageous in regard to strength.

For heavier sheets the oxygen can be [illegible] previously heating them with an ordinary blowpipe. When the iron is satisfactorily heated process [illegible]

After trials made by the International Bureau Veritas in Paris, it was ascertained that the tensile strength of the welded part was within 5 per cent of that of the metal itself.

FIG. 3.—OXYGEN RECEIVERS.

It will be seen, therefore, that on account of the facility and the low price at which the two gases, acetylene and oxygen, can be generated, the blowpipe, in its own industrial sphere, is facile princeps.

It can be used by every manufacturer with the best possible results for:

1. Simplification and great improvement in all kinds of assembling (boilers, gasoline and kerosene tanks, piping, etc.)

2. Easy, rapid and sure repairing of all parts (particularly forged), which formerly had to be abandoned on account of flaws, cracks, etc.

3. Work which would be impossible or very difficult to do without the blowpipe.

A New Direct-Reading Electric Thermometer.

In our last issue we mentioned briefly on page 208 a very interesting paper, read by Mr. Edwin F. Northrup at the Milwaukee meeting of the American Institute of Electrical Engineers. The main part of the paper consisted in a detailed discussion of the electric resistance thermometer. The author explained that resistance thermometers are useful for different particular purposes and that in any case the particular purpose for which the thermometer is to be used largely determines its special features of construction. Various types of construction to meet different requirements were discussed in detail by the author, who also dealt fully with the different methods of reading resistance thermometers.

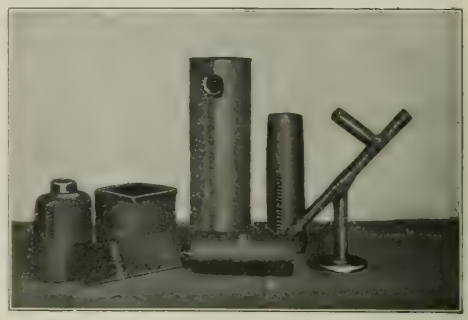


FIG. 5.—WELDED PIECES.

A special instrument described by the author and built by the Leeds & Northrup Co., of Philadelphia, embodies the highest refinements in this kind of measurement. The problem was to get a continuous photographic record of the temperature difference between two mains carrying brine for refrigerating purposes. One main was on the average on a temperature of −38° C., and the other −36.5° C. The average difference was, therefore, 1.5° C. The allowable error in the record was 0.01° C. The methods by which this problem was solved by the author were fully described in the paper.

Another very interesting new instrument also described in this paper is the "ratiometer." If the temperature of the place to be measured is uniform over a small space only, then the resistance winding of the resistance thermometer should be as short and as much concentrated at the end of the tube as possible, and thus permit of placing the entire winding at the point the temperature of which is to be measured. Thermo-couples have in this respect an advantage over the ordinary resistance thermometers, since the end of the thermo couple has a very small body, which may be placed at the exact point where the temperature is to be measured. This consideration led Mr. Northrup to design a new form of resistance thermometer which is intended to combine the advantages of both.

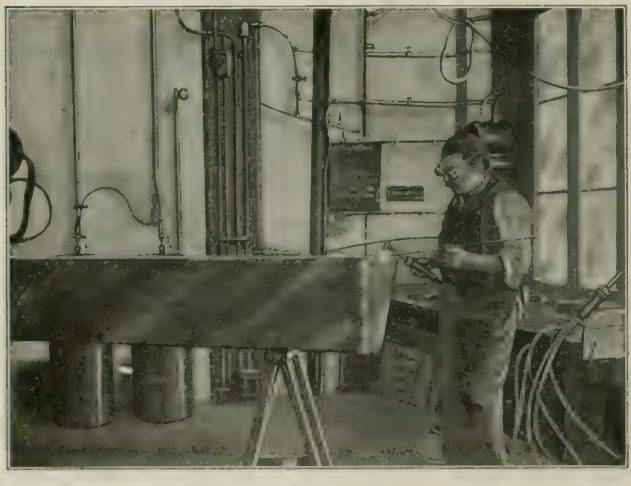


FIG. 4.—WELDING WITH BLOWPIPE.

It is called a ratiometer, since it is a special form of a general type of instrument which measures by direct deflection of a needle the ratio of an unknown to a known quantity, irrespective within wide limits of the operative current used. Thus the ratio measured may be that of an unknown to a known resistance, when the instrument becomes a deflection ohmmeter, or a capacity-reactance to a resistance, when it can be used as a speedometer, or, as in the present case, a resistance which changes with temperature to a fixed resistance, when it serves as a direct-reading deflection thermometer. Since the angle of deflection of a deflection instrument is apparently doubled by using a mirror giving twice the effective sensibility of a pointer instrument otherwise the same, a system using a mirror has been adopted, and utilized in such a manner that the instrument is quite as portable as a voltmeter or ammeter. The ratiometer will, therefore, enable one to read temperatures as readily as

volts are read from a voltmeter.

Fig. 1 is a sectional illustration of the instrument, while Fig. 2 gives the diagram of connections, and Fig. 3 illustrates details of the deflecting system and the shape of each of the pole pieces.

C_1 and C_2, Fig. 2, are two flat coils mounted on the movable system *a*, *b*, *c*, Fig. 3, having a damping frame of aluminum. The two coils have like polarity on the same side. The system rotates between two iron crescent-shaped pole pieces of opposite polarity *p* in Fig. 3. The axis of rotation of the system approximately coincides with the center of the outside circle of the crescent. Hence, when the system rotates, one coil moves so as to get more under the pole faces, and the other coil more from under the pole faces. Now, if currents flow through both coils, and in such a direction as to cause both coils to try to move from under the pole faces, the system will seek a position of equilibrium which is independent of the actual value of the currents flowing, and which depends only upon the ratio of the portions of the main current, which divides to flow in the two coils.

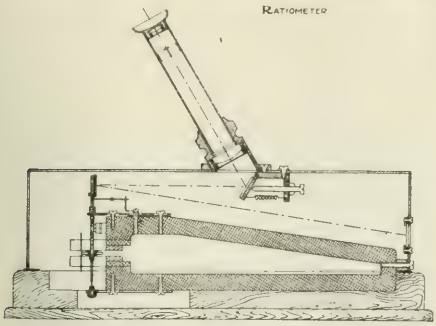


FIG. 1.—SECTIONAL VIEW OF RATIOMETER.

This is true, provided only that the system is under no spring control. The leading-in wires, three in number, are, in fact, made of such delicate strips of silver that they exercise a negligibly small control as compared with the control of the magnetic forces. If, now, the ratio of the currents in the two coils is altered, the system seeks a new position of equilibrium again, independent of the value of the battery current.

Referring to the diagram of connections, Fig. 2, R is a fixed resistance, and T a resistance which varies with the temperature. The extent of the deflection determines the ratio of T to R, whatever be the value of the battery current. Since, also, *y* is a lead wire on the R side equal to *y* on the T side, the method nearly compensates for changing resistance in the thermometer leads. It will be seen that if R = T, the battery Ba sends the same current through both halves of the system. Now, R is a resistance of wire of zero temperature coefficient, T a resistance of wire of a high temperature coefficient. Therefore, if T is placed, for instance, in a furnace, T changes according to the temperature of the furnace, R remains constant. The ratio of T to R determines the deflection, which therefore gives directly the temperature of T.

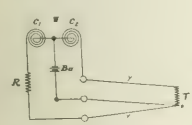


FIG. 2.—DIAGRAM OF CONNECTIONS.

The reading device is mostly explained by Fig. 1. The scale is of translucent celluloid, mounted on the lower part of the front of the case, and is divided in degrees centigrade. In the telescope the scale is seen brilliantly illuminated by ordinary room light.

The telescope can be removed when the instrument is to be packed in its carrying case. The [illegible] aluminum [illegible] and thus make the instrument [illegible] ment operates for temperatures [illegible] dry cells connected in series, or it may be used with a commercial current from [illegible].

To read any temperature [illegible] thermometer, joined by a lamp-cord to the instrument, in the place where the temperature is required. Then, with the telescope. The scale appears to move [illegible] the eye piece to the temperature reading.

The ratiometer may be made sensitive to temperature changes of about 0.1° C. or less, and may be [illegible] as accurate as the [illegible] calibrated, with the exception of possible [illegible] of 1, or at most 2°. These irregularities are due to [illegible] the construction, or imperfect balancing of the coil. The error is not progressive, however, and fair reliance can be placed upon the indications. The instrument is especially valuable for rapidly taking the high temperatures of furnaces and the like.

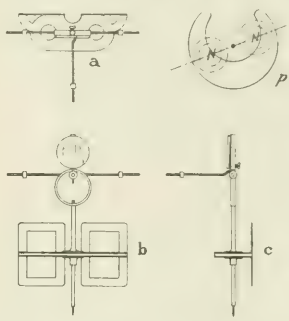


FIG. 3.—DETAILS OF DEFLECTING SYSTEM.

The ratiometer is made by the Leeds & Northrup Co., of Philadelphia.

Thermit Repair of a Gold Dredge.

The repair of a gold dredge, recently made in California, is an interesting illustration of the ease with which thermit may be applied to the greatest advantage to repairs of machinery in remote mining fields. As is well known, thermit is a mixture of finely divided iron oxide and aluminium, and the reaction (which is started by igniting a special ignition powder with a match) consists simply in the combination of the aluminium with the oxygen of the iron oxide. Since this gives out an enormous amount of heat, the iron set free is obtained in form of a superheated liquid soft steel at 5,000° F. within 30 seconds after starting the reaction.

It is self evident that it is possible to do many things with such superheated liquid steel for the repair of iron or steel machinery and for other purposes in iron and steel metallurgy. Other practical advantages of the method are the simplicity of the necessary outfit and the ease of transporting it. Since the process has repeatedly been discussed in detail in our columns, nothing further may be said here on the process in general.

The repair to be described in this note was made at [illegible] Calaveras County, California, where the Mokelumne Mining Co. (Mr. William C. Colley, general manager) had some troubles with the gold dredge shown in Fig. 4.

The trouble was principally with the upper [illegible] the driving wheel to which the whole bucket [illegible] which drives it by teeth on sprockets. This bucket line consisted of sixty-four buckets, each weighing 1,100 pounds, and having 3 cubic feet capacity.

The [illegible] shaft was 12 inches to 13 inches in diameter and had a [illegible] keyed to it by two heavy [illegible] keys. The hub was tapering and 20 inches on the large end. The shaft with the heavy sprockets slipped over this hub and was driven over the taper by three bolts, 14 inches in diameter.

Through some fault, probably of alignment, the whole

tumbler, hub and shell crept along the shaft, notwithstanding heavy bands designed to prevent this, and through a faulty and slightly large bore of the hub the keys could not be kept in

FIG. 1.—SIDE VIEW.

place and were gradually "chewed up" by the working of the hub.

The Pacific representative of the Goldschmidt Thermit Co.,

FIG. 2.—BEFORE THE THERMIT REPAIR.

Mr. L. Heynemann, 2721 Clay Street, San Francisco, Cal., was called in for consultation, and after examining the shaft and tumbler he concluded that it was possible to "freeze" or fuse the tumbler immovable to the shaft, that is to say, to cast a thermit steel band on each side of the hub which would fuse both to the shaft and the face of the hub.

The repair was made in the beginning of April, the tumbler shaft and hub being placed in the yard near the dredger with a frame built over it. Fig. 2 shows the recess in the hub and also that the smaller part of the shaft (12 inches in diameter) was on the large taper end of the hub, while the larger part (13 inches in diameter) was on the smaller end of the taper. This difference resulted in a difference in the thermit bands on each end, the band on the 12-inch end taking more metal than on the 13-inch end.

FIG. 3.—HEAVY BAND COMPLETED.

The heavier end was treated first. On April 10, after rain in the forenoon, both cores were fitted. On April 11, late in the afternoon, after the shaft had been heated to a "black red," the reaction took place. The crucible containing the thermit mixture was placed above the mould, the principle being essentially the same as is now so well known from the practice of welding rails.

Fig. 3 shows the heavy band completed, with cores, gate and riser removed. It shows the nutways and also the fin on the horizontal plane through the parting shaft center, where the top and bottom cores met.

On April 12 the cores were fitted for the other end, and by 10 o'clock that night they were ready for the reaction. The

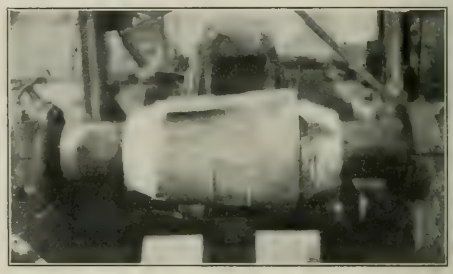


FIG. 4.—SHAFT IN PLACE.

news that this work was to be done had spread all over the country, and the natives came in all kinds of rigs to witness this peculiar kind of fireworks. The result was eminently satisfactory from an engineering point of view. Fig. 4 shows the shaft in place on the top of the dredge and the thermit bands on each side of the hub before the shell and bucket had been replaced.

American Foundrymen's Association.

The enormous success of the Cleveland meeting, held from June 5 to 7, is indicated by the fact that the registration passed the 800 mark. The secretary of the Association, Dr. Richard Moldenke, was as ever the soul of the proceedings. A special feature of this year's convention was a fine and extensive exhibition of foundry supplies and equipment, which was so much appreciated that it is intended to make such exhibits a permanent feature.

The president, Mr. Thomas D. West, presented an address on the need of a practical education. He urged that "the education of our youth should be confined to that which can be directly utilized." He also demanded specialization.

The committee on standard methods for the analysis of iron (H. E. Diller) presented a report on the determination of total carbon and sulphur. This will be given in full in our next issue. Among the papers presented were the following: V. E. Outerbridge, on the beneficial effects of adding high-grade ferro-silicon to cast iron; R. Moldenke, on foundry tests of coke made by the United States fuel testing plant at St. Louis; R. West, on comparative designs and working of air furnaces; F. L. Antisell, on electricity in the foundry; C. Vickers, on alloys; C. H. Benjamin, on experiments on the strength of various cast iron machine parts; W. K. Barrows, on influences of different ore mixtures on the resultant pig iron from the standpoint of the foundryman; W. M. Carr, on the thermit process in foundry practice. Abstracts of various of these papers will be given in our next issue.

Visits were paid to the Case School of Applied Science and to the works of the Brown Hoisting Co., Wellman-Seaver-Morgan Co., Electric Controller & Supply Co., and the Lorain Steel Co.

Mr. W. H. McFadden, of Pittsburg, is the new president.

Notes.

The Zucker & Levett & Loeb Co., manufacturers of complete electroplating outfits and supplies, as well as low-voltage dynamos for general electrolytic work, have opened temporary offices at 250 Tenth Avenue, New York City. During May their factory and offices on West Twenty-fifth Street were destroyed by fire. They now occupy a large new factory adjoining their old premises, consisting of Nos. 516 to 524 West Twenty-fifth Street, where they will have much larger facilities than before for manufacturing electroplaters' supplies. They announce that in spite of the serious nature of the fire they are now able to make delivery of goods from their new factory.

Mr. Chas. J. Bogue, the well-known New York manufacturer of low-voltage dynamos for electroplayers, electrotypers and for general electrolytic and electrometallurgical work, announces that on June 16, 1906, his business was incorporated under the name of Chas. J. Bogue Electric Co., 213 to 215 Center Street, New York City. Mr. C. J. Bogue is president, Mr. L. Robertson treasurer, and Mr. F. Plante secretary of the company.

Messrs. J. H. Gautier & Co., Jersey City, N. J., manufacturers of black lead crucibles, whose works were badly damaged by fire during March last, announce that their works are again in operation, and they are able to make deliveries, their facilities being better than ever before.

Thermit.—An illustrated pamphlet, entitled "Thermit, What You Can Do With It," has recently been issued by the Goldschmidt Thermit Co., of New York City. It describes in a simple and easily understood language the numerous applications to which thermit can be put in the industries, and especially in metallurgy.

Gas Power.—An illustrated pamphlet recently issued by the New York Electrical Society, contains an address by R. T. Lozier on fundamental principles of gas engines and gas producers. The De La Vergne Machine Co., New York City, who are the sole licensees for the manufacture of the Koerting gas engine and producer in the United States, have just issued a ten-page folder describing the Koerting four-cycle gas engine and suction producer. These engines have met with remarkable success in Europe, where a large number have been installed in municipal lighting plants, factories and other places.

Water Power.—Mr. H. P. Broughton, general manager of the Great Northern Power Co., of Duluth, Minn., states that the first 10,000-hp. unit will be put in operation at their power plant about September. These units are of the vertical shaft type manufactured by the General Electric Co., and operated under a head of 375 feet. The Great Northern Power Co. has 200,000 hp. available, 30,000 of which will be put in operation within the next year. The plant is situated 3 miles from the city limits of Duluth, the distance for transmission to their distributing center being 14 miles. The first electrochemical plant to use this power will probably be a calcium carbide plant operating under the patents of Mr. H. L. Hartenstein, which has been manufacturing this product at Constantine, Mich. Several of Mr. Hartenstein's furnace patents were abstracted in our June issue on page 257.

Ice and Refrigerating Machinery.—A 48-page pamphlet, describing various types of horizontal and vertical ammonia compression refrigerating machines and equipment for ice plants, breweries, packing houses, etc., has just been issued by the De La Vergne Machine Co., foot of East 138th Street, New York. The book is illustrated by many fine half tones, a feature of these being the clever and artistic arrangement of composite views of plants installed in various parts of the world. The booklet should be of special interest to iron metallurgists on account of the success of Mr. Gayley's dry blast system and the use of refrigerating machinery for drying the blast.

The Wilson-Maeulen Co., [illegible] metallurgical apparatus, [illegible] Street, New York City, [illegible] Company, of Le Chatelier type, [illegible] recording instruments, and [illegible] and special pyrometers.

The Green Mountain Mining Co.'s mill at Silverton, [illegible] which was described at length in our [illegible] was recently damaged by a conflagration. It has been rebuilt and is again in running order. The [illegible] signed and built complete from the ground up by The [illegible] Engineering Co., and was [illegible] the time of the [illegible] the land slide. As soon as the news of the [illegible] received in New York orders were given to the Traylor Engineering Co. to rebuild whatever was destroyed, and work was begun at once. The reconstructed building is designed for two units of 150 tons capacity each per day, but only one will be operated at present. There is sufficient power to bring the output up to 280 tons when ever desired. Double 8-foot rolls will be run for the present, each shaft requiring about 5 hp. to the nine and twenty in the mill. A watch is to be kept on the Green Mountain Camp which will be watched with interest is the Traylor centripact screen. Eleven have been introduced in the Green Mountain mill. Mr. W. M. Turner, formerly of the Cobra Mining Co., of Santiago Province, Cuba, is the new foreman of the resurrected Green Mountain mill, taking the place of D. R. Hickey, who lost his life in the [illegible] which demolished the mill March 17.

Electric Power from Blast Furnace Gases.—The very important saving which is attainable by utilizing to the utmost the calorific value of the "waste gases" of blast furnaces, has been the subject of various articles and notes in this journal; reference may be made especially to two articles by F. duP. Thomson and A. J. Rossi, on pages 95 and 150 and 190 of our Vol. III., respectively. It is well known that the Lackawanna Steel Co. and the De La Vergne Machine Co. were the pioneers in this field in this country. Important advances are now being made in the same direction by the United States Steel Corporation. The entire gas-engine driven electric generating equipment of their steel mills, except for two units placed as experimental orders with other builders, has been designed by the engineers of the Allis Chalmers Co., and is now being constructed in its shops. Orders for gas electric generating units recently placed with Allis-Chalmers Co. comprise 15,000 hp., to be furnished for the Illinois Steel Co., 4,000 hp. for the Homestead plant of the Carnegie Steel Co., and 35,000 hp. for the Indiana Steel Co. Each unit is of 2,000-kw. capacity. It consists of a horizontal twin-tandem Allis Chalmers gas engine, direct connected to an Allis Chalmers generator. All of the generators except two are designed for alternating current, 25-cycle, three phase. In addition to these electrical gas engines, the Allis-Chalmers Co. is building equally large gas blowing engines for the United States Steel Corporation, four of which are to be installed in the Homestead plant. Of the [illegible] engines purchased for this purpose, which are to be constructed by other companies, all will be built under a license from Allis-Chalmers Co., owner of the Slick patents covering the blowing "tubs" adopted for these machines. Each blowing engine has a capacity of 3,500 hp., and will deliver 30,000 cubic feet of free air per minute against a pressure of 18 pounds per square inch, which is ordinarily the maximum, but is [illegible] designed and proportioned that it can be operated at air pressure up to 30 pounds. Gas engines [illegible] from Allis-Chalmers Co. by the United States Steel Corporation will have an aggregate normal capacity of 58,000 hp., and represent a value of over $2,000,000.

E. J. Lavino & Co., Bullitt Building, Philadelphia, Pa., have been appointed the American [illegible] Keller, Leleux & Co., of Livet, France, for the introduction of the Keller furnace and processes for the [illegible]

grade steels and ferro-alloys. Messrs. E. J. Lavino & Co. are well known throughout the iron and steel industries in this country as large importers and dealers in ores, metals and alloys.

The Weston Electrical Instrument Co., Waverly Park, Newark, N. J., have this year taken their Mr. Caxton Brown into the Newark factory as secretary of the company and sales manager, and in his place have put Mr. Stanley Brown as manager of the New York office. In connection with this office there has been installed a repair department, whose duty it is to take care of the repairs in the Metropolitan district, and particularly to look after emergency calls. This new feature has been very warmly welcomed by the many users of Weston instruments in New York City, and has enabled the manufacturer to insure a higher degree of satisfaction to his customers than ever before.

Obituary.—From Germany comes the report of the death of the distinguished professor of iron and steel metallurgy in Freiberg, Geheimer Bergrat A. Ledebur, at the age of 69 years.

Personal.

Mr. J. B. Tyrrell, mining engineer, who has spent the past eight years in the Klondike, Yukon Territory, has opened offices at 206 McLaren Street, Ottawa, Can., and is prepared to report on mining properties of any kind.

Dr. Schuyler Skaats Wheeler, president of the Crocker-Wheeler Co., Ampere, N. J., sailed June 14, on the Lloyd steamship "Barbarossa," for a short European trip. He was accompanied by Prof. Francis B. Crocker, professor of electrical engineering at Columbia University, who has been associated with him in business for many years.

Dr. Rossiter W. Raymond, the distinguished secretary of the American Institute of Mining Engineers, received the honorary degree of LL. D. from Lehigh University on Commencement Day, June 13. This is the first time that this degree has been bestowed by Lehigh on any one. Wearing for the first time the garb of an alumnus of Lehigh, Dr. Raymond addressed the graduating class on "professional ethics."

Dr. Joseph W. Richards, of Lehigh University, with Mrs. Richards and their three children, sailed from New York on June 29 for Mexico, where they will pass the summer vacation. Dr. Richards will attend the International Geological Congress, which meets in the City of Mexico in September, and make an extended "metallurgical journey" through Central Mexico before returning in October.

Digest of U. S. Patents.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

Discharges Through Gases—Ozone.

(*Continued from page 252.*)

No. 507,975, Oct. 31, 1893, Jackson De Neal, Toledo, Ohio. Generates combustible gas from a solid material such as charcoal, coal-siftings, iron ore or garbage and oil. The solid material is fed downward and the oil upward into the space between the adjacent ends of two horizontal electrodes, between which an electric current is passed. The reaction chamber is surrounded by a shell, which has air passages and dampers to control the temperature. The electrodes are insulated by tubes of glass or mica, etc.

No. 511,330, Dec. 26, 1893, Ernst Fahrig, Marseilles, Ill. Produces oxygen by passing air through a revolving fan, thence through a filter of velvet coated with an albuminoid solution, to remove suspended matter and a portion of the nitrogen, and thence into a heated retort containing sodium manganate and lime. These absorb the oxygen, which is then driven off by the introduction of dry steam under pressure. The steam carries off the oxygen and is condensed therefrom. The oxygen is heated, dried by calcium chloride and sulfuric acid, cooled in iced coils and ozonized in boxes having alternate flat electrodes of glass coated with tin foil.

No. 512,265, Jan. 9, 1894, Emile Andreoli, London, England. Shows several ozonizers employing electrodes having points near to but out of contact with the dielectric. One apparatus comprises a closed box of marble or wood lined with pitch or glass, containing alternate spaced electrodes having opposed points separated by a plate of glass or mica. The air or oxygen may be cooled by a coil surrounded by ice and salt; by an initial compartment in the ozonizer containing ice and salt; or by cooling the entire ozonizer. The electrodes consist of tinned iron, copper, aluminium or tin or a tin alloy. One electrode may be flat or have pins carrying balls opposing the points of the other electrodes, the pins being tinned. Copper-wire brushes may be substituted for the points. In another construction a rectangular box of glass or porcelain is fitted with a plate having points on both sides, arranged between two sheets of tin or tin-plate, but separated therefrom by glass or mica. Enameled flat electrodes may be used; also electrodes having points cast thereon or punched therefrom, or provided with saw-shaped strips or consisting of fine wire stretched upon glass. Another type employs a tinned metallic core, or one of enameled iron or copper, in a tube of glass or earthenware closed at both ends by insulation. Around the tube is a cylinder having inwardly and outwardly projecting points, the latter surrounded by a metal tube electrode enameled inside and outside, or by a tube of glass and a tubular electrode. In another apparatus pairs of plates alternately pointed and plain and sheets of glass are arranged in the position of the laths of a Venetian blind. In another form, one electrode is a comb consisting of a tinned metal rod, to which are fixed metallic bands having teeth. The other electrode consists of metal rods inclosed in glass tubes and facing opposite teeth, or of enameled metal rods. The glass tubes are wound at intervals with fine copper wire, forming bands which are soldered to some of the teeth of the combs. The air to be ozonized is filtered through cotton or wool.

No. 523,262, July 17, 1894, Gustave A. Cannot, London, England. An apparatus for converting a mixture of oxygen and chlorine into dichlor monoxide, comprising a glass tube within which is arranged a central electrode of copper wire coated with silver, gold or platinum, and wound externally with a similar spiral wire electrode. The whole is inserted into and spaced from a glass gas-conveying tube by glass beads. The gas tube is surrounded by a glazed porcelain tube, contained in an outer case of thin metal through which cold water is circulated. The chlorine is preferably produced by the action of hydrochloric acid upon manganese peroxide; the oxygen by the action of sulfuric acid upon manganese peroxide, both in steam jacketed vessels. The chlorine is purified by passing it through a sodium chloride solution, and both the chlorine and oxygen by washing them. The gases are then dried and mixed. The drier is a rectangular box lined with glass, having a perforated horizontal partition carrying pumicestone wet with sulfuric acid. The chlorine and oxygen are admitted at one end of the box, the chlorine above the partition and the oxygen below it, their specific gravities causing them to mix. The chlorine monoxide is absorbed by a solution of caustic soda or potash.

No. 523,263, July 17, 1894, Gustave A. Cannot, London, England. Employs the apparatus of the preceding patent for the production of hypochlorous acid, employed for bleaching, especially of peat fiber. The gaseous product, chlorine monoxide, is passed through water, with which it unites to form the acid.

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The Utilization of Blast Furnace Gases for Power Purposes.

That the blast furnace, besides being primarily an iron producer, is at the same time an efficient gas generator, was early recognized. Thus J. B. Budd wrote in 1848 the prophetical words: "It would appear to be more profitable to employ a blast furnace, if as a gas generator only, even if you smelted nothing in it and carried off its heated vapors by flues to your boilers and stoves, than to employ a separate fire to each boiler and each stove." Metallurgical practice has followed to a certain extent the lines indicated in this prophecy. The blast furnace gases which were originally wasted and are for this reason often called "waste gases" even to-day, have long been usefully employed for heating hot-blast stoves, for raising steam in boilers, for calcining ore and for general heating purposes, like drying molds, etc. Further, before burning the gases, they may undergo, in certain cases, a treatment in condensing apparatus, so as to win tar and ammonia as valuable by-products. However, in recent years, with the growth of the gas engine, the necessity arose to get cheap gas, and while excellent progress has been made in the design of gas producers, yet the question naturally presented itself to use the blast furnace gases in gas engines.

* * *

Europe has taken the lead in this development and there are now some 300,000 hp. produced abroad in gas engines supplied with blast furnace gases. This country has been slow to follow, for the commercial reason that fuel is cheaper and labor more expensive here than in Europe. Nevertheless, the economy of the system was so manifest from the results obtained in European plants that this possible saving could not be overlooked by our iron masters. The pioneer plant in this country was that of the Lackawanna Steel Co., the gas plant being installed by the De La Vergne Machine Co. It was a courageous undertaking, using a large number of 2,000 and 1,000 hp. engines driving alternators. The first set was started on Jan. 2, 1903. Naturally, some difficulties were first experienced, partly due to certain inherent difficulties of operating large gas driven alternators, and partly due to troubles experienced in the gas plant. It is now unnecessary to argue whether these early troubles were due to certain points in the design of the gas engines, as was claimed from one side, or to the condition of the gas when delivered to the engines, as was claimed by the other side. This huge pioneer plant is now in successful operation, and this fact has undoubtedly acted as a stimulus toward further progress.

* * *

The United States Steel Corporation has now taken the matter up, and has done it in no half hearted way. We recently mentioned that gas-electric generating plant to the amount of 15,000 hp. has been ordered for the Illinois Steel

Co., 4,000 hp. for the Homestead plant of the Carnegie Steel Co., 35,000 hp. for the Indiana Steel Co., further, 3,500 hp. gas blowing-engines for the Homestead plant. All these engines will be furnished by the Allis-Chalmers Co., while the Westinghouse Machine Co. will provide eight 3,000-hp. gas blowing-engines for the Gary plant of the Indiana Steel Co. For several months a 350-hp. gas engine of this type has been in regular operation on blast furnace gas at the Edgar Thompson Works in Pittsburg, generating electric power for motor-driven foundry machinery. This engine was installed largely for experimental purposes, and has proven very successful. While this is being written, this engine is completing a thirty-day continuous load and duty test operating 24 hours per day and seven days per week during this period. As other instances of developments in this direction may be mentioned the 800-hp. gas plant of the American Steel & Wire Co., and the 1 350 hp. gas plant of the Pittsburgh Plate Glass Co. at Crystal City, Mo.

* * *

Now, about the saving which is attainable by the utilization of blast furnace gases for the development of electric power. Of perhaps the three best papers which have been written on this subject in recent years in this country, two were contributed to our Vol. III., supplementing each other in a very interesting manner: Mr. F. du P. Thomson, who was intimately connected with the pioneer blast-furnace gas plant of this country, took the stand of the progressive, yet conservative engineer, who claims only as much as he can safely guarantee in the present state of the art, and who knows not only the possibilities but also the drawbacks; Mr. A. J. Rossi contributed an equally clever and interesting article, but written in the mood of the enthusiastic and optimistic pioneer. A position somewhat between the two, with respect to the possibilities in exact figures, was taken by Mr. Freyn in his able paper, read some time ago before the Western Society of Engineers. Without any discourtesy to others, it is probably safest to base the following discussion on Mr. Thomson's figures. First, the requirements of the furnace plant itself must be met for heating and compressing the blast and for operating the auxiliaries (pumps, hoists, etc.). The remainder represents the surplus available for power, and Mr Thomson finds that the amount of surplus heat available for power per ton of iron produced per hour in the blast furnace is sufficient to develop .68 hp (with gas engines which have a thermal efficiency of 25 per cent).

* * *

According to the annual statistical report of the American Iron and Steel Association, issued June 30 1906, the total production of pig iron in the United States was 16,497,033 long tons in 1904, and 22,992,380 or, say, 23,000,000 tons in 1905. On the basis of the above figure of Mr Thomson, the surplus power available with an annual output of 23,000,000 of pig iron will be 1,225,000 hp. It will be interesting to make some comparisons. According to the last census report (1902) the total power in all electric central stations in this country was 1,800,000 hp. Further, to come to a subject of more interest just at present, the total hydraulic power of Niagara Falls, if fully developed, would be 3,000,000 hp. Mr H. W. Buck, of the Niagara Falls Power Co., in an able recent article in the *Outlook*, illustrated what the maintenance of Niagara Falls as a national spectacle means from the commercial point of view. It is equivalent to a great conflagration in which 50,000,000 tons of coal are annually consumed for spectacular purposes, since this is the amount of coal which would have to be burnt to produce 3,500,000 hp. with steam engines. Without entering into a discussion of the aesthetic versus the economic value of Niagara Falls, we may point out that the waste of the blast furnace gases in 1905 in this country was equivalent to a conflagration of more than one-third the size just mentioned. Since with blast furnace gases no aesthetic reasons are to be considered, it is really difficult to see why they should be wasted and have been wasted for such a long time. The recent rapid progress is, therefore, perfectly natural. Incidentally it must aid the commercial development of electric steel processes. As Dr. J. W. Richards once pointed out, if coal would be taken and used in a blast furnace for reducing iron ore, and if all the furnace gases were utilized in gas engines to run dynamos, and an electric refining furnace was used for making crucible steel, less fuel would be consumed altogether than is actually used for making steel by the crucible process at the present time.

◆◆◆

The Flow of the Lines of Current in Electrochemical Systems.

Storage batteries being a systematic means for changing electrical into chemical energy and vice versa, differ in their behavior in no way from other electrolytic cells and, therefore, form distinctly part of electrochemistry, considered as a science. Yet the practical engineer who looks at industrial applications and not at science for classification, will undoubtedly put storage batteries together with electrical apparatus and not with electrochemistry. It would be a pity if for this reason Mr. Schoop's article, concluded in our present issue, should escape the attention of practical electrochemical engineers. As it stands, Mr. Schoop's investigation was made for the special purpose of determining by experiment the configuration of the lines of current in storage batteries. But it is manifest that the same method may be applied to electrolytic cells in general, and that since the distortion of the lines of current can produce particular and important results in storage batteries, analogous results will be found in other electrolytic cells. Nor is this all. The study of the paths which the lines of current take, is just as important for electric furnace work. For it is clear that the whole problem is intimately connected with Ohm's law, and there is no essential difference between the application of Ohm's law to electrolytes and to metallic conductors. Mr. Schoop is right in saying that Ohm's law in its special form, as used for metallic wires (namely, e. m. f. = current multiplied by resistivity and length, divided by cross-section), cannot be applied to electrolytic cells. But the reason is simply the way how we usually arrange an electrolytic cell and its connections with the external circuit. Ohm's law in its most general form is equally valid for metallic conductors and for electrolytes.

* * *

The manifest advantage of determining and plotting dia-

grammatically the configuration of the system of the lines of current is that it enables us to see at a glance the variation of the current density throughout the cell, and it is an experimental fact that by varying the current density we are able to change to a certain extent the electrochemical action which will take place. In an electric furnace the reaction at a certain point depends on the temperature, and the latter depends on the watts consumed per cubic centimeter at that point, and these equal the square of the current density multiplied by the electric resistivity of that point. Since we have discussed in detail in an editorial note in our February issue the rôle which current density plays in electrolytic and in electric furnace processes, no further discussion of it is here necessary. We may, however, say that the application of Mr. Schoop's method of determining the flow of the lines of current would presumably be found specially useful in certain cases of electroplating and in electrolytic refining practice.

Professional Ethics.

Dr. Schuyler Skaats Wheeler, in his recent residential address before the American Institute of Electrical Engineers, discussed professional ethics—a subject most germane to the times. The past decade's great progress in industrialism has disturbed the settled relations of things. New and unlooked-for conditions have been obtruded into prominence. Such phenomena are described in two ways, according to the individual's hepatic efficiency. By the optimistic as "growin' pains," leading to a yet more splendid adolescence. By the pessimistic, as the rumbling of an eruption of a political Vesuvius. General considerations of ethical questions leave the greatest freedom for doubt and always lead to indefinite conclusions. The Latin dictum, "errare est humanum," is especially predicatory of personal and public morality. Where man, a social being, should guide his conduct purely from an egotistical standpoint, and where purely altruistic doctrines should sway him, is a hard question. This is due to his mixed nature just as is the distinction between the paternalistic theory of government as against the "laissez faire" theory. Mistakes are made in seeing the facts. Mistakes are made in reasoning from the observed facts. Mistakes are made from the happening of the "unexpected." The emotions and inherited instincts cause further mistakes. In short, there is the greatest chance for error, because we are not cold-blooded "reasoning machines," but plain human beings.

* * *

In the relation between client and technical advisor, mixed motives, differentiating between the engineer's personal interest and his fiduciary interest, are the chief causes of trouble. In America, we are frankly individualistic in our usual everyday life. In Europe, however, vestiges of the feudal relations are everywhere manifest. They are best exemplified in a new phase of life—the implicit confidence and trust the French inventor gives the French banker. Thus the American engineer has not the clean-cut and rigid system of ethics that the older professions have, because his is a newer calling in a newer land, and being so, is mainly individualistic. But this individualism must be counteracted by adherence to Lord Bacon's maxim: "I hold every man a debtor to his profession; from the which as men of course do seek to receive countenance and profit, so ought they of duty to endeavor themselves likewise to the help and ornament thereunto." This inspiring sentence rings as true to-day as it did four hundred years ago.

* * *

In metallurgical engineering the great changes now working havoc in the old order of things, the all-pervasive expansion with its consequent improvements and shifting of the engineer's connections, have naturally brought about unstable conditions. Young and ambitious engineers often designedly change positions in order to gain a wider practical experience, as does the German artisan in his "Wanderjahre." Not infrequently the young engineer is placed in charge of experimental work because of some special training and in spite of his enthusiasm. Experience thus gained is undoubtedly a form of capital in the intangible shape of a good memory, though quite tangible when it comes to the asking of good fees. This knowledge represents his professional stock-in-trade to a large extent, but should be left with his employer in the form of accurate data.

* * *

The direct relation between capitalist and engineer is the most important in the domain of professional ethics. The man of business has a superior knowledge of affairs and the human traits, all summed up in the words "commercial sagacity." This superior knowledge is sometimes used as was superior physical strength in an earlier era. And consequent to the widespread belief that "business success is often a survival of the slickest," there is a feeling on the part of the technical man that there is an unfair distribution of profits more legitimate than the same feeling exhibited by workmen. This results in a failure to reach the highest pitch of industrial efficiency. The course of Andrew Carnegie of taking up the practical men as protegés and partners was potent for the success of his plants. Well-measured enthusiasm cannot be conquered. Harmony and "team-play" are irresistible, and Mr. Carnegie's fairness and generosity to his engineers were more mighty factors of his victory over the Western steel plants than were his holdings of natural gas lands and coking coal fields.

* * *

The complexity of this simple subject compounds itself. But all can be reduced to the simple terms of Lord Bacon's maxim of professional indebtedness to the past knowledge which has made the profession possible. This calls upon the engineer to publish such well-digested opinions as professional papers before the proper technical society. Sometimes, of course, this is impossible. Where to draw the line between individual interests, the employer's interests and society's interests is hard, as it is always hard to draw any sharp line. Much is left to the "personal equation" and the ability to apply general principles derived of experience to a given case on which, in fact, success in law, medicine or any other profession depends. But in these days of rapid and sensitive public opinion, altruistic honor and "enlightened self-interest" seem hardly to differ. This is true no less of professional ethics than of all other questions of private and public morality, and is not inapplicable to life insurance standards.

The Iron and Steel Market.

At the few points where improvement was possible the American iron trade has shown it in the past two months. The entire market is now in as strong and healthy a condition as can be conceived. Erratic as the curve of market prosperity in the American iron trade appears, when but portions of it are viewed, it is a fact that a general law can be deduced to which the observations will conform with remarkable accuracy, and in opening this series of reports it is perhaps not unfitting that a few words should be devoted to this subject.

The law calls for a complete cycle in a period of twenty years, which may be condensed by special conditions. The war of the rebellion reduced the period to that from 1856 to 1873, but from the latter date to 1893 the full twenty years intervened. Market history since 1893 has most closely duplicated that which followed 1873. There is a period of stagnation of five or six years, prices finding their lowest level towards the close, as costs of production have been successively pared at this point and that. The curve of prices leaps suddenly upwards to a maximum for the whole period (1879 and 1899), and as suddenly falls almost to the minimum; it then rises much less steeply and to a somewhat less height, falling at about the same angle, to again turn upward at a smaller angle and to a less height, the waves thereafter being of less and less amplitude, until a great drop winds up the cycle. The fact which most impresses the one who observes it, and is the cause of the greatest confusion to the one who fails to discern it, is that the amplitude continually decreases while the wave length increases. The ordinary business man looks for a movement as great and as sudden as the immediately preceding one, and is disappointed.

In the second quarter of this year various slight disturbances occurred in the market, and each was taken in some quarters to presage a general decline, but they have all disappeared, and the present market is, as stated, thoroughly strong and healthy. It is not the time for either a great depression or a great advance.

Pig Iron.

The pig iron market during the second quarter of the year was characterized by weak spots, which developed here and there as to both time and place. During July the movement has been uniformly towards greater strength. In the South the market had broken to $13.00, f. o. b. Birmingham, for No. 2 foundry, and has reacted to $13.50, through large sales and a reduction in output caused by physical necessities. In the North, Bessemer, through the inability of merchant furnaces to buy additional Bessemer ores for this season, and basic, through the limited number of furnaces equipped with chill casting apparatus, have become very scarce, while foundry pig has strengthened in all Northern districts. The turn in pig iron was doubtless caused by a slight reduction in output, through the necessary blowing out of furnaces for relining, when many buyers expected an increase through the advent of new furnaces. Pig iron production during the first half of the year was very close to 12,500,000 tons; it was above that rate in March, and has since steadily, though but slightly, declined. Never before have American blast furnaces been subject to such steady market conditions, so as to develop the real capacity of the industry as a whole, which can now be accurately estimated at 25,000,000 tons per year; an abnormally small proportion was out for relining in March, while this midsummer the number is slightly above the normal. New capacity to come in is unusually small, until the large new plant of the United States Steel Corporation, at Gary, Ind., is heard from about the beginning of 1908.

Market prices f. o. b. Mahoning and Shenango Valley furnaces, and making the market for Pittsburg by the addition of the 85 cents freight, are: Standard Bessemer, $17.75; basic, $17.25; No. 2 foundry, $16.50; gray forge, $15.50.

Steel.

Bessemer and open-hearth billets are scarce, and are nominally about $27.50 for Bessemer and $28.50 for open-hearth, Pittsburg. Small billets are $30.00 and upward, and forging billets from $32.00 up. The supply of sheet bars is rather unsatisfactory, and the market is well held at $29.00, Pittsburg, for long bars and 50 cents advance for bars cut to specifications. The Carnegie Steel Company's surplus tonnage in billets and sheet bars is practically taken for the balance of the year on contracts. One cause of the scarcity of billets and sheet bars is the heavy demand for rails, requiring a larger tonnage than ever before from the interchangeable mills which roll rails, billets or sheet bars at will. The Carnegie Youngstown mill is scheduled on rails for the balance of the year, with a small proportion of billets, and no sheet bars, while at Mingo standard and small billets are being made to the almost total exclusion of sheet bars.

The Youngstown Sheet & Tube Company, which set out to build a complete Bessemer steel plant in the record time of twelve months to July 1 last, will bring its plant in about the middle of August. It is a standard two 10-ton vessel plant, with a large blooming mill on which no billets will be rolled; slabs will be taken to the company's present skelp mills, as a reheating proposition, and the blooming mill will also supply, for initial heat rolling, two new mills, a 4 x 4 billet mill and a mill to roll sheet bars and small billets. The finishing capacity is in excess of steel making capacity, and thus great flexibility is furnished. The management hopes shortly to reach 2,000 tons of ingots daily, although direct metal does not seem to be contemplated, and the effort will be watched with interest. The product for the remainder of the year seems to be disposed of.

Rails.

The Carnegie Steel Company is now practically sold for the balance of 1906, and was the last producer to fill up. It has about 250,000 tons sold for next year. The Illinois Steel Company has about all its 1907 production sold, perhaps 750,000 tons, while the Alabama and Colorado mills have sold more than half their product for 1907. Lackawanna, Pennsylvania and Cambria are sold in less proportion. Demand for rails in the West and South has increased much more rapidly than capacity in those districts. The price on Bessemer rails being $28.00 at mill, some Western roads have been forced to place orders in Pittsburg, and thus pay a higher delivered price than had the orders been acceptable to the Chicago mill. The new open-hearth steel mill in Indiana will hardly make rails before the season of 1908, but a large portion of its prospective output could have been sold for delivery this year or next. It will be able to turn more than a million tons a year of its steel into rails if required.

Finished Material.

In nearly all lines mills are assured of full operation the remainder of this year, with orders now on hand and business which is certain to float in. The greatest activity in July has been in line pipe and in plates and shapes for lake vessels, nine vessels having been placed with lake shipyards in two days.

The following prices are f. o. b. Pittsburg, plus regular freight to destination:

Beams and channels, 15 inches and under, $1.70; tees, $1.75; beams and channels over 15 inches, $1.80.

Plates, tank quality, ¼ inch and heavier, to 100 inches wide $1.60.

Sheets, 28 gauge, one pass cold rolled, $2.50.

Galvanized sheets, 28 gauge, $3.55.

Plain wire, $1.70.

Wire nails, keg, $1.85.

Tin plates, 100-pound cokes, per box, $3.75.

Fixation of Atmospheric Nitrogen.

Three interesting papers were recently presented on this problem, which is now no longer of purely theoretical interest, but has attained commercial importance both through the commercial success of the Birkeland-Eyde process in Norway (making nitrate fertilizers by means of arc discharges through air), and through the commercial success of the cyanamide process in Italy (making calcium cyanamide from calcium and nitrogen, the latter being obtained from atmospheric air by liquefaction and distillation).

Of the three papers to which we refer, two were presented by the inventors of the process just referred to. Prof. Kr. Birkeland, of the University of Christiania, read his paper before the Faraday Society in London; Prof. Ad. Frank, of Charlottanburg, lectured before the International Congress for Applied Chemistry in Rome. The paper by Prof. Birkeland shall be dealt with in this article, an abstract of the paper of Prof. Frank will be found in the Synopsis of our present issue on page 327.

The third paper, to which we referred above, was presented by Prof. Th. A. Guye before the Society of Chemical Industry in London. It was an elaborate presentation of the fundamental principles of physical chemistry on which the new industries are based; the content of the paper was essentially the same as Prof. Guye's article in our April issue, page 126, but further enlarged. We will refer to some new points of this paper in our next issue.

Prof. Birkeland's paper is very voluminous, and since the principle of the Birkeland-Eyde process and the main results have repeatedly been covered in our columns (*f. i.*, recently in our April issue, p. 126), we will give in full only those parts of the paper which describe the equipment of the Notodden works.

The alternating-current disc-plane, which is the essential feature of the Birkeland-Eyde process, was enclosed in a special furnace, which was lined with fire-brick, and furnished with a metal casing. Diagrams, two on page 399 of our Vol. II., and on page 126 of our Vol. III., show two such furnaces, one of an older, the other of a newer type. The fire-chamber of the furnace is narrow, in the direction of the lines of force—from 5 to 15 cm. wide—made partly of perforated chamotte air being conveyed to the disc-flame, in an evenly-distributed supply, through its walls. The system of magnets is composed of two powerful electromagnets, their extremities turned in towards the fire-chamber. The magnetic circuit is closed either as in a horse-shoe magnet, or through the shield-like, cast-steel casing of the furnace. The air is driven into the central region on both sides of the flame by gentle pressure from a Root's blower; and after passing in a radial direction arrives at a peripheral channel, whence it is conducted away. The horizontal electrodes are made of copper tubing, 15 mm. in diameter; whose terminals are within about 1 cm of one another, and are cooled by water in circulation, which keeps them from fusing. The electrodes are exchanged and repaired after being in use for about 300 hours, the exchange itself taking about 15 minutes to accomplish.

At the Notodden Saltpeter Manufactory there are three such furnaces in constant activity, each employing 500 kilowatts. These furnaces burn with an astonishing degree of steadiness, with a variation in energy of only 2 or 3 per cent, although the electrodes have no automatic regulation. It often happens that the assistant who attends to the furnaces does not need to touch any of them all through his watch; and it has happened that a furnace has burned for 40 hours without being attended to. The furnaces themselves give notice of anything going wrong, for the flame then begins to roar; and the warning comes in ample time to allow of an adjustment before the flame is extinguished. The furnaces work with a factor of reduction of 0.7. With a working potential of 5,000 volts, 3,500 volts and more are obtained on the electrodes.

As already mentioned the furnaces are lined with fire-brick. Even the large inner surfaces of the [illegible] have shown that they remain very stable, the reason of this [illegible] being that, in spite of the enormous heat [illegible] disc-flame, the temperature on the walls does not rise above 700° C during normal working, owing to the cooling effect of the current of air.

We have estimated that two fire-brick linings would be required per annum for each furnace; but this will certainly prove to be too high an estimate on the average, and the fire-brick furnaces will last longer than they are assumed to do. In the new saltpeter manufactory at Notodden, which is to obtain about 30,000 hp. from Svælgfos, 5 kilometers distant, it is decided that the furnaces shall be of 750 kilowatts under normal conditions, while during flood time they will be put up to 850 kilowatts. With a working tension of 5,000 volts the power factor[1] for these furnaces will be about 0.75.

We have had an opportunity of testing this type of furnace with as much as 1,000 kilowatts, but not yet for a sufficient length of time, as the alternator employed was too heavily laden.

The cost of these furnaces, including induction resistance, is at the present moment 18,000 krones, or $5,000, fully complete and capable of being immediately attached to an ordinary alternating current with a tension of 5,000 volts. The cost per kilowatt of the erection of furnaces can thus, at the present time, be put at 18 krones, or $5.00, which is, of course, surprisingly low.

There is reason to believe, however, that this low cost of erection per kilowatt will at some future time be further considerably reduced. From the experience gained up to the present, it would appear that in larger works the furnaces should be made to take larger quantities of energy than 1,000 kw. If, for instance, 2,000 kw. per furnace were absorbed, a considerable saving would be effected in the cost of erection and the working expenses.

There would be a gain in the power factor which would probably attain to 0.8, and there is no indication—if we draw conclusions by analogy from the experiments made with from 250 kw. to 750 kw. per furnace—of any loss in the output per kw.-year; on the contrary, a small gain might rather be expected there also.

Furnaces of 2,000 kw. such as these would cost, complete, between 20,000 and 22,000 krones, or about $5,500 to $6,050, including inductive resistance; so that the cost of erection would then be from 10 to 11 krones per kilowatt, an exceptionally low cost for the installation of electrical apparatus.

But it is now necessary to speak of the products of the process. The nitric oxide fumes which are formed in the furnaces and conducted away with the hot gases, fixate, after cooling, a further quantity of oxygen from the unconsumed part of the air, and turn into nitric peroxide; and this, when treated with water, combines to form nitric acid.

The volume of air thus treated at present in the Notodden Saltpeter Manufactory is 75,000 liters per minute; and as it contains only about 1 per cent of nitric oxide, it will easily be understood that in discussing the question of obtaining nitric acid from the air, it has frequently been pointed out that the great rarefaction of the gases to be worked with, greatly enhances the difficulties of the problem to be solved.

The gases that come from the furnaces with a temperature of 600° C. to 700° C. first pass through a steam boiler, the steam of which is employed in the further manufacture of the ultimate product, calcium nitrate. In the saltpeter manufactory now in course of erection the gases will be conducted directly through the evaporation tank, an arrangement which

[1] The author writes "factor of reduction" for [illegible] we call the power factor, i. e. the cosine of the angle of phase between current and electromotive force, or the factor by which volts and amperes must be multiplied to get the watts in an alternating current system.

implies such an important saving of heat, that the employment of coal may presumably be avoided.

After the gases are discharged from the above-mentioned steam boiler, their temperatures is reduced to 200° C., whereupon they are conducted through a cooling apparatus to be further cooled to about 50° C. The more the gases are cooled, the more easily are they absorbed by water. The gas then enters two large oxidation chambers with acid-proof lining. Here, as already mentioned, takes place the oxidation of the compound formed in the furnaces, nitric oxide becoming nitric peroxide, which is then conducted farther into an absorption system, where the gas is converted into nitric acid.

The absorption system consists of two series of stone towers, whose internal dimensions are 2 x 2 x 10 meters, each series containing five towers, two of granite and two of sandstone, filled with pieces of quartz, over which water, and the nitric acid formed, are made to trickle, while the fifth tower in each series is filled with ordinary bricks, over which trickles milk of lime. The milk of lime quickly absorbs the rarefied nitrous gases remaining, and is converted into a compound of calcium nitrate and calcium nitrite.

The first tower yields a 50 per cent nitric acid, the second about 25 per cent, the third 15 per cent, and the fourth 5 per cent. The liquids from the fourth tower are raised by compressed air to the top of the third, those from the third to the second, and those from the second to the first, thus gradually increasing in concentration up to 50 per cent, at which density the acid is conducted into a series of open granite tanks, where it is temporarily stored.

Some of this acid is employed in the decomposition of the nitrate-nitrite combination obtained by absorption by milk of lime. By the addition of nitric acid, the nitrous anhydride contained in the nitrite is driven out, and carried back to the system of towers.

The solution resulting from this, containing pure calcium nitrate, is carried, together with the rest of the stored-up acid, into another series of granite tanks, where this mixture of acid and acetous calcium nitrate lye, reacting on ordinary limestone, is converted into a solution of neutral calcium nitrate. This neutral lye is carried farther into vaporization-chambers of iron, where it is vaporized to a boiling point of 145° C., answering to a concentration of from 75 to 80 per cent of calcium nitrate, containing about 13.5 per cent of nitrogen. This substance is then run into iron drums containing about 200 liters, where it congeals, and in that form appears on the market.

Instead of vaporizing the calcium nitrate right to a boiling point of 145°, it may be allowed to crystallize after being vaporized to a boiling point of 120° C. The crystallized calcium nitrate is treated centrifugally, when a crystalline substance is obtained as the final product which is mainly used commercially as a fertilizer. Its composition is $Ca(NO_3)_2 + 4H_2O$, and is a salt with hygroscopic properties. In order, therefore, to make the salt more suitable for a manure it is converted into basic nitrate, which keeps dry, thus allowing of its being scattered with a sowing-machine. The manufacture of basic nitrate was suggested by Dr Rudolph Messel, of London.

A number of manuring experiments with calcium nitrate have been made at various agricultural institutes. The results show that lime saltpeter is quite as good as the natural saltpeter, and on sandy soil even superior to it. This last fact must be ascribed to the importance of the lime contained in the saltpeter to plants in soil that is so deficient in lime.

In order to obtain a satisfactory idea of the degree of success attending the solution of the nitrogen problem, it is necessary to know approximately the cost of production of calcium nitrate produced by the process here described.

The present cost is hardly likely to be published, but there are already available various official data from which fairly definite conclusions may be drawn.

At the opening of the new Chemico-Technical Institute at the Royal Technical College in Berlin, the director of the Institute (Prof. Otto N. Witt) spoke of the utilization of atmospheric nitrogen. In doing so he gave a detailed description of the Birkeland-Eyde process, which he knew from personal experience, having been one of the experts who visited Notodden. With regard to the practical output, he stated that, according to his observation, the yield was between 500 and 600 kilogrammes of anhydrous nitric acid per kw-year.

Now, Birkeland has found that the output of the furnaces is higher than 600 kilogrammes, if it is measured by analyzing samples of gas taken out, and including in the calculation the amount of air blown through the furnaces.[2]

The measurement of large volumes of air such as those in the present case requires large, expensive, accurate gas meters. They have used a "Duplex" from the "Cie. pour la Fabrication des Compteurs et Matériel d'usine a Gaz," of Paris, which can take a maximum of about 21,000 liters per minute. By special test determinations this meter, after being mounted in a separate little building, has proved to be correct to within ½ per cent.

The reason of the considerable difference observed between the output of the furnaces and the practical output has not yet been made quite clear, and investigations into this matter are, therefore, still being persevered with.

In the estimate for the new factory of about 27,000 e. hp. at Notodden by a commission of famous foreign experts, among whom were Grandeau, Schloesing, Silvanus P. Thompson, Turrettini and Otto N. Witt, a practical output of 500 kg. of anhydrous nitric acid per kw-year has been taken for granted.

In the prospectus issued on the formation of the company that now owns the works just named, it was stated that the cost of production per ton of calcium nitrate, containing 13.2 per cent of nitrogen, would be 72.90 krones ($20.00), while the selling price per ton is put at 145.20 ($40.00), this having been reckoned from the present price of 1,100 krones (about $300) per ton of combined nitrogen.

As the building of the new factory is already far advanced, there has been ample opportunity of seeing that the cost of the works has been very correctly estimated.

In the concluding part of his paper the author discusses the theory of the oxidation of nitrogen in electric arcs.

CORRESPONDENCE.

Kryptol.

To the Editor of Electrochemical and Metallurgical Industry:

Sir:—From Mr. F. A. J. FitzGerald's letter on page 210 of your June issue, it appears that there is still some question as to the composition of "kryptol." The following facts may, therefore, be of interest:

Two samples of this material were recently received from different agents of the "Kryptol Gesellschaft" in this country.

On the assumption that "krptol" was composed of graphite mixed with various proportions of carborundum (the two samples were for use with different voltages), they were submitted to analysis and found to contain 0.23 per cent and 0.27 per cent of incombustible matter respectively. These samples of "kryptol" were, therefore, similar to those examined by Mr. FitzGerald.

I may add that the second sample contained, beside 3.80 per cent of moisture, 95.52 per cent of carbon (weighed as CO_2), hence it is obvious that the material under examination contained no admixture of sand, clay, etc.

London, England. Richard Seligman.

[2] Mr. Edstrom, at the St. Louis Congress (our Vol. II., page 399), stated that one kw year, measured at the arc, yields 980 kg. of HNO_3.

Electrolytic Precipitation of Gold From Cyanide Solutions.

By Prof. B. Neumann, Ph. D

The cyanide process for the extraction of gold has been in use on a large scale for about 15 years. The process was developed mainly in Transvaal, but is now employed in all gold districts. It has increased in importance to such an extent that more than one-third of the world's production of gold is now obtained by its means.

The gold is precipitated from the cyanide solutions mainly by two different methods, either by zinc precipitation or by electrolysis.

Electrolytic precipitation is generally carried on by passing the electrolyte successively through several tanks, in which a great number of lead-foil strips are suspended, being connected in parallel and used as cathodes, while iron sheets are employed as anodes. The voltage at the terminals of the path is 2 to 3 volts, the best current density about 0.5 amp. per square meter, whereby the gold is precipitated in an adherent layer on the cathodes.

In the course of time, prussian blue and iron oxide are formed at the iron anodes, whereby the electrolyte becomes impure. Andreoli proposed, therefore, to substitute lead peroxide electrodes for the iron anodes, since with the former no corrosion can take place and the baths remain pure. This proposition looks undoubtedly promising at first glance. Information on the success obtained with such anodes on a large scale is, however, scarce. According to one statement from South Africa these electrodes have not proven satisfactory, on account of impurities in the electrolyte. But this is certainly not the real reason, as is proven by some experiments of the author.

Lead electrodes were electrolytically coated with peroxide according to the method of Andreoli. In this way very thin as well as pretty thick coatings were produced, and peroxide plates made by other methods were also employed for the experiments, among them one plate which had been used for several months as anode in the electrolysis of sulphuric acid.

If such plates are used as anodes in dilute potassium cyanide solutions, the same phenomenon is always observed. After a short time, at various places, white points appear, which then change to a thick white fog, passing down in the bath and fouling it completely. The white substance is lead cyanide, formed by corrosion of the anode. This is due to the fact that all electrically-produced coatings of peroxide are porous, so that the anion passes through the pores and attacks the metallic lead below the peroxide. If, on the other hand, the peroxide coating is made very thick, cracks will occur in time even if great care is taken, and the electrolyte passes through the cracks and again reaches the metallic lead. At least the author did not succeed by any method in making peroxide anodes which would have fulfilled the requirements of practice. This explains why peroxide anodes have not been generally introduced anywhere in the electrolytic precipitation from cyanide solutions.

In accordance with the results of the author is an observation of Hamilton[1] on experiments with lead peroxide anodes in Sonora, Mexico. Concerning the construction of his electrode see the article of Hamilton. His experience is that when once the surface begins to disintegrate deterioration is comparatively rapid.

In spite of the extended use of the electrolytic precipitation process on a large scale, little reliable information is available on details of the process. The figures given for the ampere-hour efficiency of the electrolytic precipitation process differ very greatly. In a discussion of a paper of Walker, Richards[2] remarked that in the Stanley & Ivanhoe process the efficiency is 100 per cent, according to experiments made in his laboratory.

On the other hand, Hamilton[3] states that in Sonora, Mexico, in runs extending over two months, the "waste of current" in one case was 87 per cent, and in another 92 per cent, so that the corresponding efficiencies are 13 and 8 per cent. (I do not understand, however, that the solutions contained considerable quantities of silver besides gold; 20 cent. solution 4.54 grams gold per 100 grams silver, gold to silver ratio as 1:7 gram gold with 43.5 gram silver), on the other hand, the current density used was 5.5 and 3.1 amp. per square meter respectively, which is much higher than the usual current density.

Christy[4] also uses higher current densities. According to his opinion the usual method has an efficiency of 1 to 2 per cent, and certainly not more than 5 per cent, while he tried to get an efficiency of 80 per cent by a special construction of the electrodes.

In a discussion of precipitation of gold from cyanide solutions, Sharwood[5] gives the efficiency at from 6 to 12 per cent, but is cautious enough to say that "this is said to be obtained in practice." According to his statement the precipitation of gold from 100 tons solution in 24 hours (4 grams of gold, 10 grams of silver per ton) requires 5 to 6 hp. On the basis of this statement the efficiency of the precipitation of the gold alone comes out as 2.4 per cent.

In view of these contradictory statements it did not appear superfluous to make some experiments in order to determine the ampere-hour efficiency which can be obtained in the electrolysis of dilute cyanide solutions when the current density or the concentration or the length of the run are changed.

Two runs were first made in which the current density, as usual in practice, was 0.5 amp. per square meter. In one run the solution contained 10 grams of gold per metric ton (cubic meter), the other solution contained 3 grams. These experiments were made to get some idea of the efficiency of the different tanks connected in series in practice. Later on other runs were made with lower and higher current densities.

Since the current density is very low, even small-scale experiments require rather large electrode surfaces, in order to avoid the necessity of using very small currents. In the present case a glass vessel was used of 22 cm. length, 30 cm. height and 12 cm. breadth, which contained about 8 liters of electrolyte. Thin iron sheets and thin lead sheets were used as electrodes, the dimensions being 27 by 21 cm. The electrode surface within the electrolyte was 21 by 24 cm. (*i. e.*, 504 square centimeters on one side, or 1,008 square centimeters on both sides), the quantity of electrolyte being 6.5 liters. Two lead cathodes were always used between three iron anodes. The useful lead surface was, therefore, 2,000 square centimeters, and a current of 0.1 amp. represented a current density of 0.5 amp. per square meter.

By an electrically-driven stirrer, running at a low speed, the electrolyte was uniformly agitated. This was done since I had found that with non-uniform stirring with a water turbine the deposition of the small quantities of gold per unit of time was considerably influenced.

In order to imitate as much as possible the conditions in practice where the concentration in the different tanks remains always the same, due to continuous supply of new solutions, I also added in my experiments as much gold to the electrolyte in form of concentrated potassium cyanide solution as had been precipitated by the current. In this way the quantity of potassium cyanide which had been decomposed by the current was also replaced.

The experiments became complicated, since it was not possible to weigh the cathodes after a certain time given the

[illegible]

[1] "Electrochemical Industry," 1904, Vol. II., pp. 131 and 372.
[2] "Electrochemical Industry," 1903, Vol. I., p. 484.
[3] "Electrochemical Industry," 1904, Vol. II., p. 134.
[4] U. S. [illegible] 750,928 of May 26, 1900.
[5] "[illegible] and Mining Journal," 1901, I., 152.

bath and weigh them. Since each lead cathode weighed 125 to 150 grams, the result of weighing would have been inaccurate, as the quantity of gold deposited is often only a few milligrams. But a greater difficulty is due to the fact that the plates, when dry, are covered with an oxide film and with traces of lead cyanide, the weight of which may be as much as that of the deposited gold. The chief reason against the usual method of weighing and again using the same cathodes was, however, as follows:

From the experiments it is seen that in the first hours relatively small quantities of gold are deposited on the lead plates. The quantity deposited per unit of time increases with proceeding electrolysis up to a maximum. If the electrolysis is then interrupted, and if the cathode is weighed and again suspended in the bath, we do not get in the next few hours the increased efficiency which we would have obtained in the hours following the first run if no interruption had occurred, but we get only the low efficiency again which was obtained in the very first hours of electrolysis.

Thus, in an experiment 0.2200 grams of gold were deposited in 8 hours of continuous electrolysis and 0.2780 gram in 10 hours, the process having not been interrrupted. The deposit of the ninth and tenth hours amounted, therefore, to 0.0580 gram. When, however, the electrolysis was interrupted after 8 hours, the plate was weighed and again suspended in the bath, and then the electrolysis continued, the next 2 hours yielded only 0.0155 grams of gold, which quantity is very nearly that obtained in the very first 2 hours of the experiment.

This experiment shows that the lower efficiency in the beginning of the experiment is intimately connected with the surface condition of the lead cathodes. For this reason it is impossible to use again the plates after they had been weighed.

The experiment was, therefore, made in such a way that always new plates were used and were in continuous operation for the whole time. For instance, the first plates for 2 hours, the second plates for 4 hours, and the third ones for 6 hours, etc., so that an experiment on continuous electrolysis for 10 hours required really 30 hours. The quantity of gold was not determined by weighing, but by the most exact method in the dry way, by cupellation in the muffle, whereby the quantity of silver was, of course, separately determined.

The results of some such experiments are given in the following table:

Table I.

Current density=0.5 amperes per square meter; 10 grams gold per cubic meter; 0.5% potassium cyanide.

Hours.	Current, Amp.	Ampere-minutes.	Voltage.	Deposited Gold in Grams.	Ampere-hour Efficiencies in Per cents.
½	0.1	3	0.8–1.7	0.0012	0.33
1	0.1	6	1.5	0.0068	0.92
2	0.1	12	1.2–2.2	0.0264	2.06
3	0.1	18	1.9–2.1	0.0647	2.93
4	0.1	24	1.9–2.2	0.1027	3.49
5	0.1	30	1.7	0.1403	3.81
6	0.1	36	1.9–2.0	0.1661	3.77
8	0.1	48	1.7–1.7	0.2089	3.55
14	0.1	84	1.7–1.9	0.3262	3.17

The figures for the efficiency refer to the whole time. They show that the efficiency becomes a maximum at a certain time and decreases again later on. At the end of this experiment 0.025 per cent free potassium cyanide was still in the solution. Quite some potassium cyanide had been destroyed during the experiment by electrolysis and air. The voltage increases with decreasing percentage of potassium cyanide.

Table II.

Current density=0.5 amp. per square meter; 3 grams gold per cubic meter.

Hours.	Current Ampere.	Ampere-minutes.	Volts.	Deposited Gold in Grams.	Ampere-hour Efficiency in Per Cents.
1	0.1	6	0.6–1.9	0.0005	0.007
2	0.1	12	1.9	0.0101	0.69
3	0.1	18	1.85	0.0195	0.88
4	0.1	24	2.0	0.0307	1.08
6	0.1	36	2.1	0.0559	1.26
8	0.1	48	2.1	0.0781	1.33
12	0.1	72	1.9–2.0	0.1160	1.31

The results of both tables show that a maximum of efficiency is obtained after some time, and that even with constant concentration of gold the efficiency decreases again slowly when the run is continued for a longer time. The experiments also show that by employing current densities, such as used in practice, the ampere-hour efficiency is surely not more than 4 per cent. even with rich solutions, but that on the contrary in longer runs, and by using the same cathodes for several weeks, the ampere-hour efficiency will be in the average considerably below this figure. In electrolyzing poorer solutions, such as in Table II., the maximum of efficiency reached is not more than 1.33 per cent. The total average of the efficiency in a plant with several cells in series will, therefore, not be more than, say, 2 per cent, but will remain considerably below 1 per cent if the run is continued for a long time, as was also found by us by special experiments.

It is evident that with smaller current densities a better current efficiency could be obtained. For practical purposes, however, the current then becomes too small; *i. e.*, the deposit per unit of time is too small. The solutions must be passed through the cells at a lower speed or it becomes necessary to connect more cells in series.

To get more information on these points two runs were made, the results of which are given in Tables III. and IV.

Table III.

Current density=0.25 amp. per square meter; 10 grams gold per cubic meter; 0.05% of potassium cyanide.

Hours.	Amperes.	Ampere minutes.	Volts.	Deposited Gold in Grams.	Ampere-hour Efficiency in Per Cents
2	0.05	6	0.5	0.0049	0.67
4	0.05	12	1.5	0.0485	3.30
6	0.05	18	1.4	0.1270	5.75
8	0.05	24	1.3	0.2200	7.48
10	0.05	30	1.20	0.2780	7.56
12	0.05	36	1.15	0.3267	7.40
14	0.05	42	1.05	0.3737	7.25

Table IV.

Current density=0.25 amp. per square meter; 3 grams gold per cubic meter; 0.05% of potassium cyanide.

Hours.	Amperes.	Ampere-minutes.	Volts.	Deposited Gold in Grams.	Ampere-hour Efficiency in Per Cents.
4	0.05	12	1.7	0.0056	0.38
6	0.05	18	1.7	0.0341	1.54
8	0.05	24	1.6	0.0666	2.26
10	0.05	30	1.55	0.1046	2.84
12	0.05	36	1.50	0.1346	3.05
14	0.05	42	1.40	0.1624	3.15

A glance at the Tables III. and IV. shows that with smaller current densities considerably better results were obtained than in the first two runs. In the richer solution the maximum efficiency was 7.50 per cent, in the poorer solution somewhat more than 3 per cent.

The gradual decrease of the voltage in Tables III. and IV. is explained by the fact that the greater deposition of gold requires replacing greater quantities of gold, whereby more potassium cyanide was introduced into the solution than was destroyed by the current in the same time.

We now give several runs in which higher current densities than in Table I. and II. were used.

TABLE V.

Current density 2.4 amp. per square meter, 10 grams gold per cubic meter; 0.07% of potassium cyanide.

Hours.	Minutes.	Amperes.	Ampere-minutes.	Volts.	Deposited Gold in Grams.	Ampere-hour Efficiency in Per Cents.
..	20	0.12	2.4	1.8–2.2	0.0004	0.13
1	15	0.12	9.0	2.2–2.3	0.0045	0.41
2	..	0.12	14.4	1.8–2.2	0.0064	0.36
3	30	0.12	25.2	2.1–2.2	0.0079	0.26
7	..	0.12	56.4	2.2	0.0084	0.12

TABLE VI.

Current density=4 amp. per square meter, 10 grams gold per cubic meter; 0.07% of potassium cyanide.

Hours.	Minutes.	Amperes	Ampere-minutes.	Volts.	Deposited Gold in Grams.	Ampere-hour Efficiency in Per Cents
..	5	0.20	1	2.5	0.0001	0.08
..	20	0.20	4	2.5	0.0004	0.08
..	40	0.20	8	2.5	0.0015	0.16
1	..	0.20	12	2.5	0.0036	0.24
2	..	0.20	24	2.5	0.0058	0.20
3	..	0.20	36	2.5	0.0066	0.15
4	..	0.20	48	2.5	0.0068	0.11
6	..	0.20	72	2.5	0.0073	0.08

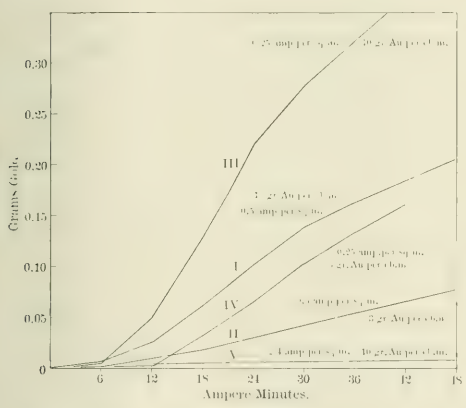


FIG. 1.—EFFECT OF CURRENT DENSITY ON DEPOSITION OF GOLD

TABLE VII.

Current density=9 amp. per square meter, 10 grams gold per cubic meter; 0.07% of potassium cyanide.

Hours.	Minutes.	Amperes.	Ampere minutes.	Volts.	Deposited Gold in Grams.	Ampere-hour Efficiency in Per Cents.
..	15	0.45	9.	2.8–3	0.0004	0.05
..	40	0.45	18.	3.0	0.0035	0.16
1	10	0.45	31.5	2.8–3	0.0051	0.13
3	5	0.45	83.3	3.0	0.0062	0.06
5	15	0.45	141.0	3.0	0.0069	0.04

The runs V., VI. and VII. were made in a smaller vessel containing 1.25 liters of gold solution with a lead cathode surface of 500 square centimeters. The electrolyte was agitated by means of a stirrer. The temperature was the ordinary temperature of the room. Gold was always present in excess, but was not regularly replaced.

The results show that with higher current densities the ampere hour efficiency becomes very poor, and do not even reach ½ per cent.

If we give a summary of the best results obtained in the different runs with the same concentration, we get the following:

At 0.25 amp. per sq. meter, the max. amp.-hour efficiency was 7.56 per cent.
" 0.50 " " " " " " " " 3.84 "
" 2.40 " " " " " " " " 0.41 "
" 4.00 " " " " " " " " 0.24 "
" 9.00 " " " " " " " " 0.16 "

In a long run with 2.5 amps. per square meter, extending over a month and a half (with 9 grams of gold per cubic

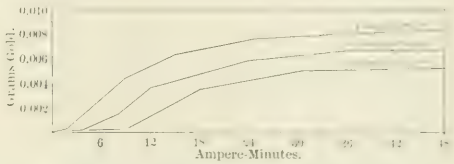


FIG. 2.—EFFECT OF CURRENT DENSITY ON DEPOSITION OF GOLD.

meter, and 0.05 per cent potassium cyanide), the average amprere-hour efficiency was 0.12 per cent, the voltage being 2.3 to 2.6.

The average ampere-hour efficiency in the electrolytic precipitation of gold from cyanide solutions on a large scale in runs extending over a longer time is, therefore, usually below 1 per cent.

In the adjoining diagrams the results are plotted graphically. Fig. 1 shows the increase of the deposition of gold under the different conditions in the course of time. Fig. 2 shows the same results on a larger scale for the runs V. to VII. In Fig. 3 the ampere-hour efficiency is plotted as obtained during the progress of electrolysis in the different solutions with the different current densities.

* * * * *

In the Siemens & Halske process strips of lead foil are used as cathodes. Lead has the advantage that the whole electrode with its gold deposit may be molten, and the gold then be easily separated from the lead.

Pfleger proposed to use fabrics of lead wire in screen form through which the electrolyte was intended to flow. This modification has not been introduced into the practice.

Andreoli proposed iron and aluminium for the cathode, and suggested to plunge the cathodes with their gold deposits into a molten lead bath which would alloy with the gold. Experiments in the laboratory gave good results, but on a large scale the method failed.

While, as stated, very low current densities are used in prac-

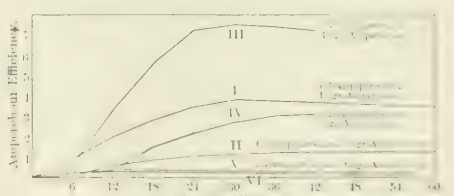


FIG. 3.—EFFECT OF CURRENT DENSITY AND CONCENTRATION ON EFFICIENCY.

tice in order to get a firmly adherent deposit of gold, there are other cases where high current densities are intentionally employed. Hamilton[6] states that at Sonora current densities of about 2.5 amps. per square meter are used with the intention of depositing the gold in the form of a fine slime, which

[6] "Electrochemical Industry," Vol. II, [illegible]

is then treated in the same way as the similar product from the zinc process. The lead foil was brushed off every day or two, but the lead foil suffered thereby. Tin plate instead of lead foil proved very successful later on.

Various inventors have proposed to use mercury as cathode (Locoy, Eltonhead, Stewart and others). The separation of the precious metals from the mercury could be performed in a very simple way. The same idea is at the bottom of the construction of those apparatus in which leaching and precipitation of gold are carried out in one and the same receptacle (Riecken process, Pelatan-Clerici process, Aurex-Sluice, etc.). In the latter forms of apparatus mercury absorbs relatively little gold, but it may be assumed that a moved mercury mass could alloy with quite considerable quantities of gold, so that the mercury cathode would show certain advantages over a lead cathode which can remain in the bath for one or two months only.

However, von Gernet[1] has shown that mercury electrodes are not practicable on a large scale. For instance, in order to precipitate within 24 hours the gold from 100 tons of cyanide solutions containing 5 dwt. (7.775 grams) per ton, a mercury surface of 10,000 square feet (900 square meters) would be required. In order to cover the whole bottom of the cell the mercury layer would have to be at least ¼ inch (0.6 centimeter) thick; therefore at least 250 cubic feet (5.4 cubic meters) of mercury would be required, that is, about 75 tons. In this calculation, von Gernet evidently has assumed double the usual current density. The cost of such a quantity of mercury alone would prevent the application of mercury as cathode even if we do not consider the losses of mercury during operation.

A proposition of Christy[2] is to the effect to deposit the precious metals from cyanide solutions in the usual way upon iron or lead cathodes, and to use these electrodes when covered with gold afterwards as anodes in a stronger cyanide solution, and to dissolve the gold from them and redeposit it on gold cathodes. This proposition has been claimed to be impracticable by von Uslar and Erlwein[3]. First, because too much dead capital is required in form of gold cathodes, and, secondly, because it would be difficult to control the process so that there are no losses of current or electrode material.

It is true, Christy's proposition has some weak points, but certainly it does not fail for the reason given by von Uslar and Erlwein. Since electrolytic refining of gold is used on a rather large scale in practice, it is proven that the quantity of gold held back in form of cathodes in the cells is no economical obstacle to carrying out the process. On the other hand, with such an expensive metal as gold a low ampere-hour efficiency is of not very great importance, especially as gold is monovalent in cyanide solutions, so that 7.356 grams are deposited per ampere-hour.

The economical question is, whether the cost of the treatment of the lead cathodes, with their gold deposit, is cheaper than the electrolytic dissolution of gold from an anode and redeposition on a gold cathode.

At the suggestion of Prof. Dieffenbach, experiments have been made in this laboratory which are based on a somewhat similar idea. We intended to combine the transportation of the gold from anode to cathode with simultaneous refining. If the desired purpose was to be attained it was necessary that the layer of gold should be easily dissolved from the anode (which had first served as cathodes) without the electrodes being attacked or destroyed in any way when being used.

The first question to be answered is whether the lead electrodes generally used as cathodes for gold deposition are suitable for this purpose. Hydrochloric acid and chlorides cannot be used in this case as electrolytes for the subsequent refining action, since lead is attacked by them when used as anode. Potassium cyanide solutions are also unsuitable, as has been established by experiments, since lead cyanide is formed at the anodes, which makes the electrodes unsuitable for further use. The proposition of Christy of using lead electrodes in potassium cyanide solutions for this purpose is, therefore, impracticable. The same must be said concerning the iron anodes recommended by him, since iron oxide and prussian blue are formed anodically, which render impossible the deposition of gold on such plates when used as cathodes later on. Lead and iron are, therefore, impractical for the desired double purpose.

With respect to the use of carbon, various practical men have stated that carbon cannot be employed well as anode in potassium cyanide solutions, since it is soon destroyed. However, the amount of destruction certainly depends much on the nature of the carbon. On the other hand, carbon, when used as cathode in a potassium cyanide solution, should be quite durable, since it is covered in a very short time with a layer of gold. As a matter of fact a carbon electrode, when used in a run extending over one and a half months with very dilute potassium cyanide solutions, proved quite suitable. Acheson graphite is excellent for this purpose. Gold is deposited on carbon quite uniformly and densely, and much higher current densities may be used than are otherwise employed.

The following experiments were made by Mr. Johnny Johnson:

Carbon plates were first covered with gold. For this purpose the carbon cathodes were placed in a glass vessel containing an electrolyte with 0.0269 gram of gold and 1.52 potassium cyanide per liter. Electrolysis was carried out for 15 hours with a current of 0.15 to 0.2 amp., the electrode surface being 225 square centimeters, the voltage 2. The gold was deposited in adhering form.

When these electrodes had thus been covered with gold they were placed as anodes in a glass vessel containing 1½ liters of rather concentrated potassium cyanide solution. Gold-plated copper sheets were used as cathodes. Table VII. shows the results obtained in eight experiments with different current densities.

Table VII.

Hours.	Amperes.	Copper in Voltameter.	Cathode Surface in Sq. Centimeters.	Amperes per Square Meter.	KCN per Liter.	Deposited Gold in Grams.	Ampere-hour Efficiency in Per Cent.
½	0.496	0.1465	5.0	992	177.6	.....	....
½	0.329	0.0972	5.0	659	177.6	.....	....
1	0.325	0.3789	11.0	296	135.	.....	....
1	0.334	0.3959	28.3	118	135.	.....	....
16	0.200	.....	28.8	69	135.	0.0065	0.03
3½	0.164	0.6782	28.8	57	121.	0.0062	0.15
3	0.084	0.3034	28.8	30	121.	0.0065	0.34
4	0.070	0.3286	28.8	24	121.	0.0092	0.45

From this table it is seen that at high-current densities no gold whatever was deposited, but even at lower current densities the quantity of the deposit and the ampere-hour efficiency were so low that the use of potassium cyanide solutions for the purpose of transmitting gold from the anode to the cathode is practically impossible.

The gold dissolved from the anode very quickly. In the last experiments the gold dissolved in such a loose form that it could be rubbed off with the finger. The experiments 3 to 5 were made with an electrolyte which contained gold at the start, but these experiments do not show any difference from the others.

For the transportation of the gold from the anode to the cathode we then used gold chloride solutions instead of potas-

[1] Proceedings Chem. and Metallurgical Soc. of South Africa, Vol. I.
[2] U. S. patent 683586.
[3] Mitteilungen über Forschungsarbeiten, Elektrochemie, Vol. VII.

sium cyanide solutions, whereby the experience of Wohlwill[1] on gold refining could be made use of.

In order to accelerate the deposition of the gold on the carbon plates we used a gold chloride solution containing free hydrochloric acid as electrolyte instead of potassium cyanide. A gold-plated carbon or platinum electrode was the anode. The electrolyte was heated to 60° or 70° C., and was agitated by means of a stirring device. The cathode surface in contact with the electrolyte was 10 square centimeters, the current 0.4 amp., the current density, therefore, 400 amps. per square meter. The gold was deposited with a good ampere-hour efficiency in such adhesive form that a strong stream of water did not tear off anything. The deposit was smooth in the beginning, but later on warts appeared.

The same apparatus was used for transporting the gold from the anode to the cathode. Fig. 4 shows the arrangement of the experiments. B is the storage battery, Z the cell with stirrer, V a voltmeter, A an ammeter, K a copper voltameter, R a regulating resistance and W a water rheostat.

The temperature of the electrolyte was maintained at 60° to 75° C. The anodic current density was 1,000 to 1,500 amps. per square meter. The cathodic current density varied in the different experiments. A platinum plate was used as cathode.

Several series of tests were carried out.

A. Experiments With Dilute Solutions.

The electrolyte consisted of an aqueous solution of gold chloride with 3 per cent free hydrochloric acid. The content of gold varied somewhat. The current density was varied between 140 and 1,000 amps. per square meter. The voltage at the terminals was 0.2 to 0.4.

Table IX.

Hours.	Minutes.	Amperes.	Copper in Voltameter.	Cathode Surface in Sq. Centimeters.	Amperes per Square Meter.	Gold per Liter of Electrolyte. before & after Experiment.		Deposited Gold in Grams.	Calculated Ampere-hour Efficiency in Per Cent.
..	10	0.33	.....	5.00	1000	22.05	....	.....	.....
..	15	0.53	0.0962	6.25	850	4.45	....	.....	.....
1	30	0.44	0.7828	8.75	503	4.70	1.15	2.3430	145.0
..	54	0.42	0.4450	10.00	418	4.75	3.45	1.3748	150.0
..	45	0.31	0.2796	10.00	310	7.25	6.10	0.9842	170.0
1	20	0.23	0.3668	10.00	233	3.00	2.60	1.3243	175.0
1	35	0.14	0.2654	10.00	141	3.30	2.85	1.0806	198.0

In the first two experiments of this table the electrolysis was soon interrupted, since the gold was deposited as a dark green slime at the cathode. In the third experiment it also adhered very poorly. The lower the current density the better became the adhesiveness of the gold to the cathode and the lighter became its color.

The explanation of the abnormally high figures for the current density is that in calculating the same the gold was assumed to exist in the chloride solutions as a three-valent ion, while as a matter of fact part of it migrates as monovalent ion, as will be discussed later.

The ampere-hour efficiency becomes also better with decreasing current density. It is also remarkable that the solution was always poorer in gold after the experiment than before. That means that more gold is deposited on the cathode than is dissolved from the anode, and the loss of gold from the electrolyte is greater at higher current densities than at medium ones.

Experiments were then made with a similar electrolyte, to which, however, common salt was added to increase the conductivity.

[1] Wohlwill, Zeit. f. Elektrochemie, 1903, Vol. IV., page 381; see also Tuttle, "Electrochemical Industry," Vol. I., page 157, and Wohlwill, "Electrochemical Industry," Vol. II., pages 221 and 261.

B. Experiments With Dilute Solutions With Addition of Sodium Chloride.

The apparatus was the same as before. The electrolyte contained 30 grams of hydrochloric acid per liter, 200 grams of sodium chloride and 7.5 grams of gold. On account of the better conductivity, the voltage was somewhat lower than before. The conditions of the expermients and the results may be seen from the following Table X.

Table X.

Hours	Minutes	Amperes.	Copper in Voltameter.	Cathode Surface in Sq. Centimeters.	Current Density in Amperes per Square Meter	Gold per Liter of Electrolyte. before & after Experiment		Deposited Gold in Grams.	Calculated Ampere hour Efficiency in Per Cent.
..	45	0.498	0.4411	10.0	498	7.5	3.00	0.9988	109
..	35	0.424	0.2926	10.0	424	7.5	3.50	0.7068	117
..	40	0.324	0.2559	10.0	324	7.5	3.85	0.6842	129
..	45	0.221	0.1962	10.0	221	7.5	4.25	0.5673	140
1	10	0.134	0.1848	10.0	134	7.5	4.48	0.5621	147

The deposits of gold in this experiment were much more solid than in the former experiments, especially at lower cur-

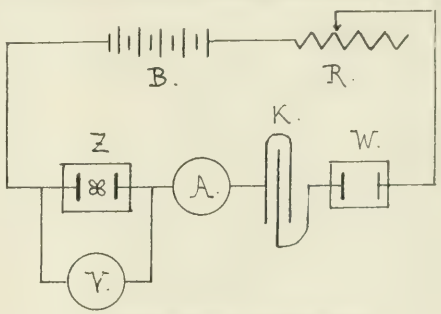


Fig. 4. Arrangement of Experiments.

rent densities. The color of the deposit was a light yellow. The quantities of gold deposited have become less, and the loss of gold from the electrolyte is less than before. If little gold is only left on the anode, chlorine is also set free.

Some experiments were then made in which some Kieselguhr was added to the chlorine solutions containing free acid. The object was to remove the hydrogen bubbles and to increase the density of the deposit.

C. Experiments With Gold Chloride Solutions Containing Hydrochloric Acid With Kieselguhr.

A solution was first made up of the same composition as in the experiments of series A. Kieselguhr slime was then added. The addition amounted to 1 gram of Kieselguhr per 150 cubic centimeters of solution. The more coarse particles of Kieselguhr removed not only the hydrogen bubbles but also separated the gold deposit from the cathode. The following experiments were made with the finer particles of Kieselguhr.

Table XI.

Hours	Minutes	Amperes.	Copper in Voltameter.	Cathode Surface in Sq. Centimeters.	Current Density in Amperes per Square Meter	Gold per Liter of Electrolyte. before & after Experiment		Deposited Gold in Grams.	Calculated Ampere hour Efficiency in Per Cent.
.	50	0.514	0.5057	10.0	514	7.5	3.20	1.4142	135.0
..	55	0.394	0.4275	10.0	394	7.5	3.90	1.2765	144.5
1	7	0.309	0.4080	10.0	309	7.5	4.05	1.2251	145.4
1	15	0.214	0.3161	10.0	214	7.5	4.22	0.9929	152.0
2	25	0.095	0.2714	10.0	95	7.5	4.40	0.9802	174.5

The condition of the deposits was better than in the experiments of series A, but the deposits were not as solid and dense as in series B. As before, the deposits improved with decreasing current density. Evidently small particles of gold were also torn off by the Kieselguhr, so that the efficiency is somewhat lower than with the solutions of the same composition in series A. The deposit of gold also contained some Kieselguhr.

Since, according to Wohlwill, an electrolyte which contains 25 to 30 grams of gold per liter is best suited for gold refining, some experiments were made with more concentrated gold solutions. One experiment was made without any addition, the other with the addition of common salt to increase the conductivity.

D. Experiments With Concentrated Gold Solutions.

The electrolyte contained 23.3 grams of gold and 30 grams of hydrochloric acid per liter in the first experiment, and 29.15 grams of gold, 30 grams of hydrochloric acid and 200 grams of sodium chloride in the second experiment. The voltage was 0.3 in the first case and 0.2 in the second case.

Table XII.

Hours.	Amperes.	Copper in Voltameter.	Cathode Surface in Sq. Centimeters.	Current Density in Amperes per Square Meter	Gold per Liter of Electrolyte. before & after Experiment.		Deposited Gold in Grams.	Ampere-hour Efficiency in Per Cent.
1	0.344	0.4058	10.0	344	23.3	22.4	1.1508	137
1	0.352	0.4156	10.0	352	29.15	27.8	1.1605	135

The content of gold in the solution decreased little in comparison with the concentration, but the absolute quantity which disappeared from the solution was about the same as before.

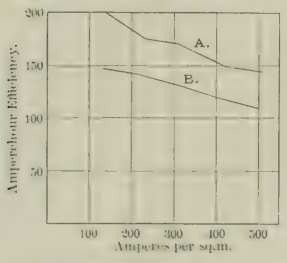


Fig. 5.—Efficiency as function of current density

The ampere - hour efficiency in the last experiment is the same as in the corresponding experiment with dilute solutions. In the preceding experiment, however, the ampere-hour efficiency is much lower than in the corresponding experiment with dilute solutions. The deposits from concentrated solutions are considerably more dense and solid than those from dilute solutions. Especially the last deposit was excellent in various respects of quality.

By plotting the results of the different series of experiments graphically, the diagram of Fig. 5 is obtained. The results of series A and B only were plotted, since series C showed the same character but exhibited some irregularities on account of the content of Kieselguhr. The curves show clearly how the ampere hour efficiency decreases in all electrolytes with increasing current density.

A few words may now be said concerning the abnormally high ampere-hour efficiencies. These varied in the experiments between 109 and 108 per cent, on the usual assumption that gold is three-valent in a gold chloride solution. But the so-called gold chloride solutions do not contain the gold as $AuCl_3$, but in form[11] of $HAuCl_4$. This compound, as well as its alkali salts, are complex salts, which first decompose into H and $AuCl_4$.

$AuCl_4$ easily gives off Cl, and the resulting $AuCl_3$ is then decomposed into its constituents.

[11] Hittorf, Poggendorff's Ann., 1859, Vol. 106, page 522.

Wohlwill[12] has further shown that from a gold anode in presence of chlorine gold goes anodically into the solution only if the conditions are fulfilled (HCl, NaCl) that the complex compound $HAuCl_4$ or $NaAuCl_4$ (that is the anion $AuCl_4$) can be formed. The gold is, therefore, deposited on the cathode by a secondary reaction in any case.

By the use of a voltameter Wohlwill has found in his investigations that more gold is always deposited on the cathode than corresponds to the equivalent Au/3. Without any doubt gold dissolves anodically also as AuCl, just as from a copper anode cuprous ions are formed besides cupric ions.

Wohlwill has proven that this AuCl is again decomposed into gold and $AuCl_3$: $3AuCl = AuCl_3 + 2Au$ (just as Cu_2SO_4 is decomposed into $CuSO_4$ and Cu).

This decomposition of AuCl is, however, not complete in the neighborhood of the anode, and a part of the monovalent gold ions reaches the cathode and is there discharged. The number of monovalent gold ions passing from the anode into solution depends greatly on the current density, as is also shown by our experiments. With the highest current densities the formation of monovalent gold ions is a minimum, at the lowest current density it is a maximum. Therefore, the excess of gold deposited above the theoretical value on the assumed three-valent basis is a maximum in the latter case.

The phenomena of solution at the anode have been discussed in detail by Wohlwill, and there is no further need of referring here to this subject. From Wohlwill's experiments it follows that the "anodic solution equivalent" of gold is higher than its "cathodic deposition equivalent," that is, the loss of weight of the anode exceeds considerably the increase of weight of the cathode.

The question now comes up whether this method of transporting gold from an anodic carbon plate to a gold foil cathode, with a gold chloride solution for refining purposes, can be carried out in practice. This question must certainly be answered in the affirmative. With a potassium cyanide solution, however, refining would not be possible, but it is practical with a chloride solution containing free acid. The cyanide solutions often contain large quantities of silver besides gold, also some copper, lead and iron. Of course, the same metals are then also found in the raw gold, obtained by electrolytic precipitation from the cyanide solutions. In refining this raw gold with a chloride solution only gold is deposited at low voltages, while silver is precipitated as silver chloride; copper and iron pass into solution, but are not deposited, while lead is oxidized and passes into the anode slime. With such a refining process very pure electrolytic gold is, therefore, obtained.

To sum up the results of this investigation we have found:

That for the electrolysis of dilute cyanide solutions the lead peroxide anodes recommended by Andreoli are unsuitable;

That in the electrolysis of such gold solutions the ampere-hour efficiency is exceedingly low and will remain in the average far below 1 per cent;

That carbon electrodes of a certain quality are quite durable in dilute cyanide solutions, and may be used under certain conditions in the place of lead cathodes;

That the use of carbon electrodes becomes necessary if it is intended that the raw gold deposited on the same from the cyanide solution should be directly refined in a succeeding electrolysis, which can be done in a gold chloride solution containing free acid.

In gold chloride solutions, to which sodium chloride is added to improve the conductivity, exceedingly high ampere-hour efficiencies may be obtained with low current densities.

Such a process would make unnecessary the whole treatment of the lead cathodes, which can alloy only with 2 to 12 per cent of gold, and the final product would be fine gold of about 0.988 purity instead of raw gold of 0.800 or 0.900 purity.

Electrochemical Laboratory, Institute of Technology, Darmstadt, Germany.

[12] Zeit. f. Elektrochemie, 1897-8, Vol. IV., page 381.

Standard Determination of Carbon and Sulphur.

As was already briefly mentioned in our last issue the Committee on standard methods for the analysis of iron of the American Foundrymen's Association recommended in a report at the Cleveland meeting the following methods of determining total carbon and sulphur:

Carbon Determination.

The train shall consist of a preheating furnace, containing copper oxide (Option No. 1), followed by caustic potash (1.20 S. Gr.), then calcium chloride, following which shall be the combustion furnace, in which either a porcelain or platinum tube may be used (Option No. 2). The tube shall contain 4 or 5 inches of copper oxide between plugs of platinum gauze, the plug to the rear of the tube to be about the point where the tube extends from the furnace. A roll of silver foil about 2 inches long shall be placed in the tube after the last plug of platinum gauze. The train after the combustion tube shall be anhydrous cupric sulphate, anhydrous cuprous chloride, calcium chloride, and the absorption bulb of potassium hydrate (Sp. Gr. 1.27), with prolong filled with calcium chloride (Option No. 3). A calcium chloride tube attached to the aspirator bottle shall be connected to the prolong.

In this method a single potash bulb shall be used, a second bulb, as sometimes used for a counterpoise being more liable to introduce error than correct any error in weight of the bulb in use, due to change of temperature or moisture in the atmosphere.

The operation shall be as follows: To 1 gram of well-mixed drillings add 100 c. cm. of potassium copper chloride solution and 75 c. cm. of hydrochloric acid (conc.). As soon as dissolved, as shown by the disappearance of all copper, filter on previously washed and ignited asbestos. Wash thoroughly the beaker in which the solution was made with 20 c. cm. of dilute hydrochloric acid (1-1); pour this on the filter and wash the carbon out of the beaker by means of a wash bottle containing dilute hydrochloric acid (1-1), and then wash with warm water until all the acid is washed out of the filter. Dry the carbon at a temperature between 95° and 100° C.

Before using the apparatus a blank shall be run and if the bulb does not gain in weight more than 0.5 mg., put the dried filler into the ignition tube and heat the preheating furnace and the part of the combustion furnace containing the copper oxide. After this is heated start the aspiration of oxygen or air at the rate of three bubbles per second, to show in the potash bulb. Continue slowly heating the combination tube by turning on two burners at a time, and continue the combustion for 30 minutes if air is used; 20 minutes if oxygen is used. (The Shimer crucible is to be heated with a blast lamp for the same length of time.)

When the ignition is finished turn off the gas supply gradually, so as to allow the combustion tube to cool off slowly, and then shut off the oxygen supply and aspirate with air for 10 minutes. Detach the potash bulb and prolong, close the ends with rubber caps and allow it to stand for 5 minutes then weigh. The increase in weight multiplied by 0.27273 equals the percentage of carbon.

The potassium copper chloride shall be made by dissolving 1 pound of the salt in 1 liter of water and filtering through an asbestos filter.

Option No. 1.—While a preheater is greatly to be desired, as only a small percentage of laboratories at present use them, it was decided not to make the use of one essential to this method—subtraction of the weight of the blank to a great extent eliminating any error which might arise from not using a preheater.

Option No. 2.—The Shimer and similar crucibles are largely used as combustion furnaces, and for this reason it was decided to make optional the use of either the tube furnace or one of the standard crucibles. In case the crucible is used it shall be followed by a copper tube 3-16 inch inside diameter and 10 inches long with no ends cooled by water jackets. In the center of the tube shall be placed a disc of platinum gauze and for 3 or 4 inches on the side toward the crucible shall be silver foil, and for the same distance on the other side shall be copper oxide. The ends shall be plugged with glass wool and the tube heated with a fish tail burner before the aspiration of air is started.

Option No. 3.—Barium hydrate is so extensively used as an absorbent for the carbon dioxide that your committee decided to make no recommendation as to whether its use should be made optional, but leave that point to be decided at the general meeting of the Metallurgical Section.

Sulphur Determination.

Dissolve slowly a 3-gram sample of drillings in concentrated nitric acid in a platinum dish covered with an inverted watch glass. After the iron is completely dissolved add 2 grams of potassium nitrate, evaporate to dryness and ignite over an alcohol lamp at red heat. Add 50 c. cm. of a 1 per cent solution of sodium carbonate, boil for a few minutes, filter, using a little paper pulp in the filter if desired, and wash with a hot 1 per cent sodium carbonate solution. Acidify the filtrate with hydrochloric acid, evaporate to dryness, take up with 50 c. cm. of water and 2 c. cm. of concentrated hydrochloric acid, filter, wash, and after diluting the filtrate to about 100 c. cm. precipitate with barium chloride, filter, wash well with hot water, ignite and weigh as barium sulphate, which contains 13.733 per cent of sulphur.

Metallurgical Calculations.

By J. W. Richards, Ph. D.
Professor of Metallurgy in Lehigh University.

HEAT BALANCE SHEET OF BLAST FURNACES.

Problem 54.

(Data partly from paper by the author, Transactions American Institute of Mining Engineers, 1905).

A blast furnace running on Lake Superior ore has the following charges, per 100 of pig iron produced:

Hematite ore = 177.0; composition	H^2O	= 10.0 per cent
	SiO^2	= 10.0 "
	Al^2O^3	= 3.5 "
	Fe^2O^3	= 70.5 "
Limestone: 44.4; composition	SiO^2	= 5.0 "
	MgO	= 4.8 "
	CaO	= 47.6 "
	CO^2	= 42.6 "
Coke: 95.8; composition	SiO^2	= 5.3 "
	CaO	= 5.3 "
	H^2O	= 1.0 "
	C	= 88.0 "
Pig iron produced: 100; composition	Si	= 1.0 "
	C	= 4.0 "
	Fe	= 95.0 "
Gases produced: composition dry:	CO^2	= 13.0 "
	CO	= 22.3 "
	N^2	= 64.7 "

Blast used: Contained 5.00 grains of moisture per cubic foot of dry air, at 24° C. = 75° F.
Charges in pounds: Coke, 10,200; ore, 20,000; stone, 5,000.
Product per day: Pig iron, 358 tons = 801,920 pounds.
Coke used per day: 768,626 pounds.
Temperature of blast: 720° F. = 382° C.
Temperature of waste gases = 538° F. = 281° C.
Displacement of blowing engines = 80,000 [illegible] per min.
Heat in one unit of pig iron = 325 Calories.
Heat in one unit of slag = 525 Calories.
Cooling water, per day, heated 50° C. = 300,0[illegible] gallons.

Required. (1) The volume of gases per 100 kg. of pig iron.

(2) A balance sheet of materials entering and leaving the furnace, per 100 units of pig iron.

(3) The volume and weight of blast per 100 kg. of pig iron.

(4) The efficiency of the blowing plant.

(5) The heat balance sheet of the furnace.

(6) The proportion of the fixed carbon of the fuel burnt at the tuyeres.

(7) The proportion of the whole heat generated at the tuyeres.

(8) The proportion of the iron reduced in the furnace which is reduced by solid carbon from FeO.

(9) The theoretical maximum of temperature at the tuyeres.

(10) The theoretical maximum if all the moisture were removed from the blast.

Solution: (1) To find the volume of the gases:

Carbon in the coke used 95.8×0.88 = 84.3 kg.

Carbon in the limestone $44.4 \times 0.426 \times \frac{3}{11}$ = 5.2 "

Total carbon entering furnace = 89.5 "

Carbon in 100 of pig iron = 4.0 "

Carbon entering gases = 85.5 "

Carbonous and carbonic oxides in gases = 35.3 per cent.

Carbon in 1 m³ of dried gas $= 0.54 \times 0.353$ = 0.19062 kg.

Dry gas per 100 kg. of pig iron $= \frac{85.5}{0.19062}$ = **448.5** m³. (1)

Dry gas per 100 oz. of pig iron = 448.5 ft.³

Dry gas per 100 pounds of pig iron =
448.5×16 = 7176.0 "

Dry gas per 2240 pounds of pig iron =
$7176.0 \div 22.4$ = 160.742 "

Dry gas per minute $= \frac{160.742 \times 358 \text{ (tons)}}{60 \times 24}$ = 39.962 "

(2) BALANCE SHEET OF MATERIALS, PER 100 OF PIG-IRON

Blast 422.2, Fuel 95.8, Flux 44.4, Ore 177.5

Charges		Pig Iron		Slag		Gases	
Fe_2O_3	135.6	Fe	95.0	.....		O	40.7
H_2O	17.8					H_2O	7.8
SiO_2	17.8	Si	1.0	SiO_2	15.7	O	1.1
SiO_2	2.2			SiO_2	2.2		
CaO	21.1			CaO	21.1		
MgO	2.1			MgO	2.1		
CO	19.0					CO_2	19.0
C	84.3	C	4.0			C	80.3
SiO_2	5.3			SiO_2	5.3		
CaO	5.3			CaO	5.3		
H_2O	0.9					H_2O	0.9
O	96.4					O	96.4
N_2	321.3					N_2	321.3
H_2O	4.5					H	0.5
						O	4.0
Total	740.0		100.0		58.0		582.0

The charges are calculated simply from the weights of ore, flux and fuel used, and their percentage composition. The blast is calculated as given in solution of requirement (3).

The 1.0 of silicon in the pig iron requires $1.0 \times (60 \div 28) = 2.1$ parts of SiO^2 to furnish it, leaving 15.7 of unreduced SiO^2 to go into the slag, and 1.1 of oxygen to the gases.

(3) The balance sheet shows that the solid charges furnish to the gases 41.8 of O, 100 of CO and 18.7 of HO. The HO goes as such into the gases, and therefore its oxygen is not present in the sample of dry gas analysed. The oxygen in CO is $19.0 \times 1.32 = 14.0 = 13.8$ kg., which added to the 41.8 gives 55.6 of oxygen going into the gases as CO or CO^2 from the solid charges.

The oxygen in the CO and CO^2 of the gases is to be calculated from the oxygen in unit volume of gas and the total volume of gas produced. The oxygen in 1 cubic meter of gas will be:

$$\text{O in CO} = 0.223 \times \left(0.09 \times \frac{28}{2}\right) \times \frac{16}{28} = 0.16056 \text{ kg.}$$

$$\text{O in } CO^2 = 0.130 \times \left(0.09 \times \frac{44}{2}\right) \times \frac{32}{44} = 0.18720 \text{ "}$$

$$0.34776 \text{ "}$$

A quicker solution is to note that CO^2 represents O^2, its own volume of oxygen, and CO represents O, or half its volume of oxygen, and since 1 cubic meter of oxygen weighs $0.09 \times (32 \div 2) = 1.44$ kilograms, the weight of oxygen required is

$$\left(\frac{0.223}{2} + 0.13\right) \times 1.44 = 0.34776 \text{ kg.}$$

The total oxygen in 448.5 m³ of gases is
therefore 448.5×0.34776 = 156 kg.
subtracting that furnished by the solid charges = 55.0 "
there remains oxygen furnished by blast = 100.4 "
If the blast were perfectly dry air, its nitrogen would be $100.4 \times (10 \div 3)$ = 334.7 "
and the weight of dry blast = 435.1 "
and its volume at 0° C. $435.1 \div 1.293$ = 336.5 m³

The blast, however, is not dry, and the 100.4 kilos. of oxygen includes that brought in by the moisture. The moisture weighs 5.66 grains per cubic foot of measured dried air; it is, therefore, simplest to calculate the oxygen entering the furnace per unit volume of air proper entering. Since 5.66 grains = 5.66 ÷ 437.5 = 0.01294 ounces av., the calculation can be made in ounces per cubic foot or kilograms per cubic meter in identical figures as follows:

Oxygen in 0.01294 parts of water $= 0.01294 \times \frac{8}{9}$ = 0.0115

Oxygen in 1 volume of air proper, at 24° C.

$$= 1.293 \times \frac{3}{13} \times \frac{273}{273 + 24} = 0.2743$$

Sum = 0.2858

The blast received, per 100.4 kg. of oxygen thus received, is therefore, measured dry at 24° C:

$$\frac{100.4}{0.2858} = 351.8 \text{ cubic meters.}$$

and in cubic feet, per 100.4 pounds of pig iron:

$$\frac{100.4 \times 16}{0.2858} = 5628.8 \text{ cubic feet}$$

The volume of the moist air containing this will be the volume of assumed dried air plus the volume of the water vapor. The latter is, per unit volume of dried air:

$$0.01294 \div \left(0.09 \times \frac{18}{2} \times \frac{273 + 24}{273}\right) = 0.01738$$

and the volume of moist air received, at 24°, is, therefore

$351.8 \times 1.01738 = 357.9$ cubic meters
Or $5628.8 \times 1.01738 = 5726.6$ cubic feet

The weights of water, oxygen and nitrogen received per 100 of pig iron are (as already given on the balance sheet):

HO $0.01294 \times 351.8 = 4.52$ kg
O $0.2743 \times 351.8 = 96.4$ "
N^2 $0.9143 \times 351.8 = 321.3$ "

And the volumes of these, considered theoretically at 0° C. and standard pressure, per 100 kg. of pig iron made

H^2O vapor	4.52 ÷ 0.81	=	5.6 cubic meters
Air	417.7 ÷ 1.293	=	322.8 " "
	Sum	=	328.4 " "

There are several other modifications of this solution which will suggest themselves to anyone who studies out the question, but while half a dozen ways may be equally correct, that one is logically to be preferred which is most easily understood and is the shortest. One solution, however, based on volume relations, is well to know. Water vapor, H^2O, represents half its volume of contained oxygen, while air has 0.208 oxygen The cubic foot of dried air at 24° C. was accompanied by

$$0.01294 \div 0.81 \times \frac{273 + 24}{273} = 0.0174 \text{ cubic foot}$$

of moisture, which was removed in making the test. The oxygen per cubic foot of dry blast was, therefore, at 24° C.:

O as H^2O	0.0174 ÷ 2	= 0.0087 cubic foot.
O^2 as air	1.0000 × 0.208	= 0.2080 " "
		0.2167 " "

Weight of this oxygen:

$$1.44 \times \frac{273}{273 + 24} \times 0.2167 = 0.2868$$

And volume of dry blast, at 24°, per 100 of pig iron:

$$\frac{100.4}{0.2868} = 350.1 \text{ cubic meters.}$$

The difference of about 0.4 per cent between this and the previously-calculated result is due to the figures not being carried out to a sufficient number of decimal places.

(4) *The efficiency of the blowing plant* is found by calculating the volume of air and moisture received by the furnace per minute, calculated to 24° C. = 75° F., and comparing this with the piston displacement of 40,000 cubic feet per minute.

Volume of moist air received, at 24° C., per 100 lbs. of pig iron made	= 5726.6 ft.³
Volume per day = 5726.6 × 8019.2	= 45,922,750 ft.³
Volume per minute = 45,922,750 ÷ 1440 =	31,891 "
Efficiency of blowing plant $\frac{31,890}{40,000}$	= 79.7 per cent (4)

(5) *The heat balance sheet* is based for most of its data upon the balance sheet of materials, the calculations already made and the additional data given in the statement.

The balance sheet shows 80.3 kilos. of carbon oxidized in the furnace. Of this, the amount oxidized to CO and remaining as such in the gases is obtained from the amount of CO in the gases, as follows:

C in CO = 448.5 × 0.223 × 0.54 = 54.0 kg.
C oxidized to CO^2 is therefore 80.3 — 54 = 26.3 kg.

The heat in hot blast is calculated from its volume assumed to be at 0° C., and already calculated, viz.: 322.8 cubic meters of air proper and 5.6 cubic meters of water vapor, with mean specific heats per cubic meter between 0° and 382° C. of 0.313 and 0.40 respectively.

The heat of formation of the pig iron may be taken from the amount of carbon in the iron, as 705 Calories per kilogram of carbon, and that of silicon omitted from consideration.

The heat of formation of the slag, since it contains 29.5 of silica and alumina to 28.5 of lime and magnesia, may be taken as 150 Calories per unit of silica and alumina.

The total heat received and generated, and to be accounted for in the furnace, is therefore, per 100 kg. of pig iron:

Carbon to CO	54.0 × 2430	= 131,220 Calories
Carbon to CO^2	26.3 × 8100	= 213,030 "
In H^2O vapor In air proper	5.6 × 0.40 × 382 322.8 × 0.313 × 382	= 30,385 "
Solution of C in iron	4 × 705	= 2,820 "
Formation of slag	29.5 × 150	= 4,425 "
	Total	= 390,880

The items of heat distribution are: 325 Calories in each kilogram of pig iron, 525 Calories in each kilogram of [illegible] in the waste gases at 281°, heat in cooling water, heat in radiation and conduction (by difference), evaporation of the moisture in charges, expulsion of CO^2 from carbon [illegible] composition of moisture of blast, reduction of silicon and iron

Reduction of Fe	95 kg. × 1,746	=	165,870 Calories
Reduction of Si	1 " × 7,000		7,000 "
Expulsion of CO^2 from $CaCO^3$	16.7 " × 1,026	=	18,000 "
Expulsion of CO^2 from $MgCO^3$	2.3 " × 666		
Evaporation of H^2O	18.7 " × 666.5	=	11,312 "
Heat in waste gases: N and CO 400 m³ × 0.3106 CO^2 58.3 m³ × 0.446 H^2O 23.1 m³ × 0.382	× 281	=	43,830 "
Decomposition of moisture of blast: H^2O 4.5 kg. × (29,040 ÷ 9)		=	14,511 "
Heat in slag 58 kg. × 525		=	30,450 "
Heat in pig iron 100 × 325		=	32,500 "
Heat in cooling water (300,000 gallons per diem) 300,000 × 8.3 (lbs.) × 50 ÷ 8019.2		=	15,525 "
Loss by radiation and conduction (difference)		=	51,180 "
	Total	=	390,880 " (5)

(6) *The proportion of the fixed carbon burned at the tuyeres* is obtained by calculating the weight of carbon which the oxygen entering at the tuyeres could oxidize to CO, as follows:

Oxygen entering at the tuyeres		= 100.4 kg.
Carbon burned to CO	$100.4 \times \frac{12}{16}$	= 75.3 "
Fixed carbon charged		= 84.3 "
Proportion burned at tuyeres		= 89.3 per cent

A more just proportion is that between the carbon burned at the tuyeres and the total fixed carbon oxidized, because the fixed carbon which carbonizes the iron cannot be oxidized under any circumstances. This proportion in this furnace is

$$\frac{75.3}{84.3 - 4.0} = 93.8 \text{ per cent.} \quad (6)$$

Indicating a very fair approximation to Grüner's ideal working. If we make the further allowance, that the silicon in the pig iron is necessarily reduced by solid carbon, and that therefore the solid carbon needed to reduce the 1 kilogram of silicon ($1 \times (24 \div 28)$) is in no case available for combustion at the tuyeres, we have the approach to Grüner's ideal working measured by the ratio

$$\frac{75.3}{80.3 - 0.9} = 94.8 \text{ per cent.}$$

in spite of which, however, the furnace is not doing very good work.

(7) The proportion of the heat requirement generated or available at or about the tuyeres is determined as follows:

Combustion of C to CO	75.3 × 2430	= 182,979 Calories.
Heat in hot blast		30,385 "
Formation of pig iron		= 2,820 "
Formation of slag		4,425 "
	Total	= 220,609 "

Against which must be charged heat required to decompose moisture of blast $= 14,511$ "

Leaving net heat available $= 215,098$ "
which is $215,098 \div 390,880 = 55$ per cent.

of the total heat generated and available in the furnace.

Another way in which it is sometimes put, is that the oxidation of carbon to CO or CO^2 furnishes a certain amount of heat to the furnace (346,250 Calories in this case), of which a certain amount is generated by combustion at the tuyeres (182,970 Calories), making the ratio thus considered, in this case, 53 per cent.—which is almost the same figures as above, but not so significant, since it is illogical to consider the heat brought in by the hot blast as not being available heat for doing work in the tuyere region.

(8) The proportion of iron reduced by solid carbon is found by finding how much carbon is used up in that reduction.

Fixed carbon charged $= 84.3$ kg.
Fixed carbon carbonizing the pig iron $= 4.0$ "

Fixed carbon oxidized in the furnace $= 80.3$ "
Fixed carbon oxidized by the blast $= 75.3$ "

Fixed carbon oxidized above the tuyeres $= 5.0$ "
Carbon needed to reduce 1 kg. silicon $= 0.9$

Carbon used for reducing FeO $= 4.1$ "

Amount of Fe. thus reduced $= 4.1 \times \frac{56}{12} = 19.1$ "

Proportion of the total Fe. thus reduced $= \frac{19.1}{95} =$ **20** per cent. (8)

(9) The maximum temperature of the gases in the region of the tuyeres is that temperature to which the heat there available will raise the products of combustion. This question is best resolved by simply considering the combustion of 1 kilogram of carbon, evolving 2430 calories, while the heat in the hot carbon itself just before it is burnt, and that in the hot blast required, will also exist as sensible heat in the products,—the whole diminished by the heat necessary to decompose the water vapor blown in.

Since, per 75.3 kg. of carbon burned at the tuyeres there enter 5.6 m^3 of water vapor and 322.8 m^3 of air proper, measured at 0°, the volume of blast per kilogram of carbon oxidized at the tuyeres is

H^2O $5.6 \div 75.3 = 0.0738$ $m^3 = 0.0598$ kg.
Air $322.8 \div 75.3 = 4.2869$ m^3

The products of the combustion are, per kg. of carbon burned:

CO $22.22 \div 12 = 1.8519$ cubic meters
N^2 $321.3 \div 1.26 \div 75.3 = 3.3805$ " "
H^2 equal to H^2O decomposed $= 0.0738$ " "

Total $= 5.3122$ " "

The heat available to raise their temperature is:

Heat of combustion of 1 kg. carbon $= 2430$ Calories
Heat in hot blast $= 30.385 \times 75.3 = 523$ "
Heat in hot carbon at t° (or very nearly) $= 0.5t - 120$ "
Less heat absorbed in decomposing steam
$14.511 \times 75.3 = 193$ "
Net heat available in gaseous products $= 2640 + 0.5t$ Cal.

Calorific capacity of gaseous products
$= 5.3122\ (0.303t + 0.000027t^2)$

Therefore $5.3122\ (0.303t + 0.000027t^2) = 2640 + 0.5t$
Whence $t = 1910$ (9)

This represents the absolute maximum of temperature which the gaseous products formed at the tuyeres can possess.

(10) If all the moisture were removed from the blast, the heat available would be:

By combustion of 1 kg. carbon $= 2430$ Calories.
Heat in 4.4685 m^3 of dry air at 382° C.
$= 4.4685 \times 0.313 \times 382 = 574$ "
Heat in 1 kg. of carbon at t° $= 0.5t - 120$ "
Net heat available in gaseous products $= 0.5t + 2884$ Cal.

Calorific capacity of gaseous products
$= 5.3976\ (0.303t + 0.000027t^2)$

Therefore $5.3976\ (0.303t + 0.000027t^2) = 2884 + 0.5t$
Whence $t = 2018°$ (10)

It is to be noted that this is 108° C. = 194 F. higher than with moist blast; and while the slag and iron in passing through this zone of high temperature will not reach this maximum temperature, yet they would be heated approximately 100° C. higher when using dry blast, if all other conditions were kept constant.

N. B.: In working this problem, the metric measurements and English measures have been purposely used interchangeably, in order that readers may understand better that if weights are taken in pounds and the heat unit is the pound Calorie (1° C.), the same numbers represent a solution in either system. When volumes are concerned, cubic feet and ounces, or ounce calories, have practically the same numerical expression as cubic meters and kilograms or kilogram calories.

Gold and Silver Refining.

The electrolytic refining of gold has been carried out on a large scale for a number of years in Germany, and was first applied in this country in the United States Mint in Philadelphia by their smelter and refiner, Dr. D. K. Tuttle. An authoritative illustrated description of the equipment of the electrolytic refining room of the Philadelphia Mint, from the pen of Dr. Tuttle, may be found on page 157 of our Vol. I., while a more detailed discussion of the electrolytic process of gold refining was given by the original inventor of the process, Dr. E. Wohlwill, of Germany, on pages 221 and 261 of our Vol. II.

The annual report of 1905 of the director of the United States Mint contains some interesting notes by Dr. Tuttle on the progress made at the Philadelphia Mint in electrolytic refining, which are herewith reproduced verbatim:

"Operations in the nitric acid refining plant were discontinued in February, 1905. The electrolytic process for refining gold continues to give satisfaction, and now that the method for parting and refining silver is perfected, all the refinery operations are conducted by electrolysis.

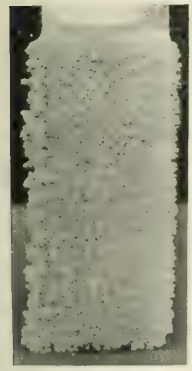
"At the date of the last annual report electrolytic refining was confined to high-grade gold containing a small percentage of silver. Since that time the hope has been realized which was expressed in the report for 1902 (p. 123), where it was said: 'The ideal refining plant for a mint would be one in which electrolytic separations are the leading features. The bullion to be parted and refined will be divided into two classes, by selection and by blending in making up the materials for anodes. The one class will have silver as the predominant metal, but carrying as much gold as will permit its treatment by electrolysis in a silver bath. The product from this operation will be fine silver, and the residues will consist of gold, platinum and other impurities. The other class of bullion is to be largely gold, as at present, selected for electrolytic treatment in a gold chloride solution. The residues from each process would pass to the other for final treatment. Work is now being done looking to a realization of this scheme.'

"By persistent effort this has now been accomplished, and the energy obtained from a few bushels of coal in the form of an electric current is made to do the work of dray loads of expensive acids.

"Doré bars of silver containing small quantities of gold are successfully refined in industrial establishments by the Moebius or similar processes, but, since in mint practice silver has to be added to the gold and used as parting material, an economical process must require the minimum percentage of silver in the anodes. As will be seen from the following description, our anodes in the silver cells consist of 30 per cent gold and 70 per cent of silver, copper, lead, etc. It is believed that the successful treatment of such bullion on a large scale by electrolysis is an innovation in practice.

"In the electrolytic plant installed in the Philadelphia Mint two distinct processes are employed, the choice of one or the other depending on the character of the bullion to be treated. If it be gold, high in fineness, but containing from 40 to 60 parts of impurity per thousand, such as silver, platinum, copper, lead, etc., it is refined by what is known as the 'Wohlwill process."[1] * * * *

"When silver is the chief element, with lesser percentages of gold, copper, lead, etc., present to be parted, a different electrolyte and a modified system of working are employed. The electrolyte used is a 3 per cent solution of silver nitrate in water, to which is added 1½ per cent free nitric acid. The tanks are of earthenware, 40 inches by 20 inches and 11 inches deep. In each of these are suspended from conducting rods 42 anodes and 40 cathodes. The anodes are composed of 300 parts of gold in 1,000, the remaining 700 parts consisting of silver, copper and other impurities as parting material. They are cast into bars 7¼ inches long by 2½ inches wide and three-eighths of an inch thick. The cathodes are strips of fine silver of same length and width rolled to 0.016-inch thick.



ELECTROLYTIC SILVER CATHODE.

"Eight cells so equipped are connected up in series, and a current with a density of 0.05 amp. per square inch passes through the system. The silver and other solubie metals are extracted from the anode by the combined action of the current and electrolyte, while the gold remains as a chocolate-brown substance, sufficiently coherent to retain the original form of the anode. Meanwhile pure silver is deposited in a crystalline but coherent form on the cathode. Heretofore a coherent deposit has not been obtained from a silver nitrate solution, the product in the Moebius and other processes in commercial use being non-adherent crystalline granules, which fall from the cathode to the bottom of the cell.

"The deposit in a coherent form is due to a happy observation of the melter and refiner, in which it was discovered that the addition of a very small amount of a colloid, such as gelatine, to the electrolyte changed completely the nature of the deposit, so that the "vertical system" of anodes and cathodes became for the first time possible.[2] The cathodes are washed with water, melted without fluxes and cast into bars.

"The anodes retained persistently a small amount of silver, even if subjected to the current after oxygen is freely evolved from their surfaces. A fact new to the scientific world should here be noted, namely, that if the action of the current be prolonged on the anodes after most of the silver has been dissolved, the nascent oxygen evolved will attack the [illegible] and produce a small but definite quantity of gold peroxide, soluble in concentrated nitric and sulphuric acid. It is deposited from these solutions as dioxide, or, if nitric acid is used, in finely divided form. The [illegible] to its occurrence, should be known to the operator. It is probably [illegible] by simply heating the oxidized anode to 250° and gold passes into solution in acids.

"The silver remaining in the anode is removed by a treatment in hot nitric acid, the resulting solution being used to replenish the electrolyte. The gold is then thoroughly washed with water and melted.

"If platinum be present it will remain with the gold and we usually pass this through the gold refining cells, which is a very inexpensive operation, and gives a much purer product while affording the means for recovering other values. The losses in these operations need be very slight if care and cleanliness be used. All accidental sloppages of solutions are mopped with cotton cloths, which are burned and the ashes preserved. The necessary losses should be less than 1 ounce in 10,000. How much less we hope to show by precise figures after more extended experience."

[1] The description which now follows is essentially the same as in Dr Tuttle's article, our Vol. I., page 157.

[2] See also the paper by F. D. Easterbrooks, on parting of bullion, in our Vol. III., page 373. According to a private communication from Dr. Tuttle, his silver cathodes are often 1000 fine. One of the same is illustrated in the above diagram. Concerning the anodic information of gold peroxide, mentioned in the next paragraph, Dr. Tuttle remarks that this is, of course, analagous to the formation of lead and silver peroxides under similar conditions. Editor.

The Lines of Current in Storage Batteries—An Experimental Study.

By M. U. Schoop.

(*Concluded from page 271.*)

Small plates of spongy lead are very suitable as testing electrodes for the separate determination of the capacities of the positives and negatives of a lead accumulator. In this case the little testing plate is generally placed in the electrolyte above the upper rim of the plate, proper connections being made to a galvanometer. The chemical reaction consists in the discharge of the little plate of spongy lead by means of the current passing through the instrument, whereby the lead is slowly changed into sulphate.

What happens if the same little testing plate is placed between the main electrodes right into the field of the lines of current, no connections being made to a galvanometer? Like any intermediate electrode the spongy lead plate will act as a bipolar electrode with anodic polarization on one side and cathodic polarization on the other side. But since we have here something like a short-circuited couple the products of polarization will tend to destroy each other immediately, and the final result of the electrochemical reaction on the plate is a slow formation of sulphate or, what is the same, a slow discharge. It may be easily seen that the speed of this reaction depends on the thickness of the plate, that is, on the voltage drop within the plate.

In order to investigate a storage battery electrode showing a non-homogeneous discharge, it would therefore suffice to place two identical small spongy lead plates into the electrolyte on both sides of the electrode in exactly the same distance from its surfaces, and to observe the reading of a sensible galvanometer connected to the two testing plates. If the galvanometer needle shows a deflection, one of the two little testing plates is met by a greater number of lines of current than the other one.

We may, for instance, combine two parallel small spongy lead plates to a little instrument, as shown in Fig. 7, which explains itself. It is, of course, highly important that the distances of the two little testing plates from the main electrode should be exactly the same. The best arrangement is to let the two little testing plates rest directly on the support as shown in Fig. 8, a direct contact of the little testing plates with the main electrode to be tested being avoided by small perforated hard rubber pieces pasted on the rim.

Such a measurement will establish whether the current density is the same on both sides of the electrode. As may be stated at once, this is never the case in those traction cells with n positive and n + 1 negative plates, in which the negative end plates do not differ from the other negative plates with respect to thickness and capacity.

On that side of the positive end plates, which looks towards the negative end plates, the current density is considerably greater than on the opposite side, since it would be wrong to suppose that the terminal negatives are active on their inner side only.

FIG. 7. ANALYZER.

With large stationary types of storage batteries, this fact has been taken into account long ago, at least in good products, although perhaps primarily for the sake of saving lead. With transportable cells, however, terminal negatives with half capacity are not used in practice to my knowledge. Since the electrodes of portable cells, especially of traction cells, must be very thin, anyhow, for the sake of a high capacity, per unit of weight, it is quite difficult to select an arrangement by which a uniform discharge of all the plates can be attained. It might, perhaps, be possible to design the connections of the end negatives with the cross-bars, so as to increase their resistance.

As may be expected, and as is often found in practice, the end positives of discarded traction cells show far more signs of disintegration than the other positives, and often the end positives are more or less curved toward the terminal negatives in the manner indicated in Fig. 9.

In traction cells the current is introduced generally in the middle of the cross-bar which connects the different plates. This results evidently in a drop of voltage whereby the end plates carry less current than the plates in the center of the cell. The double capacity of the negative terminal plates, however, more than counterbalance this effect of the voltage drop.

With the little instrument described above it is easy to prove in an easy manner that this disturbing effect of the terminal negatives relates not only to the positives next to them but to all other plates. Of course, the readings of the millivoltmeter always get smaller the nearer we come to the center of the cell, and they disappear altogether at the center where the current is introduced into or taken off from the cell.

FIG. 8.—USE OF ANALYZER.

Concentration differences annihilate themselves, first, because the drop of concentration in the pores of the plates causes diffusion from higher to lower concentration, and, secondly, because so-called concentration currents are produced which are even more effective in bringing about uniformity of concentration. Dolezalek has shown this mathematically while Schoop has proven it by experiment.[1]

At this place we may mention another kind of local currents which always occur when the charge or discharge of a plate is not homogeneous (*i. e.*, in all cases of practice). The part of the plate which is more strongly discharged (the bottom part) contains more dilute acid within the pores while the less discharged part of the plate (the upper parts) have a stronger concentration of acid in their pores. By diffusion and concentration currents these differences of concentration tend to annihilate themselves until equilibrium is reached. It is easy to prove this experimentally by the little instrument with the two little spongy lead plates described above. But the needle of the galvanometer gives larger deflections if the little testing plates are not placed behind each other, but at a distance of about 8 mm. side by side. What is now measured is simply the voltage drop corresponding to the distance of the little plates in the electrolyte. If we now turn the instrument slowly around its longitudinal axle until the galvanometer shows zero current, we have reached an equipotential curve, on which the lines of current must be perpendicular.

The current density is, of course, a maximum in next proximity to the surface of the main electrode and decreases gradually with increasing distance from the main electrode. The figure of the lines of current from a spongy lead electrode which is not homogeneously discharged at the top and bottom, looks about like that shown in Fig. 10. For the investigation of these currents it is easier to place the electrode which is being tested longitudinally on the side. If it is left in its normal perpendicular position the analyzer is to be modified to suit the purpose.

FIG. 9.—DISTORTION OF BATTERY PLATES.

The presence of such very strong parasitic currents can be proven, if, for instance, a charged spongy lead plate is exposed to about one-third of its surface for a few minutes to the oxidizing action of the air. But the same phenomenon may also be observed whenever there is any lack of homogeneity at different places of an electrode in a cell.

This simple method gives us reliable means of testing whether the charge or discharge of an electrode is homogeneous. I have used the same experimental method for proving that any electrolytic shunt results in a lack of homogeneity of the distribution of current density over the surface of an electrode, and that, for instance, in a storage battery, on account of the space left between the lower ends of the plates and the bottom of the cell, the current density is higher in the lower parts of the plates than in the upper ones. In storage battery factories this point is of importance for the formation of plates.

As has already been mentioned the straying of lines of current is only one of several factors which produce a non-uniform distribution of the current density over the electrodes. The effect of the straying of the lines of current may be more or less masked or counterbalanced by other factors, like the construction of the lead grid, or the mechanical details of the connections between the plates and the external circuit, or the porosity of the active masses.

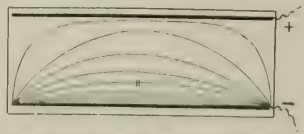


FIG. 10.—DIAGRAM OF EQUIPOTENTIAL CURVES.

In general, an electrode will tend to correct itself any lack of uniformity of current density over its surface. For instance, during discharge, those parts of the electrode which have a higher current density will get a greater resistance, so that the current will gradually pass over to those parts which were less discharged before. On the other hand, during charge, those places which have first been charged will have more concentrated acid in the pores, and this will result in a counter e m f which will force the lines of current out of their position.

It would, therefore, be wrong to think that the distribution of the lines of current over an electrode or at a certain place of the electrode surface or of the cross section of the electro

[1] 'Electrochemical Industry,' 1903, Vol. II., pages 272 and 310.

lyte remains always the same. On the contrary, there will be continuous and gradual changes in the current density at any place, and I may say right here that they are far more evident during a discharge than during a charge.

In order to prove the fluctuations of the current distribution in a working storage battery and to get exact figures, we may again use the little instrument described above, which is placed at equal time intervals between the plates (at the top, in the middle or at the bottom). With large cells which have a considerable distance between plates this instrument may be used without modification.

With traction cells with thin plates, however, the distance between the plates is mostly 3 or 4 mm. only, and is too small to employ the little instrument directly. It is, therefore, neces-

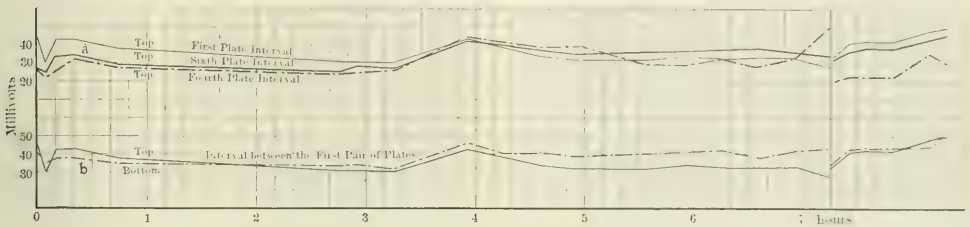


FIG. 11.—CURVES SHOWING CHANGES IN DIFFERENT INTERVALS BETWEEN PLATES.

sary to place the plates at a greater distance from each other, using glass tubes or hard rubber forks for keeping the distance constant. The glass tube of the analyzer is provided with marks which show the depth to which the little testing plates are immersed into the electrolyte.

I may say that the polarization produced at the little testing plates is very small, and amounts to only 1 or 2 millivolts. To measure it, one has to open the circuit after the measurement or, what is preferable, to place the analyzer into a nearby vessel containing acid of the same concentration, and to observe simultaneously the millivoltmeter.

The curves shown in Fig. 11 relate to a traction cell with five positive and six negative plates, 300 x 100 x 3 mm., with a distance of 15 mm. of the plates from the bottom. This cell as tested, differs from a normal traction cell only in so far as the plates are placed at a distance of 14 mm. instead of 3 mm., the uniform distance between the plates being produced by celluloid combs instead of corrugated, perforated hard rubber insulators. This cell had; therefore, in the whole ten intervals between the plates; each interval was to be tested at three different places, at the top, in the center and at the bottom, with respect to current density. This meant thirty measurements. It is preferable to have two observers: one for the millivoltmeter and the other for the analyzer. In order to get reliable results which agree with each other, the observers should have some experience in working together.

The results may, of course, be plotted graphically in different ways.

The Fig. 11a (top of the diagram) shows, for instance, the changes of current density during an 8 hour discharge in three different plate-intervals, all the measurements being made at the top, that is, slightly below the surface of the electrolyte. One sees at a glance that the distribution of the current is subjected to very great changes, the curve of these changes having a decided wave form.

Observations of the fourth plate intervals show this very clearly. The current density is here a minimum at the beginning, but the curve rises rapidly after 2½ hours and goes down again after another hour and a half, and only near the end of discharge it goes up again rapidly.

Similar, but in the opposite direction, are the changes of the first plate-interval.

After 7¼ hours the discharge was interrupted for the night, and was continued the next morning. The [illegible] concentration due to diffusion during the night [illegible] is marked. The plate-interval which had the lowest current density the evening before has now the highest. The [illegible] plate-intervals have just reversed their places.

This series of tests also shows clearly that the end [illegible] have a greater current density than the other positives [illegible] the curve of the first plate interval).

The curves at the top of the diagram, Fig. 11a, relate to different plate-intervals, all being tested at the top. In Fig. 11b (the bottom of the diagram) the differences of current density in the same plate-interval at the top and at the bottom are plotted graphically. It will be seen here the difference of the readings gets greater and greater toward the end of discharge (15 millivolts). But in this case the night's rest also renders the concentrations more uniform at the top and the bottom, so that the next morning the two plates continue to discharge with about the same current densities at the top and the bottom.

IV.

The described current fluctuations of a pair of plates occur always. This may be proven by still another arrangement. Between the electrodes to be tested are placed in exactly symmetrical distance from the same two little testing plates connected to a millivoltmeter. The current is closed, and the position of the testing plates is so adjusted that the instrument shows zero current at the beginning, whereby it is of no account what the relative distance is between the little testing plates.

The arrangement is somewhat similar to a Wheatstone bridge, since the bridge will carry no current as long as there is no change in the branches of the circuit during the experiment.

In a great number of experiments under the most different conditions (new and old electrodes, large and small distance between the plates, with or without electrolytic shunt), it could be established that always and under all circumstances the conditions of the current will change, and that these changes are in general very much more evident with sponge lead plates than with lead peroxide plates.

This may be due to the well known fact that sponge lead is subjected to changes of structure, and that the contact with the support has a tendency to get poorer and poorer with each charge or discharge, while with the positive plates the PbO$_2$ layer which grows into the lead guarantees an excellent contact. Moreover, according to the investigation of Streintz PbO$_2$ has a conductivity about half that of mercury, and is, therefore, to be considered as an excellent conductor.

If we want to get information concerning the behavior of a single plate we may use a method similar to that in determining the capacity of a single positive or negative plate. This method consists essentially in using as opposite electrode a plate of four or five times greater capacity, so that the changes of current density occurring on the opposite electrode are neg-

ligible compared with those on the other plate of smaller capacity.

As an example I give the tests of a negative lead electrode, the formation of which had been started according to Planté. It was alternately charged and discharged against a normal positive electrode in ordinary acid. The opposite side of the negative Planté plate was covered with a layer of insulating

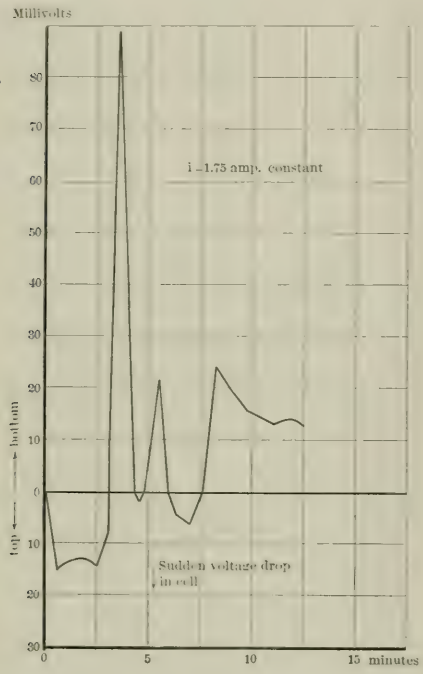


FIG. 12.—DIFFERENCES BETWEEN THE BEHAVIOR OF PLATE AT TOP AND BOTTOM.

paint and the plates were otherwise so arranged as shown in Figs. 1 and 2 (page 269, July issue).

It is hardly necessary to point out that the picture of the distribution of the lines of current is immediately changed if anything whatever is changed in the conditions of the experiment—whether it be the connections to the plates through which the current is conducted to the battery, or the distance between the plates or the space available for the electrolytic shunt, etc. A change would also be found immediately when the Planté spongy-lead plate would be replaced by a positively-formed lead plate.

As Fig. 12 shows, the upper parts of the plate where the current enters the plates, carried a higher current density in the beginning. After 4 minutes, however, a very sharp maximum in the opposite sense appeared, which was almost coincident with a sudden drop of voltage of the cell.

On account of the relatively high current strength of 1.75 amps. the changes in the distribution of the current over the Planté plate were very marked.

A control of the measurements of the different current densities at the top and the bottom of the plate may also be obtained by simply observing the plate, especially the changes of color of the active masses and the beginning of gasing.

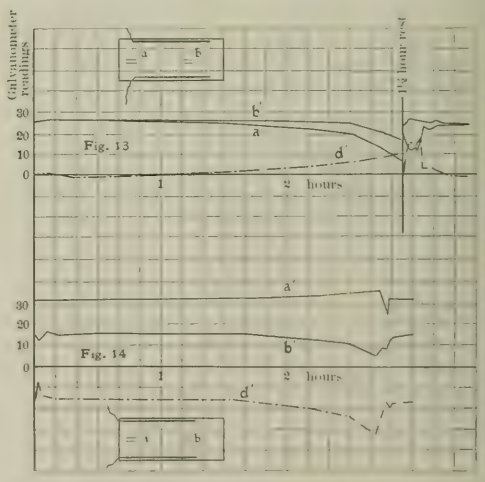
As may be seen from what has been said, the "current-difference curve" is the resultant of two component currents, of which the one current is the real difference current, while the other is to be considered as a local current between the different places of the non-uniformly discharged Planté plate, which is superposed in positive or negative sense to the other current.

In order to find the value of the latter and to separate it from the current difference proper, we could, for instance, open the circuit after each observation and could make a second measurement immediately after the first. For the elimination of the parasitic currents, however, it appeared to me more suitable to replace the bridge method by an arrangement in which the single little plates are replaced each by pairs of little plates. In this case instead of a one-measurement two must be made immediately after one another. What is measured is again the voltage drop in the electrolyte, which is proportional to the current density at that place.

An experiment of this kind with normal and very carefully prepared storage-battery plates for traction cells will finally be described.

As will be seen from the small sketches in Figs. 13 and 14, showing the arrangement of the experiments, the second experiment (Fig. 14) differs from the first experiment (Fig. 13) only in the fact that in Fig. 13 the arrangement is absolutely symmetrical, while in Fig. 14 an electrolytic shunt is provided only on one side (*b*). In both cases the position of the two analyzers is exactly the same, so that any difference in the readings of the millivoltmeter must be due entirely to the differences in arrangement of the electrolytic shunt.

In both figures the curve a' corresponds to the readings made with the analyzer at *a*, while curve b' corresponds to the readings with the analyzer at *b*, while the dotted line d' represents the difference between the current densities ($b' - a'$).



FIGS. 13 AND 14.—COMPARATIVE EXPERIMENTS SHOWING THE EFFECT OF AN ELECTROLYTIC SHUNT.

To avoid any misunderstanding, I may emphasize that these curves are not intended to give any information on the current densities on the electrode surfaces, but that they demonstrate the conditions in the electrolyte at the places *a* and *b*. They show how the distribution of the current in the electrolyte is subject to changes as soon as the smallest change is made in the arrangement of the experiments. The following figures, which give the results of both tests, will show the method of the measurements:

First Test.

Readings of Millivoltmeter.

Voltage at Terminals.	Time. Hrs.	Min.	d'	a'	b'	i amps.
1.69	0	0	25½	25½		8.0 constant
1.67		2	½	26	26½	
...		7	0	26¾	26¾	
1.66		13	0	26½	26½	
1.41		21	½	26¼	26	
1.45		32	¾	26 1/5	26	
1.4		55	0	25½	25½	
...		1.35	1½	24	25½	
1.34		2.07	3¼	21¾	25	
1.32		2.16	4	21	25	
1.28		2.30	5½	19	24½	
1.3		2.46	8	11½	19½	
	1½ Hours Rest.					
...		0	0	23	23	
1.4		1	7½	16	23½	
1.2		2	13	13	26	
1.0		5	14½	12	26½	
Polarization rev'd		8	16½	9	25½	
1.2		8	2	23	25	
1.15		12	3	21½	24½	
...		15	2	23¼	25½	
...		20	½	25	24½	

Second Test.

Voltage at Terminals.	Time. Hrs.	Min.	d'	a'	b'	i amps.
...		0	16	31	15	8.0 constant
...		1	8 2/3	31 1/3	12 1/3	
1.73		3	14½	31	16½	
...		5	15	31½	16½	
1.67		27	15¾	31¾	16	
1.6		1.38	17¼	32½	15¼	
1.53		2.15	21	33½	12½	
1.49		2.30	24	34½	10½	
1.13		2.45	24¼	33½	9¼	
0.5		2.47	17	25	8	
Polarization rev.		2.48	19	32	13	
1.03		2.50	18	32	14	
0.93		3.00	17½	32	15½	

To sum up, the chief results of this investigation are as follows:

1. The ions which transport the electricity through an electrolyte do not select the "geometrically shortest" path, but the "electrically shortest" path, so that Ohm's law for metallic wires of uniform cross-section can be applied to electrolytes only in the special case, if there is no straying or distortion of lines of current.

2. In all those cases (that is, in all cases of practice) where the cross-section of the electrolyte is larger than the surface of one electrode, there is an "electrolytic shunt"; the lines of current no longer pass in a straight direction through the electrolyte, and the distribution of the current density over the electrode surfaces is no longer uniform. What I call the "electrolytic shunt" really acts as a shunt, *i. e.*, has the effect of diminishing the resistance, as may be shown by the use of a sensible voltmeter.

3. A large number of the troubles experienced with storage battery plates, like distortion of the plates, dropping out of active material, may be easily explained in accordance with what was said above under 1 and 2, since on account of the layer of acid between the lower rims of the plates and the bottom of the vessel (which layer of acid acts as electrolytic shunt), the plates carry more current near the bottom than at the top. This non-uniform current density at different parts of the plate results in non-uniform changes of volume. This necessarily results in mechanical distortions, whether we have to do with pasted plates or Planté plates. This is a point which, up to the present time, has not been properly considered, especially in the formation process.

4. At a storage-battery electrode which is not homogeneously charged or discharged strong local currents occur. Their existence and direction may be easily proved experimentally by means of suitable analyzers. It seems that lead peroxide plates behave in general far more uniform than spongy lead plates during charge and discharge.

Finally, I will say, that in the present paper I do not claim to have exhausted even approximately the subject. I rather hope that this paper will inspire others to continue these researches. It would be especially desirable if storage battery engineers would pay a little more attention to the important problem of the distribution of the lines of current in storage batteries.

Quarternary Steels.

The Iron and Steel Institute issued at its May meeting (in accordance with its usual practice) the abstracts of the Reports on Research Work carried out by the holders of the "Andrew Carnegie Research Scholarships."

The first of these is by Dr. Guillet, and deals with quarternary steels. The length even of this official abstract of a report is such that further condensation is necessary in order to indicate the author's conclusions within the limits which can be allotted thereto in the pages of a technical journal.

Nickel-Manganese Steels.—The author divides these into pearlitic, martensitic and Y-iron steels. All attempts of rolling the last-named have completely failed. The transformations which these steels undergo are similar to those already recorded in connection with nickel-steels and manganese steels.

Nickel-Chromium Steels.—The different structures which nickel-chromium steels can present are:

Pearlite, with ferrite or carbide.
Martensite.
Martensite with carbide, and
γ-iron with iron.

The mode of formation of the different structures on the addition of chromium to a nickel steel appear to be as follows:

A. If chromium be added to a pearlitic nickel steel the steel may, to begin with, preserve its pearlitic structure if the amount of chromium added be sufficiently small, and if the sum of the elements Cr + Ni be not very high. Under these conditions the only influence which the chromium exerts is to give the ferrite a much finer grain structure. If the addition of the chromium is sufficiently high, and if the percentages of carbon and of nickel are similarly sufficiently high the structure becomes martensitic.

B. If the chromium be added to a martensitic nickel steel the steel may at first remain martensitic, provided the addition of chromium is sufficiently small. Subsequently carbide is formed, the quantity appearing to depend upon the percentage of carbon, and finally a mixture of martensite, carbide and γ-iron is usually obtained. It sometimes happens that when the percentage of chromium is sufficiently high a structure composed of γ-iron and of carbide is obtained.

C. The addition of chromium to a nickel steel containing γ-iron at first causes no change in the structure, but when the percentage of chromium becomes sufficiently high carbide is formed. This phenomenon commences with a percentage of chromium which is lower in proportion as the amount of carbon is high, and, other things equal, the quantity of free carbide in a steel is higher in proportion as the percentage of carbon and of chromium in the steel rises. In a word, the effect of the chromium is superadded to that of the nickel, with the production of martensite or of γ-iron, but if the percentage of chromium is sufficiently high, carbide is produced in amounts which increase according as the percentages of carbon and of chromium increase. The mechanical properties

of these steels may be deduced from their microstructure, for instance, pearlitic nickel-chromium steels possess a degree of tensile strength and elastic limit which becomes higher the larger the percentage of carbon, nickel and chromium present. The elongations are rather lower than in ordinary steels corresponding to the same percentage of carbon. Martensitic nickel-chromium steels possess exceedingly high tensile strength and elastic limit; the elongations are somewhat low. Nickel-chromium steels containing martensite, together with the double carbide, have practically the same mechanical properties as the martensitic steels. The tensile strength and elastic limit diminish, however, in proportion as the percentage of carbon increases. Nickel-chromium steels with γ-iron have a higher tensile strength and elastic limit than corresponding steels not containing chromium. Their elongation and resistance to shock are somewhat lower and their hardness greater. Steels containing carbide together with γ-iron have practically the same properties as steels containing Y-iron alone, except that their elongation is rather lower. In short, the properties of nickel-chromium steels resemble those of nickel steels of the same microstructure, except that the addition of chromium somewhat raises the tensile strength and the elastic limit, without, however, much diminishing the elongation and the resistence to shock. In addition to this the hypo-eutectoid chromium-nickel steels containing carbide—a structure which is not met with under similar conditions among nickel steels—are somewhat brittle, while, at the same time, possessing average elongation.

The effects of quenching annealing and case hardening are discussed.

Nickel-Tungsten Steels.—In the case of two series of steels, in which the percentages of carbon and of nickel was such that, had it not been for the tungsten present, they would have been intermediate between the pearlitic and martensite steels, it was found that: (1) The tungsten commences by becoming dissolved in the iron, and favoring, to a slight extent, the production of martensite; (2) when the iron-nickel solution is saturated with tungsten the latter forms a carbide, the quantity increasing with the percentages of tungsten and of carbon present; (3) the addition of tungsten distinctly raises the tensile strength, and slightly raises the elastic limit in normal steels. The elongation, contraction and resistance to shock are lower than in the case of steels not containing tungsten, but vary slightly with the proportion of that element present, whereas, the hardness increases in proportion to the amount of tungsten.

It was also noted that tungsten has a material effect on the mechanical properties of the steel which, after suddenly quenching in water at 870° C., possesses an exceedingly high tensile strength and elastic, with a medium elongation, contraction and resistance to shock.

Nickel-Molybdenum Steels.—Concerning these, the author states that: (1) The molybdenum dissolves first of all in the iron and furthers the formation of martensite. In this respect its influence is decidedly greater than that of tungsten. As a matter of fact, no steels containing pearlite were found, whereas, martensitic steels were encountered when the percentage of molybdenum exceeded 2 per cent, a phenomenon not met with in the case of nickel tungsten steels; (2) when the amount of tungsten present becomes sufficiently high, carbide is formed. In the series of steels under investigation the appearance of the carbide in steels containing the same proportions of nickel and of carbon as the tungsten nickel series, appeared to take place with the same percentages of molybdenum as it did in the case of tungsten. (3) the influence of molybdenum is of the same nature as that of tungsten, but it acts with very much greater intensity. (4) this difference between the action of tungsten as compared with that of molybdenum is much less pronounced in the case of quenched steels. Commercially, the author does not regard these as interesting or as economically valuable as the nickel tungsten steels.

Nickel-Vanadium Steels.—The chief point raised is that on quenching a 6 per cent nickel steel, in order to obtain the maximum tensile strength, a very much smaller quantity of vanadium is required than in the case of a steel containing only — 2 per cent of nickel. There is a decided advantage in raising the proportion of nickel as high as possible if it be desired to obtain the maximum mechanical properties after quenching. At the same time it is necessary to preserve the pearlitic structure sufficiently to secure that the metal may be easily worked. Secondly, it is highly undesirable that to increase the amount of carbon present in the steel, as this injures the mechanical properties of the steel and increases its brittleness. The author believes more firmly than ever in the commercial future of nickel-vanadium steels, although he is now in a position to state that investigations have shown the necessity of restricting its composition within the following limits: Carbon, 0.10 to 0.30 per cent; nickel, 2 to 7 per cent, and vanadium, 0.10 to 0.30 per cent. It is quite possible that it might become necessary to reduce the amount of vanadium to 0.05 per cent.

Nickel-Silicon Steels.—The author divides these into seven classes, but these need not be described in detail in this abstract, as this series has little commercial value. The presence of silicon raises the transformation points of nickel steels, and consequently alters the percentages of nickel at which transformations from one structure to another usually occur. In this way it facilitates the transformation of γ-iron into martensite. If silicon, added to a nickel steel, permits quenching to effect a more intense influence, it must not be forgotten that this is at the expense of its resistance to shock, particularly at right angles to the direction of rolling. In short, the addition of silicon to a nickel steel is not to be recommended.

Nickel-Aluminium Steels.—The steels containing nickel and aluminium which have been examined, present, from a micrographic point of view, the same phenomena as those containing aluminium alone. When the percentage of this metal is sufficiently high the pearlite assumes a granular form, and its characteristics closely resemble those of troostite. The addition of aluminium produces, however, in nickel steels, an increase in tensile strength and in elastic limit, which has not been observed in the case of aluminium steels. The elongations decrease and finally reach very low values. It is of interest to note that aluminium does not impart any tendency to the formation of martensite, its effect being just the opposite.

Manganese-Chromium Steels.—For corresponding percentages of manganese or of nickel, the manganese-chromium steels are absolutely comparable to nickel-chromium steels. From the commercial point of view it would, therefore, appear that nickel can, in certain types of commercial steel, be replaced by manganese. This is particularly the case with the sample containing carbon, 0.250 to 0.4 per cent; nickel, 2 to 3 per cent, and chromium, 0.5 to 1.0 per cent, in which the nickel may be replaced by 1 to 1.5 per cent of manganese. This characteristic would seem to merit special investigation, which the author hopes to be able to carry out shortly.

Manganese-Silicon Steels.—Broadly speaking, the addition of silicon weakens the mechanical properties of normal manganese steels, but it confers great advantages upon quenched steels, as it induces a marked increase in the tensile strength and in the elastic limit. When the percentage of manganese is low a superior description of spring steel is met with. In no instance does the addition of silicon to a manganese steel containing γ iron appear to be justified. From the commercial point of view manganese steels with a high percentage of manganese should contain less than 1 per cent of silicon. Manganese silicon steels with a low percentage of manganese (*i. e.*, below 1 per cent) are of great interest, the use of these steels for making springs being widespread.

Chromium-Tungsten Steels.—The principal conclusion to be derived from the investigation of chromium tungsten steels have been as follows:

A. The constituents found in these steels in the normal state are pearlite, martensite and a carbide, which is apparently a triple carbide of iron, chromium and tungsten, and may be accompanied by martensite, by sorbite (or troostite) or by γ-iron.

B. The structure most frequently met with in normal chromium-tungsten steels, which is that of the rapid tool steels employed commercially, is built up of grains of carbide on a background of troostite or of sorbite.

C. With regard to the mechanical properties the chief characteristics may be summarized as follows:

(1) Pearlitic steels: tensile strength and elastic limit higher than that of ordinary carbon steels; elongation and resistance to shock lower.

(2) Martensitic steels: tensile strength and elastic limit very high; elongation and resistance to shock low.

(3) Steels containing the carbide: resistance to shock very low.

Variations in these properties depend upon the constituent which accompanies the carbide; with martensite the tensile strength and elastic limit are very high; with sorbite the tensile strength, usually moderately high (about 90 kilogrammes per cubic centimeter) varies slightly, the elastic limit is medium, as are also the elongations; with γ-iron the tensile strength depends upon the percentage of carbide present, and is high in proportion as this constituent is present in bulk, the elastic limit is low, and average elongations are obtained.

D. Generally speaking, treatment reveals nothing that has not already been noticed except in the case of quenching. The influence of the latter treatment may be summarized as follows:

(1) On pearlitic and on martensitic steels quenching has an influence which has already been investigated in several categories of steels, viz.: it transforms pearlitic steels into martensitic steels, at the same time raising in a marked manner their tensile strength and their elastic limit and lowering their elongation and resistance to shock. These transformations are of much greater importance than in the case of ordinary carbon steels. The martensitic steels are slightly softened by quenching, owing to the formation of a small amount of γ-iron.

(2) Steels containing the carbide give exceedingly interesting results on quenching, varying with the temperature and the duration of heating. The higher the temperature of heating, above 850° C., the period of heating the temperature remaining constant (or the longer the period of heating the temperature remaining constant), the smaller is the amount of carbide which remains after quenching. In the steels under investigation the author was able, by varying the duration of heating, while maintaining the temperature at 1,200° C., to cause the whole of the carbide to disappear and to obtain with steels, which normally contained sorbite and carbide, a structure which would appear to be that which is required in rapid tool steels, viz.: an exceedingly fine structured martensite, almost invisible under the microscope. This proves that, commercially speaking, in order to obtain the best results with rapid tool steels they should be subjected to a definite temperature treatment and period of heating prior to quenching, which should vary with the quantity of carbide contained in the steel, and consequently with its composition.

Regarded commercially, a study of these classes of steel leads to the following conclusions:

Pearlitic steels possess very little interest for mechanical purposes. They reveal properties of less interest than nickel chromium steels and are much higher priced. It may be, however, that they may prove useful in making races for ball bearings or the balls themselves, owing to their extreme degree of hardness.

The only commercial application of these steels that appears feasible is for making high speed tools, but this is, of course, of great importance, as has been clearly shown by the extra ordinary rapidity with which these steels have [illegible] universal acceptance. The investigations appear to [illegible] to the conditions under which the steel is utilized commercially as a high speed steel. It should [illegible] normal rate looking to any of the following [illegible]

Pearlitic steels, which will not do for high speed [illegible]

Martensitic steels, which are too difficult [illegible]

Steels containing martensite and carbide, in which the presence of γ-iron renders too great a degree [illegible]

Steels for industrial use should, therefore, be either [illegible] troostite or should contain carbide. The [illegible] should be subjected to is such that, after annealing, the whole of the carbide should be dissolved. The rate of cooling and the composition appear to influence them but slightly; the length of time and the temperature of heating alone are of importance.

In reviewing this research as a whole, the author points out that a given set of mechanical properties corresponds to a definite structure. It is necessary to distinguish between a simple structure in which a single constituent is found and a more complex structure in which two or more constituents are present. The simple structures are martensite and γ-iron. The martensitic structure corresponds with the following physical properties:

Tensile strength and elastic limit very high.

Elongation medium, sometimes very low.

Great hardness and great difficulty in working and forging.

A γ-iron structure corresponds with the following properties:

Tensile strength fairly high, elastic limit low.

Elongation and resistance to shock usually good, although under certain conditions they may become average only.

The steels are easily forged, but are difficult to machine. Steels of a complex structure may be described as follows:

(1) All the graphite steels are unserviceable; from the moment the graphite appears the steel is brittle, and it becomes impossible either to roll or hammer it.

(2) Steels containing a carbide possess properties characteristic of the other constituent accompanying it. These properties are, however, more marked, and in particular the brittleness is increased.

With pearlitic steels it becomes impossible to foretell any of the qualities, as they depend not upon the quantity of the pearlite alone, but more particularly upon the elements, other than carbon, present. It may, however, be said that the tensile strength and elastic limit are seldom very high, although nothing can be known beforehand as to the brittleness or the elongation. Quenching usually transforms pearlite to martensite, provided it is carried out at a favorable temperature. An exception, however, must be made in regard to steels containing a fairly high percentage of aluminum. Simple treatment leaves the martensite unchanged, or simply exerts upon it a tendency towards the formation of γ-iron with the resulting lowering of its properties.

Sorbite is transformed into martensite if the steel is quenched at a suitable temperature. γ-iron is not produced on quenching, except in certain steels which are on the boundary line of the martensitic steels. With regard to steels containing carbide, the alterations induced by quenching depend upon the nature of the carbide. For the carbide of chromium steels a temperature of 1,200° C. is required for solution for carbides containing tungsten or molybdenum even when chromium is present, a temperature of 850° C. is sufficient, but the lower the temperature and the higher the amount of carbide present, the longer should the period of heating be sustained; carbide of vanadium is never altered by quenching. Annealing does not, as a rule, alter the structure. If it does it becomes [illegible], the needles of martensite better formed, and the particles of γ-iron and the carbides are somewhat larger. An exception must be made in the case of certain steels which are on the border line of the martensitic steels. These

become transformed with the formation of martensite, while certain steels containing silicon or vanadium have their percentage of graphite increased.

Cooling and hammering are generally without influence on structures, although an exception must be made in the case of certain nickel steels, particularly those which are on the border line of martensitic steels. No matter what transformation may occur, the conclusions to be drawn as to the latter are based upon the considerations obtaining in the case of normal steels. Thus, the case of a γ-iron steel may be taken as an example. On becoming martensitic such a steel gains considerably in tensile strength and elastic limit, while it acquires a lower elongation and resistance to shock.

From these researches it would appear that the area for the commercial employment of these steels is considerably restricted, and that steel manufacturers should not depart from the limits which have been based upon the examination of the micro-structure. To begin with, the manufacturer should reject all steels having a structure which contains either graphite or martensite. It is needless to revert to the subject of the graphite; as regards the martensite, its presence creates such difficulties in working and forging the steels that it is difficult to see to what uses steels possessing this structure can be applied.

Steels with a carbide possess no interest when they at the same time contain γ-iron. It is only when they are pearlitic or sorbitic that they can be of use, and even in this case they afford opportunities for application in a few special cases only; the most interesting being for tool steels and for steels for bearings.

There remains, therefore, the two structures:

Pearlitic steels and

Steels containing γ-iron.

The latter can only be obtained by using high percentages of nickel or of manganese or both, to avoid obtaining steels too readily transformed by quenching, annealing and cooling. Indeed, the percentages required are considerably higher than is usually imagined. Hence the cost of manufacture is somewhat high. However, it is necessary to recall the fact that the elastic limit of these steels is very low, and that they are very difficult to work. This renders the field for their employment exceedingly limited. Thus the following conclusion is arrived at:

Setting aside steels containing pearlite and a carbide, or sorbite and a carbide, which possess considerable interest for tool-steel purposes, and, in certain special cases, for machine parts, and those containing γ-iron, which can only be resorted to in exceptional circumstances, the only type which should be sought for general application is that of steels with a pearlitic structure. This statement must be further limited by saying that steels possessing this structure and its corresponding mechanical properties should not, as a rule, contain much carbide. The field for their industrial application is thus limited to steels containing relatively low proportions of foreign elements, amongst which by far the most interesting which the author has studied are:

Nickel-vanadium steels and

Nickel-tungsten steels.

To which must be added, if the results obtained have been rightly interpreted,

Chromium-vanadium steels

which the author hopes shortly to make the subject of a separate, complete investigation.

Balance. Under the title, "The Equal Arm Balance" Cyril Penney and Bissett published in the January issue of the *Physical Review* a paper reducing the basic equations of the balance and applied the theory successfully to two commercial balances. In the March issue of the *Western Chemist and Metallurgist* V. H. Gottschalk performs a very useful work in translating this mathematical theory into non-mathematical statements.

Two Electrochemical Processes for the Extraction of Silver and Gold.

The experiments to be described below were made by Dr. Mooshegh Vaygouny, at the University of California from 1903 to 1905, the complete account having recently been published in form of a thesis for the degree of Ph. D.

The experiments were made with some of the new well-known Tonopah gold and silver ores. The object was to find an lixiviation process for the extraction of both gold and silver, without any preliminary roasting of the ore; the solution should neither be costly nor have any properties deleterious to health; in order to avoid the necessity of its being constantly renewed, it should be regenerated; in other words, the process should by cylic.

Two independent cyclic processes were developed by Dr. Mooshegh Vaygouny.

Chloridation.

The first or "chloridation" process depends upon the use of acid solutions of ferric chloride or sulphate, in presence of high concentrations of soluble chlorides, especially hydrochloric acid or sodium chloride, which act as solvents of silver chloride. Some information on this process has already been given on page 36 of our Vol. III.

The average extraction, under proper conditions, was over 95 per cent silver and from 60 to 88 per cent gold, varying with the character of the ore. The metals dissolved were recovered by electrolysis. It is of special interest that under proper conditions the re-oxidation of the ferrous salts formed during the decomposition of the sulphides, and the consequent regeneration of the ferric salts could be readily accomplished without the use of any diaphragm whatever. It was further found that there was no difficulty in carrying the re-oxidation of the solution so far as to even charge the latter with much free chlorine, which could then act as an efficient solvent of gold.

A very small percentage of glue added to the solution was found to give good coherent deposits of the metal.

Since an account of this "chloridation" process has already been published, these few remarks must suffice.

Persulphatation.

The second method depends on the use of persulphates, also in conjunction with concentrated solutions of chlorides.

In this method the chief point of interest is that both the oxidation of the sulphides of silver and of gold is brought about under the influence of one and the same compound, namely, the persulphate used, without any necessary dependence upon the generation of chlorine by electrolysis.

In bringing this double effect about, advantage is taken of the fact that persulphates decompose chlorides, setting chlorine free, ready to attack gold, especially in presence of free acids, and that, on the other hand, they attack sulphides, readily converting them into sulphates while they themselves break down to simple sulphates.

The reaction taking place in the first case seems to be of the type

$$2NaCl + K_2S_2O_8 + H_2SO_4 = Na_2SO_4 + K_2SO_4 + H_2SO_4 + Cl_2$$

the decomposition of the chloride being due to the formation, first, of free persulphuric acid, thus:

$$K_2S_2O_8 + H_2SO_4 = K_2SO_4 + H_2S_2O_8$$

which then reacts upon the chlorides in a manner now well known:

$$2NaCl + H_2S_2O_8 = Na_2SO_4 + H_2SO_4 + Cl_2$$

While with the ordinary concentrations of acids and persulphates the attack of chlorides can be called all but rapid at best—a fact which is rather fortunate from the practical standpoint, seeing that the nature of the work to be done by the chlorine thus liberated, namely, the solution of gold, requires

only a slow but continued oxidizing effect—that of the sulphides, on the other hand, is much more readily accomplished. In fact, this decomposition goes on readily, even in neutral or slightly alkaline solutions, though it is certainly more vigorous in the presence of acids. The reactions here in question seem to take place as follows:

$$Ag_2S + K_2S_2O_8 = Ag_2SO_4 + K_2SO_4 + S$$

though much of the sulphur forming the metallic sulphides is oxidized completely over to sulphuric acid.

These being the mechanism and character of the reactions immediately involved in the use of solutions of persulphates for the solution and extraction of the precious metals, it is evident that, as with the chloridation method, so also here it is essential for the successful working of the method in actual operation to have these solutions charged with a soluble chloride. The introduction of such chlorides is, indeed, important in this case, not merely because it is necessary to have some chloride present in the solution so as to react with the persulphates and liberate the chlorine necessary to dissolve the gold in the ore, but also because AgCl is no more soluble in simple persulphate solutions than in ordinary solutions of iron salts, so that there is here the same danger of much silver being lost in consequence of the unavoidable presence of chlorine compounds occurring as impurities in ores, unless some appropriate solvent of AgCl is used here as with the chloridation method.

But there is even a more grave reason for which such a solvent of silver should be used in this connection, and that is that silver salts form an insoluble percipitate of peroxide of silver (AgO), in presence of persulphates, which is only decomposed and dissolved satisfactorily by strong chloride solutions.

In endeavoring to select an appropriate solvent for these insoluble silver compounds, it was at first thought that ammonia would be a desirable substance to use in conjunction with this method, inasmuch as it would act as a very efficient solvent of any silver chloride which might form in the course of the treatment, while, at the same time, the action of persulphates upon the sulphides would not thereby be impaired in any way. This belief, had, however, soon to be abandoned.

It was, indeed, observed that, far from being a suitable substance for the purpose, ammonia was decidedly objectionable, inasmuch as it was found to be rapidly decomposed into free nitrogen and water whenever any appreciable amount of silver and a persulphate were found in solution with it. This oxidation of ammonia under these circumstances is due, doubtlessly, to a catalytic reaction depending upon the formation, primarily, of the peroxide AgO, which then attacks, more or less violently, any free ammonia which may be present in the solution, and causes its complete decomposition in a well-known manner.

The reaction is, indeed, so characteristic that Dr. M. Vaygouny has found it to be a very handy way for detecting the presence of silver salts in chloride solutions, which is the more useful, as it is often impossible to detect silver in such solutions whether by any other simple precipitation method or by a mere dilution of the solution, with the hope of thus diminishing the solvent power of the liquid on AgCl, and thereby causing a turbidity of this compound to appear. In carrying out this test, a strong solution of a persulphate, preferably the ammonium salt, is added to the solution to be tested, and then this latter is made strongly alkaline with NH_4OH, when, if silver be present, a more or less rapid evolution of nitrogen gas will be observed to take place.[1]

The use of ammonia as a solvent of insoluble silver salts being thus out of question, and none of the other non-chloride solvents being sufficiently stable in presence of persulphates, the choice of concentrated solutions of a chloride thus becomes a necessity.

Now, in the first place, persulphates appear to be good reagents for treating ores of the precious metals, inasmuch as they are good oxidants, capable of being regenerated by the electric current; further, though a small percentage of a chloride in concentrated solutions of sulphates is very advantageous for the electrolytic formation or regeneration of persulphates, higher percentages of chlorides are quite injurious to such regeneration. The question may, therefore, arise here as to whether the means always stated for holding some salts in solution in actual practice would not render the process as a whole inoperative, at least as a "cyclic" process. It is, indeed, only too evident that, an essential condition for the formation of persulphates by means of the electric current being a high rate of discharge, of SO_4 ions at the anode, the introduction of unduly large proportions of a non-sulphate electrolyte, such as NaCl, will tend to diminish the yield of the current in persulphate formation. How this difficulty may be overcome in practice will be discussed further on.

In comparison with the chloridation method the following advantages of the persulphatation process may first be mentioned:

1. Theoretically the use of a persulphate should be preferable to that of ferric salts alone, because, unlike the latter, whose reaction products—the ferrous salts—are notorious for their tendency to reprecipitate the precious metals from their dissolved state, persulphates break down merely to sulphates, that is, salts having no such injurious effects upon the solutions of these metals.

2. For this reason, principally, simple gold ores could be treated by persulphatation to a better advantage than by chloridation, or perhaps by any other wet method.

3. In practice one may have to treat ores that are too rich in limestone to be economically handled by the chloridation process, inasmuch as they would first have to receive a preliminary acid treatment before they could be subjected to the action of ferric salts, lest these latter should be precipitated out and the process rendered more or less inoperative. Such ores should be amenable to persulphatation to a better advantage, seeing that persulphates can decompose sulphides of silver in neutral or slightly alkaline media as well as in acid.

On the other hand, with very "base" ores, persulphatation cannot be expected to be nearly as useful a method as chloridation, owing to the fact that the satisfactory oxidation of such ores by wet methods such as these, requiring, as they do, the heating of the reacting solutions, persulphates would be rapidly decomposed under these conditions, and hence be useless as economic agents for treating this type of ores.

The persulphate used by the author was the potassium salt throughout; for the reason that the ammonium salt is rather unstable, while sodium persulphate could not be had on the market in California nor can it be made easily or readily in a solid condition.

Both the decomposition of sulphides and the solution of gold can be best brought about by persulphates when some free acid is present. Moreover, more or less ferric salts are introduced into the solution through the agency of the acidity of the liquids upon the compounds of the iron which may be in the ores, and hence the oxidizing power of the solution is thereby enhanced.

Comparative experiments of the author show that the chief function of the acid lies in that it facilitates the solution and extraction of the gold rather than that of the silver values. On the other hand, the variation in the concentration of the persulphate in solution effects especially the extraction of silver alone. From the different tests of the author it appears that the best conditions for the process involve the use of a solution containing 2 per cent. of persulphate of potassium and 5 per cent. of sulphuric acid with 20 per cent. of sodium chloride.

An intermediate leaching of the [illegible] with a solution of an alkali, such as sodium hydrate, facilitates the ensuing

[1] This catalytic reaction has also been studied and similarly explained by Hugh Marshall (J. Chem. Soc., 1901, Abstr. II., page 156).

[illegible] in Elektrochemie, 1898, page 34.

attack of the sulphides. This was found to be due to the fact that such alkalies dissolve away the sulphur and help to remove this element, which otherwise forms a coating around the particles of the sulphides, and thus hinders their further decomposition. The reaction giving rose to the formation of free sulphur may be representel thus:

$$Ag_2S + K_2S_2O_4 = Ag_2SO_4 + K_2SO_4 + S$$

That the beneficial effect of NaOH is in reality due to its power to merely dissolve considerable amounts of spongy sulphur and help to remove it out of the ore mass, thus laying bare the remaining particles of sulphides subject to the further attack of the main oxidizing solution, is proven by the fact that the now impure, brownish solution of NaOH, recovered after the treatment of the ore with it and after filtration of same, gave a decided precipitate of yellow, flocculent sulphur when neutralized with H_2SO_4, along with a little H_2S.

Some experiments were also made with alkaline ores so as to determine to what extent such ores might be treated with the persulphatation process without necessitating the preliminary neutralization of their carbonates with correspondingly large amounts of acids. With respect to the treatment of such ores the persulphatation process has a decided advantage over the chloridation process.

As has already been mentioned above, the regeneration of the persulphate solution presents difficulties, while the electrolytic recovery of the precious metals from the sulphate solutions is as easy as in the chloridation process.

The difficulty of the regeneration of the persulphate solution is due to the presence of the large amounts of chlorides. To work the process on a large scale it would seem to be necessary to remove part of the sulphate in solution from an aliquot part of the latter, conferring it into a persulphate in a special electrolytic cell (containing no chlorides), and finally returning it to the main solution as needed.

There are several methods of solving this problem. If the sulphate worked with in practice be the potassium or ammonium salt, part of the liquors holding this latter in solution may be shunted off—best directly after the recovery of the precious metals—and a desired amount of the sulphate made to separate out in the solid form, either by concentrating the solution with respect to this salt by evaporation, or by cooling it sufficiently, according to practical facilities. In either case, the solution being more nearly saturated with respect to the sulphate than the chloride present, the former salt will naturally separate out first, in form sufficiently pure and free from admixed chlorides to be redissolved in a fresh, sufficiently chloride-free solution of the same sulphate, and electrolyzed in special electrolytic cells, with a view to converting it at least in part, to persulphate, which may then be removed from the solution and returned to the stock liquors.

While, with potassium sulphate chosen as the starting material to work with in practice, the evaporation method seems to be much more likely to be practicable, with the sodium salt (Na_2SO_4), on the other hand, the separation of the solid Na_2SO_4 from the shunted solutions by cooling seems to be equally likely to meet with success in practical operations. Indeed, the very rapid decrease of solubility of the Na_2SO_4 with the lowering of the temperature of its solutions, and the very slight decrease under the same conditions of the solubility of NaCl, which may be present in the same solution, render this method highly useful as a means of bringing about the separation of the former salt from a concentrated solution of both these compounds in a form sufficiently free from the latter for all purposes concerned, and that with only a slight lowering of the temperature of the solution.

Troubles may be experienced in the latter method, due to the difficulty of preparing the persulphate of sodium in solid form owing to its excessive solubility. The author thinks that it is not absolutely necessary to prepare the persulphate in the solid form, but that the sulphate of sodium (whether original or recovered from the shunted parts of the stock solutions) may be dissolved in water and then converted, to any desired extent, into a solution of the corresponding persulphate, which solution may then be returned, in body, to the main or stock liquids.

German Bunsen Society—Report of Dresden Meeting—II.

By H. Danneel, Ph. D.

Nitrogen Oxides from Air.

The various papers which were presented on the subject of the fixation of atmospheric nitrogen (and a digest of which was given in our July issue, p. 256) elicited considerable discussion.

Prof. Biltz asked whether it would not be practical to try and produce nitrogen pentoxide. Its temperature of decomposition is very low, and a catalytic process could be easily applied, in order to get a high efficiency and high reaction velocity. N_2O_5 dissolves easily in water, forming HNO_3.

Prof. Foerster stated that a mixture of steam, NO_2 and O_2 easily reacts, forming nitric acid.

Prof. Warburg said that it is easy to absorb the dilute NO_2 gases by KOH or NaOH, if some O_2 is added to the gases. Prof. Nernst called attention to the fact that this might yield a very suitable analytical method for testing the dilute NO_2 gases.

Prof. Helbig stated that N_2O_4 in liquid as well as in gas form is easily oxidized by O_2 to N_2O_5. It seems, therefore, that the effect of O_2 is not catalytic, but a simple oxidation.

Prof. Haber doubts whether an output of 900 kg. HNO_3 per kilowatt-hour has really been obtained on a large scale. In general it will hardly be possible to get more concentrated gases than 2 per cent, and the efficiency will scarcely be higher than 600 kg. HNO_3 per kilowatt-hour. Prof. Klandy replied that 5 per cent gases and an efficiency of 900 kg. have probably been obtained in the laboratory on a small scale only.

Dr. Erlwein, of the Siemens & Halske Co., then showed furnaces and apparatus for the production of calcium cyanamide. Ground calcium carbide is placed in retorts heated to 500° or 900° C. Nitrogen is then introduced and is quickly absorbed by the carbide. The cheapest method of producing the nitrogen is by means of liquid air. At the cyanamide plant in Piano d' Orta, Italy, the cost of producing the nitrogen from liquid air is 0.75 to 1 cent per kilogram, and 330 kg. of bound nitrogen are obtained per horse-power-year.

Dr. Kraus called attention to a patent of Polzeniusz, who mixes 10 to 15 per cent of $CaCl_2$ with the carbide. This mixture absorbs nitrogen at a temperature of 700° to 750° C., while pure carbide requires a temperature of 900° C. Dr. Arndt thinks that this effect of $CaCl_2$ is due to the fact that it is partially dissociated already at 700° C., while Prof. Foerster and Prof. Nernst believe that the molten $CaCl_2$ acts here as solvent, so that the reaction proceeds more easily. This is evident, for instance, from the fact that only such additions are effective, the melting point of which is below that of carbide, for instance, $CaCl_2$ and CaF_2, but not $CaSO_4$.

In reply to a question of Dr. Goldschmidt, Dr. Erlwein stated that ordinary cheap commercial carbide may be used, and that there is no need of taking expensive brands.

Prof. Hempel called attention to the large quantities of nitrogen wasted in water-closets. A problem of engineering of the future will be to find a method which combines the cleanliness of water-closets with a possibility of preserving the nitrogen compounds. Prof. Haber called attention to the possibility of utilizing the coke oven gases.

Explosives.

Prof. Will then read a paper on technical methods of testing explosives. For the efficiency of an explosive, not only the heat which is developed is of importance, but also the reaction velocity. The pressure of explosion is a measure of the operating capacity of an explosive. The author gave a review of various technical methods for testing explosives, and pointed out that to get a correct idea of the value of an explosive several methods must be employed.

Standard Cells.

A paper by Dr. H. von Steinwehr, of the German Reichsanstalt, discussed the influence of the size of the grains of Hg_2SO_4 on the e. m. f. of the Clark cell. Thomson has first shown that the vapor tension of small drops of water is larger than that of greater surfaces of water. Ostwald has applied this to the solubility of salts, and has found, for instance, that fine HgO is more soluble by several per cent than coarse HgO. The e. m. f. of a combination Hg, fine HgO, coarse HgO, Hg may amount to 0.6 millivolt. Hg_2SO_4 behaves in a similar way. In order to get constant standard cells it is necessary to use very coarse Hg_2SO_4.

Catalysis.

Prof. G. Bredig spoke on catalysis in heterogeneous systems and a new mercury oxide. Platinum black and colloidal platinum in solution accelerate greatly the reaction $2H_2O_2 = 2H_2O + O_2$. If used often the platinum becomes less active, and cannot be rejuvenated. Platinum also loses its efficiency if "poisoned" by certain reagents like KCN. This poisoning can be counteracted by purification. Platinized platinum sheets ("macroheterogeneous catalysis") behave just like a colloidal platinum solution ("microheterogeneous catalysis"), if we neglect small differences. In both cases Nernst's law is fulfilled, according to which the reaction velocity is due to a diffusion phenomenon, the diffusion drop being within a thin layer in the catalyser. By experiments of Nernst and Brunner this layer has been found to be 0.02 mm. thick. The same value also follows from the experiments of the present author. Metallic mercury is also a catalytic agent accelerating the decomposition of H_2O_2. A brown oxide of the formula HgO_2 is formed. It is strongly explosive.

Allotropic Forms of Gold and Silver.

Prof E. Cohen discussed this subject. According to Thomson, there exist three allotropic forms of gold, as has been concluded by him from calorimetric measurements. According to Berthelot, there exist five allotropic forms of silver. The present author argues that different allotropic modifications of the same metal should differ at least in their free energies. He has prepared these alleged different modifications exactly according to the instructions of Thomson and Berthelot, and has measured their potentials against each other. No difference was found.[6] The author concludes that the "different" forms are identical.

Decomposition of Nitrites.

Prof. R. Abegg discussed the decomposition of nitrites. Silver nitrite, $AgNO_2$, when heated decomposes, according to the formula

$$2AgNO_2 = Ag_2O + NO + NO_2,$$

or written as ionic reaction

$$Ag^{\cdot} + 2NO_2' = Ag + NO_3' + NO$$

The reaction is completely reversible and the equilibrium depends on the NO pressure. One NO_2' ion robs the $Ag^{\cdot}$ ion of its positive charge, and simultaneously gives off one O to another NO_2' ion. We have, therefore, a real oxidation equilibrium, and the energy of the reaction may be found by measuring the e. m. f. of the combination Ag, Ag solution, solution of $NO_2 + NO_3 + NO$, Pt. The oxidation potential of the reaction $2NO_2' = NO_3' + NO$ may be measured, than the potential of the reaction $Ag = Ag$ if the [illegible] good at all.

The potential of the latter reaction is -0.77 volt, and that of the former reaction was found to be -0.5 volt. [illegible] the decomposition goes on with an e. m. f. of 0.27 volt, if all substances participating in the reaction are present with the concentration 1.

In general it follows that all those nitrites will decompose, the metal of which is more noble than -0.5 volt; e. g. platinum nitrite, gold nitrite, etc. The e. m. f. of decomposition changes, of course, when the concentrations change. If, for instance, complex salts are formed in the solution which remove the NO_2' ions, a nitrite with a metal nobler than -0.5 volt may not decompose. On the other hand it is possible to force a nitrite with a non noble metal to decompose according to the law of mass action, if one removes the reaction products; for instance, if one takes care to remove NO continually from the solution.

Alkali metal nitrites are stable. Their solution also contains the ion H, which may be reduced according to the reaction

$$H^{\cdot} + 2NO_2' + H = NO_3' + NO$$

The reaction of the H ions proceeds easily if the H gas and the NO gas are continually removed from the solution. For this reason the alkali metal nitrites, when left in the open air, must decompose with development of H_2 and must become alkaline; the oxygen in the air continually oxidizes any NO which is formed to N_2O_3. This is a well-known fact.

Radiation Laws.

Prof. Lummer, of the German Reichsanstalt, gave a very interesting review of recent progress in the theory of radiation. It is, of course, impossible to abstract such a paper. The author distinguished two kinds of radiations. First, rays which simply transport energy from the radiating body; besides light rays, Roentgen rays also belong to this class. Second radiations in which simultaneously material particles are thrown off; radium rays and cathode rays are examples of this class. Electrons always participate in radiations. Thus ultraviolet light dissipates electric charges. The author then gave a review of the corpuscular, or electronic, theory of the constitution of matter, and discussed the production of the spectrum of an element from this point of view.

Ammonium.

Prof. A. Coehn spoke on ammonium. The metallic radical NH_4 has not yet been prepared, since it was found impossible to fulfill the conditions under which it will be stable. On the other hand, it is possible to produce electrolytically ammonium amalgam which, however, is decomposed when no longer subjected to the influence of the cathodic potential; during the decomposition NH_3 and H are formed. The decomposition is accompanied by an evolution of energy, and since it is somewhat similar to the disintegration of radium it did not seem impossible that radiations would be produced in this case, just in the case of radium. Prof. Coehn endeavored to prove experimentally that such radiations occur.

The mercury cathode, which was contained in a glass vessel (somewhat similar in shape to the small type of German smoking pipe), was placed in an NH_4Cl solution and electrolyzed for 3 minutes with constant current. The electrode was then quickly removed and placed below a metallic disc. When this disc had previously received a negative charge this electrostatic charge decreased, on account of the presence of the disintegrating amalgam. When the disc had not been previously charged it assumed a positive charge, and the amalgam remained negatively charged. A very small quantity of amalgam only is required to produce this effect. If only 0.0004 gram NH_4 was in the amalgam the electrostatic effect was evident. With potassium or sodium amalgam no such effect was observed.

The author has not directly proven that this is a radiation phenomenon, due to the disintegration of the amalgam, although it seems so.

Colloids.

Three papers relating to this subject were presented. Dr. A. Lottermoser gave a very concise and complete summary of the whole subject of hydrosoles, Dr. Siedentopf spoke on colloidal alkali metals. He also exhibited the new "ultra-microscope."

Prof. R. Zsigmondy discussed the size of the particles in hydrosoles. The study of hydrosoles has been greatly furthered in recent years by the invention of the "ultra-microscope." In some hydrosoles the particles can be directly seen under the microscope. These visible particles the author calls submicrons, while those particles which are no longer visible are called amicrons.

The author determined the size of the latter particles in a very clever way. Such small gold particles may be used as a catalytic agent to disturb the equilibrium of an over-saturated silver solution. The silver then crystallizes around each gold amicron, so that the different amicrons get bigger in size and can now be counted under the microscope.

Preliminary experiments on the size of the microns gave dimensions of 1.7 to 3 $\mu\mu$. By the way, it may be mentioned that 5 $\mu\mu$ is the order of size of large molecules like starch molecules, which have a molecular weight of 30,000. The submicrons are seen under the microscope to be in continual motion.

Passivity and Catalysis.

Dr. O. Sackur presented the following picture of the anodic solution of metals. A metal tends to get into equilibrium with the hydrogen ions which are always present in the solution, according to the formula

$$2Me + 2H^{\cdot} = 2Me^{\cdot} + H_2,$$

or with bivalent metals, according to the formula

$$Me + 2H^{\cdot} = Me^{\cdot\cdot} + H_2$$

On account of the anodic polarization oxygen ions discharge and remove the H_2, forming water according to the equation

$$2H_2 + O_2 = 2H_2O$$

Since the hydrogen has now disappeared, metal will again pass into solution to get into equilibrium with the hydrogen as described above.

If there is any reason why this cannot take place the metal will no longer dissolve, and the metal is then in the passive state. Now, we know that the formation of water from $2H_2$ and O_2 is very slow if there is no catalytic agent present. Therefore, every metal which has not an accelerating catalytic effect on the formation of water must assume the passive state with anodic polarization. Instead of the solution of metal we then get liberation of oxygen.

If we, therefore, arrange the metals in a series according to their ability of assuming the passive state, and in another series according to their accelerating catalytic effect on the formation of water, then these two series must be essentially the same. The author has measured the catalytic effect of the metals in water formation by "reststrom measurements," and has found that the parallelism sketched above really exists. Passivity is, therefore, intimately connected with the reaction velocity.

Dissociation Theory.

Prof. Dutoit spoke on molecular conductivity and the laws of dissociation in organic and inorganic solvents. He referred to Prof. Kohlrausch's criticism of the dissociation theory, emphasizing the difficulties which the dissociation theory has to meet with solvents other than water. There are apparently a number of organic solvents in which, for instance, the law of Kohlrausch of the independent migration of ions does not hold true.

The author has made extended experiments with various solvents and various concentrations, and found in nearly all cases a very great influence of light on the result. If the effect of light is excluded, or numerically taken into account, he finds that solutions in solvents other than water behave exactly like aqueous solutions. The behavior is entirely analogous to that of aqueous solutions. The law of Kohlrausch of the independent migration of ions was found to hold good in all solutions which were investigated. The law of mass action (the dilution law of Ostwald) holds true in dilute solutions.

Besides the papers abstracted above a number of papers were presented which were of purely chemical interest.

Notes on Electrochemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

The Faraday Society.

A rather sparse attendance of members marked the ordinary meeting held on June 12. (A preliminary account of which was already given in our July issue, page 275.) Mr. Murray Morrison presided. The first two papers (a), "The Electrolytic Deposition of Zinc-Using Rotating Electrodes," by Dr. T. Slater Price and Mr. G. H. B. Judge, and (b) "A Simple Form of Rotating Cathode for Electrochemical Analysis," by Dr. F. Mollwo Perkin, were read in abstract by Dr. Perkin, with a running comment and a series of blackboard drawings.

In regard to the second paper, some later figures were given as to the rate of deposition on nickel cathode. With .1266 grammes of copper present in a solution, .1259 grammes were deposited in 40 minutes. With 0.875 grammes of iron present, apparently 0.895 grammes were deposited in 35 minutes, this discrepancy being due to the deposition of carbon at the high current density of 1 amp. per square decimeter. As a cathode material nickel was just as good as platinum, nitric acid successfully dissolving off a copper deposit, sulphuric acid successfully dissolving off iron deposits. Iridium anodes, although costing 150 per cent more were recommended in place of platinum.

These two papers being discussed jointly, Dr. Borns asked where the carbon came from. Dr. Perkin replied that it was due to the presence of oxalate or tartrate. Dr Walker suggested that the throwing over of water with some rotating electrodes was evidence to the effect that electrochemists had often failed to appreciate the importance of studying mechanics. He also asked whether the iron deposit mentioned by Dr. Perkin had been examined micrographically. Mr. Gaster asked whether the lubricating oil used for the vertical spindles got into the electrolyte.

Mr Spiers mentioned some experiments in which he had been engaged in the deposition of zinc for accumulator purposes. Very fine, smooth deposits of zinc were obtained on rotating cathodes running from 800 to 1,000 r p m. The results he had noted were not quantitatively accurate, but the current efficiency was over 90 per cent. The electrolyte consisted of saturated zinc sulphate, to which 7 per cent of sulphuric acid was added. In reply, Dr Perkin pointed out to Dr. Walker that it would be extremely difficult to etch cathodes with such thin deposits. There would probably be some interesting information available on this subject, as Mr Cowper-Coles was carrying out some large-scale experiments on the deposition of pure iron.

The third paper, by Mr. S. Binnings and Dr. Perkin, entitled "The Electrolysis of Solutions of Thiocyanates in Acetone and Pyridine, Part I.," had not been circulated among the members. It was perhaps on this account that Dr Perkin's interesting little lecture was followed with close attention. As to whether the electrolytic product was similar to canarin was a little obscure. In some respects the evidence was adverse, as

when boiled with pyridin the electrolytic canarin blackened. The product from the electrolysis of the acetone had a more orange color, but gave the same reaction on boiling. Using a platinum anode solution of 20 per cent of ammonium throcyanide, an e. m. f. of 50 volts was required for a current of 5 amps., but with a rise in temperature the voltage fell to 3.5, thus increasing the watt-hour efficiency. At the higher voltage the current efficiency was 25 per cent. The product being an anode one, it was at once found that the anode became coated, and the flow of current ceased. The anode was, therefore, kept clear by brushing, bristles being fixed in the revolving cathode. The curious results found with using lead anode, when lead throcyanide was, were also mentioned.

In conclusion, Dr. Perkin confessed the reason for the reading of the first part of a paper which had not yet been circulated. He and Mr. Binning had been at work on the subject for some months. They thought it well to put their present conclusions on record, because they had learned that Dr. Kahlenberg was at work on pyridin solutions, with a view to finding evidence tending to discredit the ionic hypothesis. Dr. Kahlenberg would be certain to stumble on this, and the authors wished to be smart enough to "get there" first.

A very brief discussion followed. Dr. Perkin answering Dr. Borns, told of a dog which died after receiving three doses of canarin, amounting to 12 grammes, with its food in three days. Mr. Digby asked how the current efficiency was affected by the rising temperature. The falling voltage had given an increased watt-hour efficiency; had the current efficiency remained constant? Dr. Walker retorted on the previous speaker by pointing out that the current efficiency depended upon the resultant product, and that it was not certain what the resultant product was.

After the usual votes of thanks the meeting then adjourned.

Exhibits at the Royal Society's Conversazione.

Very few of the exhibits at the Conversazione held on June 20 were of any electrical or metallurgical interest. Three, however, call for mention in these notes.

Dr. T. G. Moody exhibited specimens illustrating the indifference of oxygen towards iron in presence of water, and the effect of the admission of carbonic acid. A specimen of Swedish iron remained perfectly bright after exposure for three weeks to water and 32 liters of air feed from carbonic acid. At the end of the period named, air, cleaned by its passage through a tower packed with moist pumice stone, but containing its normal amount of carbonic acid, was admitted. After 72 hours, during which time approximately 17 liters of air passed through, considerable rusting had occurred.

Mr. R. G. Durrant exhibited tubes, diagrams and experiments to show that ionic separation occurs when solutions of acids or of salts are allowed to diffuse into sensitized jellies or solutions. Acids when they diffuse into blue litmus jellies produce a purple zone in advance of the diffusion front, and (where the acid has a bleaching union) a yellow-red zone behind the diffusion front. An acid mixed with litmus on diffusing, leaves the litmus behind. When concentrated silver nitrate solution diffuses into purple litmus, under suitable circumstances, red, blue, colorless and gray zones—corresponding to H, OH, NO and Ag, appear in this order. At certain periods sharp bands may form, whose boundaries, if measured from the diffusion start, almost exactly correspond to known mobilities of these ions.

Dr. F. D. Chattaway exhibited copper mirrors obtained by the deposition of metallic copper on glass. The method of silvering glass by depositing the metal in a thin film by reduction of some soluble silver compound has long been employed in the production of mirrors, but, hitherto, no method of similarly depositing copper in a brilliant film has been discovered. The exhibit showed a number of glass vessels on which copper had been deposited by a low reduction of the black oxide.

Market Prices During June.

The chief feature of the month has been the lower prices and unsteady condition of the copper and tin markets. From the 1st until the 13th copper was exceptionally firm and steady at about £86 per ton. By the 19th the price had receded to £82.5.0, and although there was a recovery on the 21st to £83.17.6, a further decline followed, with the result that £81.5.0 was reached by the 26th, and closed on the 30th at £81.7.6. Tin opened in the neighborhood of £180, and continuing to fall reached £175 on the 14th; prices rallied on the 22d to £180.10.0, subsequently declining to £177 for cash. English lead has been fairly steady, large quantities changing hands, the prices varying between £16.15.0 and £17.5.0 per ton. Cleveland pig iron rose from £2.9.6 on the 1st to £2.11.0 on the 11th, being slightly irregular subsequently. On the 30th the price closed at £2.10.0. Antimony is quoted at £120 per ton, and quicksilver at £7.1.0 to £7.5.0 per 75-pound flask.

Shellac is quoted at £10.5.0 per case, and the Calcutta corner seems a strong one. Ammonia sulphate fetched £11.16.3 on the 30th. The coal tar derivatives and alkaline products are practically unchanged.

London, July 4, 1906.

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS

Electric Furnaces.

Induction Furnace.—T. F. Snyder, 825,359, July 10. Application filed July 15, 1904.

The construction of the induction furnace is shown in Fig. 1. The core *a*, which is made of iron laminations or strips clamped together at the corners, is arranged to pass horizontally through the wall *b* of the furnace. This core is in the shape of a rectangle, and the two horizontal arms which pass through the furnace are provided with a series of water-troughs *c* set at intervals between the laminations. The water is passed through for cooling purposes, *d* being the inlet pipe and *e* the discharge pipe. Primary coils *f* are wound upon the core *a* on either side of the furnace, and are electrically connected with the generator *e*. The furnace is built up around the magnetic core in such a way as to provide a melting chamber *g*, which is provided with a reverberatory roof *g'* having charging openings g^2g^2 and a vent *h* for the escape of gases. A bridge *k* is built across the central portion of the furnace from wall to wall, immediately surrounding the horizontal core *a*, and dividing the vertical chamber into two vertical wells on either side of the bridge. A number of bars of copper *m* are set in the furnace across the bridge. When molten material is introduced into the chamber *g* above the level of the bridge, a complete secondary circuit is thus formed around the core *a* and in inductive relation to it. The vertical wells on either side of the bridge *k* are filled with molten metal, slightly below the top of the bridge and forming molten "electrodes" united at the bottom by the copper bars *m*. Slag is provided in the chamber resting on the surface of the molten metal and filling the chamber to a level above the surface of the bridge, so as to close the electrical circuit. The material to be smelted is fed through the openings g^3g^3 upon the surface of the molten slag, and is gradually reduced; the metal which is liberated sinking into the wells on either side of the bridge. The tap holes *i*

and *o* for the slag and the metal also serve as a means for regulating the resistance, since when the level of the molten material above the bridge *k* is lowered the cross-section of the secondary circuit is reduced at this point, thus increasing the resistance. The secondary circuit, which is arranged in a vertical plane, as shown, is composed of conducting materials having different specific gravity—that is to say, the heavier material at the bottom should be a good conductor, while the lighter material at the top should be of comparatively high resistance. Since the heating effect of an electric current increases in proportion to the resistance of the conductor through which the current is passed, it will be seen the maximum heating effect of the current in the secondary circuit will be developed at the top—that is to say, in the slag upon which the ore to be smelted is fed. The metal of the molten electrodes is a comparatively good conductor, but not so good as the copper bars at the bottom, so that those electrodes will be heated to a sufficient extent to keep them molten. In the smelting of iron the copper at the bottom may be molten if desired. It will remain at the bottom by virtue of its specific gravity. The slag, on the other hand, will always be light enough to remain at the top, the ore to be smelted in turn floating upon the surface of the slag.

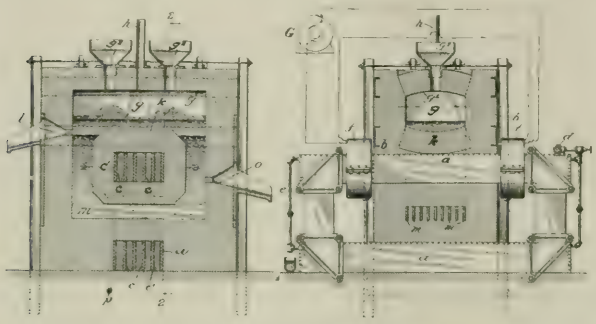


FIG. 1.—ELECTRIC INDUCTION FURNACE.

Low-Carbon Ferro-Alloys.—E. F. Price, 825,348, July 10. Application filed Nov. 12, 1905.

The process depends on the possibility of producing a high percentage ferro-silicon, low in carbon, and on the subsequent use of this ferro-silicon as a reducing agent. The process is a continuous operation comprising two stages. In the first stage, ferro-silicon high in silicon and low in carbon is produced by electrically smelting a charge of silica, iron ore (or iron) and carbon. The molten silicide is tapped from the smelting furnace and allowed to solidify. The ingot is then broken into fragments, which are mixed with a granular body of the compound to be reduced—for example, chromite—and the mixture is melted in an electric arc furnace. A basic flux, such as lime, is mixed with the charge to convert the silica produced by the reduction of oxide ores into a fusible slag. The furnace is operated continuously, the metal and slag being drawn off through tapping holes at different heights and new charge is added to the furnace.

Calcium Carbide.—E. F. Price, 820,742, 820,743, 820,744, 820,745, July 24. Application filed Oct. 13, 16, 1903, and April 4, 1904. (Assigned to Union Carbide Co.)

The charge, which is a mixture of finely divided lime and coke, is showered downwardly through a preheating chamber in which it is subjected to the action of a flame and heated gaseous product of combustion. It is then collected or massed into a body in an electric furnace, and heated to the required temperature, either by passing an electric current through the heated charge or the molten product acting as a "resistance conductor," or through a separate resistance conductor in proximity to the charge.

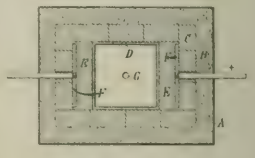
The preheating is effected by withdrawing the waste carbon monoxide from the electric furnace and burning it in the preheating chamber. Other fuel, such as natural gas or oil, may be employed as a substitute for or adjunct to the carbon monoxide.

Electric Furnace.—John F. Hammond, 825,386, July 10. Application filed Sept. 18, 1905.

The furnace comprises a muffle, in the interior walls of which heating coils are variously disposed and connected, so as to form a plurality of groups. By means of a special switch or commutating device the electric current can be made to energize the heating coils in various combinations. For instance, first all coils may be in series, then the different coils may be connected in different series-parallel connections, and finally all groups are in parallel. The object is to have full control of the temperature, and especially to accomplish that the increase of temperature of the furnace is very gradual. The construction of the commutating device is described in detail with diagrams of the different methods of connection.

Electric Dental Furnace.—H. Platschick, 820,962, July 24. Application filed Nov. 27, 1905.

Details of construction of an electric muffle furnace, in which the muffle is easily detachable from its casing, from the electric wires and also from the prometer. "More paritcularly, the invention consists in combining in the muffle itself with the thermo-electric couple, which is usually constituted by different alloys based on platina, suitable resistances so adjusted as to compensate for the differences which may exist between the electric and thermic constants of divers couples of the same make by reason of the irregularity of the alloys. This arrangement allows of delivering over to the trade muffles which are interchangeable—that is, all the couples present between their terminals the same thermo-electric constants, and which consequently all give in connection with one and the same galvanometer the same pyrometric indications when they are subjected to equal temperatures. The necessity of a fresh standardizing or of a particular adjustment when a muffle has to be replaced is thus avoided, the result being a great saving of time and money."



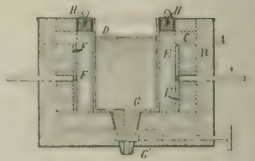


FIG. 2.—RESISTANCE FURNACE.

Resistance Furnace.—I. L. Roberts, 821,830, May 29, 1906. Application filed May 31, 1904.

The construction is shown in Fig. 2, the upper and lower diagrams being a plan and vertical section respectively. A is an iron jacket, B a heat-insulating material, like a mixture of asbestos and magnesium carbonate, ground together and well packed in place. The chamber C is built up of refractory material, like fire-brick. Within the same is supported the crucible D of conductivity material, like iron. Between the walls of the chamber and the vessel is a granulated resistor, like granulated or powdered coke, mixed, or not, with sand or magnesia or lime. This is in contact with the terminal or "electrode" plates F F. In a modification of this construction, the crucible rests on the layer of granulated resistance material, and the crucible itself forms the other terminal.

Rotary Electric Furnace.—W. McA. Johnson, 825,058, July 3, 1906. Application filed Oct. 13, 1903.

This rotary furnace is particularly designed for the reduction of zinc ores and the distillation of the metal. The construction is shown in Fig. 3. The furnace consists of a plurality of transverse sections 1 of cast iron and cast iron end plates 2. Each section 1 is provided with a lining 3 of refractory brick. In assembling the furnace, gaskets or plates 5 of asbestos or other refractory insulating material are clamped between the several sections and between the end sections and the end plates 2. In this manner an air-tight structure is provided, composed of several independent and electrically insu-

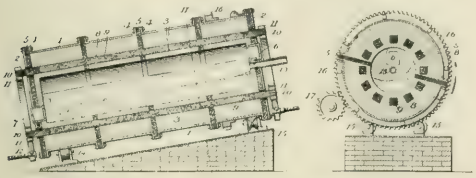


FIG. 3.—ROTARY ELECTRIC FURNACE FOR ZINC DISTILLATION.

lated parts, whereby loss of current by leakage through the furnace walls between the terminals is reduced to a minimum. A plurality of longitudinal passages 8 extend through the refractory lining. They are filled with powdered coke or graphite, which acts as resistor, terminal connections being provided in form of the graphite plugs 10. The zinc vapors are led off through pipe 13 to condensers.

Calcium Carbide Furnace.—J. E. Hewes, 821,574, May 15, 1906. Application filed July 18, 1905.

The charge of lime and carbon is placed in trough-shaped hearths, and subjected to the action of an arc between two electrodes arranged at an angle to each other. The hearth is slowly moved past the electrodes, so as to remove the formed carbide from the zone of fusion immediately after it is formed. The process is thereby rendered continuous. For this purpose the trough-shaped hearth plates are placed on two endless traveling chains or carriers. As the chains are moved along, additional hearth plates are added from time to time to the feeding end, and corresponding plates with the product are removed from the distant end of the series. This forms the traveling hearth. The two electrodes are at a right angle, one being vertical and the other horizontal, the vertical electrode being at a higher elevation than the horizontal electrode. In starting the furnace, the vertical electrode is lowered to the horizontal electrode, so as to close the circuit. The horizontal electrode is then drawn slowly backward, so as to approximate 8 inches from the vertical electrode, and then the vertical electrode is slowly raised. The height to which it is raised (together with the speed of the traveling hearth, determines the thickness of the carbide formed.

Calcium Carbide.—John M. Morehead, 825,234, July 3. Application filed March 24, 1905.

This patent refers to the process of manufacturing carbide in the electric furnace, described and illustrated in our Vol. III., page 153. In the first claim, the inventor claims "the process of producing calcium carbide, which consists in establishing a region of reduction under a gas-tight hood, feeding the charge into and downward within the hood into the region of reduction, sealing the hood by a layer of the charge material extending below the lower end of the hood, and withdrawing the waste gases from the hood."

Electrolytic Processes.

Electrolytic Production of Nitrogen Compounds.—J. W. Wood, 826,301, July 17. Application filed Feb. 21, 1902.

The inventor proposes the following peculiar method of producing nitrogen compounds from the nitrogen of the air. By means of a nozzle worked with a blower, air bubbles are forced through an electrolyte of dilute nitric acid or sulphuric acid, and serve to aerate the solution thoroughly, and thus charge it with more or less nitrogen. If now electric current is passed through the cell oxygen is liberated at the anode, which is said to unite with the nitrogen and form nitric acid, which is syphoned off. The hydrogen set free at the cathode is also claimed to unite with the nitrogen present, forming hydrate of ammonia. "The nitrogen which is thus constantly removed from the liquid is replenished by the nitrogen of the air, as said air passes in the form of bubbles upward through the liquid."

Support for Anodes.—Frank Engelhard, Fred. H. Engelhard and William A. Engelhard, 825,591, July 10. Application filed Sept. 14, 1905.

The object is to provide a device for supporting anodes in electro-plating solutions so that cruder forms of anodes than heretofore can be used, such as the tops or unused portions of old anodes, ingots of any form, etc. These pieces are pierced from side to side and are then slipped on to either a straight or a curved support. The essential feature is that this straight or curved support consists of a strong inner core or rod of brass, which is entirely surrounded in a lead envelope. Details of construction are described and illustrated.

Treatment of Nickel-Copper Matte.—W. McA. Johnson, 825,056, July 3. Application filed Sept. 30, 1903.

Mr. Johnson proposes here a new solution of the old problem of treating nickel-copper matte. (See, for instance, the processes of D. H. Browne, our Vol. I., p. 381, and N. V. Hybinette, our Vol. IV., p. 33.) The copper-nickel matte may contain 39 per cent of nickel, 39 per cent of copper, 1 per cent each of iron and cobalt, and 20 per cent of sulfur, together with small proportions of platinum and palladium. This matte is crushed, ground to 80-mesh, placed in leaching vats, provided with agitators, and subjected therein to the action of a heated solution of hydrochloric acid of about 10 per cent concentration, at a temperature approximating 100° C. Means are provided for conducting away the sulfureted hydrogen formed by reaction of the acid upon the sulfids of the matte. The solution thus obtained contains nickel, cobalt and iron as chlorides. It is freed from cobalt and iron by treatment with chlorine and sodium carbonate or by the action of hypochlorites, according to known methods. The cobalt is recovered from the precipitate as oxide (Co_3O_4), and may be sold as such. The residual nickel-chloride solution is concentrated, if necessary, brought to neutrality or to a faintly acid reaction, and electrolyzed with insoluble anodes, the solution being maintained during the electrolysis at a temperature of about 65° C. Nickel is thereby precipitated in reguline form, and the evolved chlorine is utilized for the manufacture of bleaching powder or otherwise. The residue from the acid treatment may contain 75 per cent of copper, 5 per cent of nickel, and some 20 per cent of sulfur. This residue is subjected to an oxidizing fusion in a reverberatory furnace, and thereby converted into blister copper, the nickel being slagged off and retreated in a small blast furnace. The blister copper is poured hot into refining-furnaces, and therein brought to "set" copper, carrying 0.3 to 1 per cent of nickel. This is then cast into anodes and electrolytically refined to produce cathode copper of 99.94 per cent purity, the precious metal slimes being treated in any approved manner. The nickel salts accumulating in the acid electrolyte are periodically separated therefrom and partially purified from copper by repeated fractional crystallization, the last traces of copper being separated by electrolysis and the nickel salts crystallized for platers' use.

Electrolytic Production of Metals of the Earthy Alkalies.—C. Suter and B. Redlich, 813,532, Feb. 27, 1906. Application filed Jan. 2, 1904.

"A process for the electrolytic production of metals of the earthy alkalies, consisting in melting the salt, effecting the

electrolysis thereof, and causing the slow movement of the cathode from the surface of the electrolyte during the electrolysis." The metal produced at the cathode (for instance, calcium) is thus continually removed from the melt, it becomes gradually solid, and then assumes in its turn the function of the cathode. At the same time the solidified metal becomes covered by adhesion with a thin coating of the electrolyte, whereby it is protected from all oxidation by the air. (This is the method by which calcium is now being made on a commercial scale in Germany. See our Vol. II., p. 270.)

Extracting Gold, Silver, Etc.—J. A. Comer, 813,620, Feb. 27, 1906. Application filed Jan. 2, 1904.

The crushed ore is subjected to a solution of potassium cyanide. The solution is drawn off or decanted from the pulp. The sedimentary deposits suspended in the solution are then separated from it by impregnating the solution with compressed air, rendering the sediments fluffy and flocculent and filtering. The precious metals are then deposited from the solution by electrolysis.

Extracting Metals from Ores.—A. Lénárt, Jr., 826,435. July 17, 1906. Application filed Aug. 18, 1905.

The object is to extract metals from ores by means of the acid radicals evolved in the electrolysis of alkali salts, particularly chlorides. The cathodes are surrounded by diaphragms, to permit removal of the cathodic product of electrolysis. Cathodes and anodes are arranged in direct proximity to each other, so as to reduce the resistance. To make this possible, the ore is not made the anode, and the chamber in which the electrolysis is carried out is arranged beneath the ore chamber in which extraction takes place, and is separated therefrom by a permeable floor. It is stated to be of importance that the "flowing electrolyte should first pass over the anodes and only afterwards over the diaphragms containing the cathodes, so that the electrolyte is already so far saturated with the acid anions before reaching the diaphragms that the lye diffused through the latter cannot precipitate the metals as hydrate or oxychloride from the electrolyte, which during the preceding circulation through the extraction apparatus has absorbed metallic salts."

Manufacture of Sulphuric Acid.—W. McA. Johnson, 825,057. July 3. Application filed Oct. 10, 1903.

Sulfur dioxide, or sulphurous acid, is oxidized in an electrolytic tank provided with anodes of lead, the electrodes being arranged according to the series system—that is to say, in such manner that the intermediate or unconnected plates act as bipolar electrodes. Under these conditions the sulphur dioxide dissolved in the bath is oxidized at the anodes to sulfuric acid. For the cathode surfaces it is desirable to use a metal having a low cathodic potential, whereby the cathodic separation of sulfur is reduced to a minimum. For this purpose copper electro deposited upon the cathode surfaces is preferred. In practice the procedure is as follows: Sulfuric-acid solution of about 20 per cent concentration is saturated with sulfur dioxide by means of a coke-tower or other suitable device. This solution is then caused to flow through the electrolytic cell and past the electrodes therein, whereby the sulfur dioxide is oxidized to sulfuric acid, and a solution is obtained which may approach 30 per cent concentration. A certain proportion of this acid is withdrawn and concentrated or utilized in any desired manner, while the remaining portion, diluted with water to 20 per cent and again saturated with sulfur dioxide, is returned to the electrolytic cell, this cycle of operation being repeated indefinitely.

Cell for Sodium Chloride Electrolysis.—G. A. Gabriel, 822,109. May 29, 1906. Application filed Feb. 26, 1906. (Assigned to Bleach & Caustic Process Co.)

The cell is a modification of the Hargreaves type. The special feature is the means for leading the liquid, which percolates through the porous diaphragm, away from the diaphragm, thus obviating the danger of back percolation. The method is indicated in the left-hand diagram of Fig. 4, where 13 is the porous diaphragm, and 11 a forminous screen forming the cathode proper. A series of downwardly-inclined wires 15 is provided, the upper ends of which are twisted around the strands forming the screen 11. These wires 15 form conduits, which, owing to the adhesion of the liquid percolating through the diaphragm, lead the liquid away from the diagram and the screen itself, so that the liquid drops from the ends of the wires or is conducted to the "drain-plate" 16. The holes 22 permit the liquid to flow over both sides of the plate, and therefore increase its draining capacity. The liquid collects on the bottom of the cathode chamber, from where it is removed. For making the diaphragm the apparatus, shown in the right-hand diagram, is used. A rectangular frame 30 has attached to it on its four sides a series of coiled springs 31 with hooks 32, which pass through and engage the diaphragm 13 to be

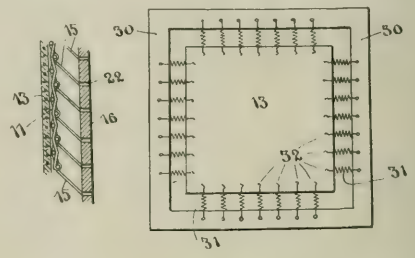


FIG. 4.—DETAILS OF CONSTRUCTION OF SODIUM CHLORIDE ELECTROLYTIC CELL.

treated. The fabric when thus stretched in the frame is moistened and allowed to dry under the tension of the springs 31. When once dried, the fabric is again dampened, and is then covered by a coating of cement and water, or clay, and silicate of soda, etc.; this mixture is rubbed thoroughly into the interstices of the fabric, and the surfaces made smooth and even on both sides. The fabric and cementitious material is then allowed to set and harden to the consistency in which it is to be used, after which the screen is removed from the frame and placed in a mold to give it the requisite shape.

Bleaching Liquor.—F. L. Bartelt, 813,688, Feb. 27, 1906. Application filed Oct. 7, 1904.

The object is to provide a convenient and automatic arrangement for the production of bleaching liquor in small steam laundries. On a raised platform a belt-driven dynamo (or a converter, etc.) is placed together with the "solution tank." This is provided with an indicator by which the height of the solution in the tank may be seen, and by which the filling of the tank may be regulated. From the bottom of this tank connection is made through several independent pipes with the "electrolyzer," which is placed at a lower level. The bleaching liquor produced flows off into the "liquor tank," which is at still lower level. In the solution tank is provided a float, adapted under certain circumstances to close one or more circuits. Thus it may close a bell circuit and an electrolyzing circuit. When the level of the solution in the tank descends below a certain point, the float closes both these circuits, so that the bell is rung to call the attention of the attendant, and at the same time the switch is operated to cut off the current to the electrolyzer, as otherwise the current would produce a liquor detrimental to the clothes.

Electrode for Production of Bleaching Liquors.—R. Kother, 806,413, Dec. 5, 1905. Application filed Sept. 7, 1905.

A bipolar electrode is constructed as follows: A thin platinum wire netting or platinum foil is placed across a glass or hard rubber plate, so as to cover entirely the anode side. But the platinum is also folded round the edge of the glass plate

and a graphite plate which forms the cathode is pressed against the turned edges of the platinum so as to make good contact.

Treating Organic Substances in Presence of Vanadium Compounds.—E. Oppermann, 823,435 June 12, 1906. Application filed Aug. 31, 1904. (Assigned to Meister, Lucius & Bruening.)

The inventor makes use of the great oxidizing power of the higher oxides of vanadium and the energetic reducing power of the lower oxides of vanadium. Vanadium compounds in an electrolytic bath are excellent depolarizers, both at the anode and cathode. The rapidity of carrying oxygen and nitrogen to the "acceptors" being very great (indigo is reduced instantaneously, and one part of vanadium salt oxidizes 1,000 parts of aniline to aniline-black in presence of oxidizing agents), only small quantities of vanadium salt need be used for reducing or oxidizing by electrolysis. The various oxides of vanadium produced in the electrolytic bath have a more or less strong reducing or oxidizing power. As an example, the oxidation of anthracene may be described: Into a vessel lined with lead, serving as anode, is introduced sulphuric acid of 20 per cent strength, with a quantity of vanadic acid dissolved in it, for instance, 1 per cent. The cathode may be of lead. The anthracene to be oxidized is introduced into the vessel as a fine powder while actively stirring, and the mixture is heated to about 80° C. With a current density of 300 amps. per square meter the voltage is 1.6. After a short time anthraquinone may be proved. If somewhat more ampere-hours than the theoretical figure have passed the cell, the anthraquinone need only be filtered, and the electrolyte may be used for another operation. Other examples dealt with in the specifications are the oxidation of aniline to quinone (hydroquinone), the reduction of azobenzene, and the reduction of azoxybenzene.

Electrolytic Production of Hydrogen and Oxygen.—L. van Scoyoc, 813,844, Feb. 27, 1906. Application filed May 15, 1905.

Details of construction of an electrolytic apparatus for the production of hydrogen and oxygen, in which the operation is rendered continuous by the use of automatic float-valves. The level of the acidulated water in the electrolyzer is maintained constant by means of a float-valve in the supply pipe. The two electrodes are placed in two compartments which, of course, are open at the bottom. Each compartment is divided into a lower and an upper chamber, connection between the two being made by automatic float-valves. When the pressure of the gases generated in the lower chambers becomes great enough to lower the level of the water, the valve is opened and the gases pass into the other chamber and gas-bags.

Electrolytically Preparing Metals for Lithographic Purposes.—O. C. Strecker, 810,889, Jan. 23, 1906. Application filed April 19, 1900.

Metal plates for printing purposes are generally prepared by coating them with a layer of hygroscopic materials directly or by brushing, powdering, etc., in a mechanical way or by coating the plates with materials capable of forming a hygroscopic layer by chemical reaction with the metal of the plate. The present inventor employs an electrolytic process for the same purpose, the plate being used as electrode before or after transferring the drawing. The procedure is as follows: A zinc plate is ground by means of a pad of steel turnings and pumice powder, covered with soft leather. The metal is then rinsed with water and dried. On this plate a lithographic design or transfer, the negative of the intended print, is fixed. After this the plate is gummed with a solution of gum-arabic of medium strength and dried. After being perfectly dry, the greasy substance is washed out by lithophine (a solution chiefly consisting of asphaltum in benzene or spirits of turpentine). The excess is wiped off and the plate dried. Then the gum-arabic is washed off the plate by water, and the negative is left on the plate as a layer of asphaltum. The negative is properly inked by means of the roller, and then the plate is covered with the spots. Alterations and corrections are made with lithographic ink in the well known manner, and the plate is now ready for electrolytic treatment. It is used as anode in a 2.5 per cent sodium fluoride solution, with a current density of 0.1 to 0.5 amp. per square foot, for 2 to 5 minutes. The reaction is $3Zn + 6NaF + 6H_2O = 3ZnF_2 + 6NaOH + 6H$. The alkali formed has to be neutralized before using the electrolyte again. The whole plate may either be suspended vertically in the solution ("dipping method"), or it may be placed horizontally, solution poured on it and a cotton-covered metal plate, forming the cathode, passed over it ("sponging method"). For further details reference must be had to the specification.

Electroplating.—S. H. Thurston, 822,873, June 5, 1906. Application filed April 1, 1905.

The object is to make the electrolytically deposited metal stick to the surface on which it is plated. For this purpose this surface is first treated mechanically. The surface receives an "amorphous, inherent, adherent, coherent and permanent coating" of the desired metal by heating particles of this metal against and into the surface. For this purpose the inventor uses a beating machine, in which "beating rods" of wire (of the desired metal) are suspended from a revolving axle and are beaten against the surface. The rough coating thereby produced on the surface serves as an "anchorage" for the electrolytic deposit which is applied afterwards. Among other applications the inventor states that this method enables one to plate on aluminium.

Sterilizing Milk.—R. C. Turner, 806,600 and 806,601, Dec. 5, 1905. Applications filed Jan. 25, 1904, and March 16, 1904.

The milk or other liquid, to be sterilized, is passed through a series of vessels, flowing in a stream from one vessel to the next. Simultaneously an electric current is passed from the bottom of one vessel upward through the body of milk, then downward through the stream, falling from this vessel to the next, etc. For this purpose an electrode is placed on the bottom of each vessel. The electrodes may be connected to the electricity supply in different ways. For instance, the electrodes in the first, third, fifth vessels may be connected to the positive pole, those in the second, fourth, sixth vessels to the negative pole.

Purification of Water.—L. Dion, 820,482, May 15. Application filed May 24, 1904.

Mechanical details of construction of a cell in which water is to be purified by being passed between series of electrodes.

Batteries

Converting Spongy Lead Into Sulphate.—C. J. Reed, July 17. Application filed May 7, 1903. (Assigned to Security Investment Co.)

To convert a spongy lead plate into lead sulphate without using an external current, the inventor forms a short-circuited couple, consisting of the lead plate and another plate provided with a coating of platinum black, in a dilute solution of sulphuric acid.

Storage Battery Plate.—A. E. Knight, 826,173, July 17. Application filed Nov. 8, 1905.

The electrode is composed of two grids, each of which consists of a thin sheet of lead with punctures distributed over it and with ribs attached to one face, which form compartments into which the active material is firmly packed. The grids are secured together with the perforated sheets on the outside, and the active material in the chambers of one grid is separated from the active material in the chambers of the other grid by a thin film of lead foil, which is firmly clamped between the grids to make a good electrical contact therewith, whereby the conductivity of the electrode is increased. The film of lead foil is provided with very fine perforations.

Separator.—I. W. Horton, 825,837, July 10. Application filed March 17, 1904.

The separator comprises two parts arranged one to [illegible]

the adjacent faces being provided with longitudinal and transverse grooves, so as to provide intercommunicating acid wells within the bounding surfaces of the separator. The object is to have as large a portion of the active material as possible in contact with the electrolyte.

Battery.—C. W. Harper, 825,882, July 10. Application filed April 8, 1904.

Details of construction of a pocket battery to be used in connection with a portable telephone system for deaf persons.

Storage Battery.—G. A. Ford, 824,348, June 26, 1906. Application filed July 20, 1904.

The battery is built up of trays or pans in a substantially horizontal position, above one another. The trays are made of gutta-percha, and have a perforated bottom. The perforations are plugged with a porous material. Perfect circulation of the electrolyte is obtained by means of special openings. Within each pan an electrode is provided of peculiar construction. There is a central web with vanes between it, and the edges of the vanes lie at an angle to the plane of the plate. An auxiliary electrode-plate has a web coextensive with the other web, and is provided with fins projecting into the spaces between the vanes.

Storage Battery.—H. H. Porter, 813,52, Feb. 27, 1906. Application filed Sept. 21, 1903.

The patent refers to the construction of a separator between the plates. It consists of a thin layer of wood with parallel rectangular vertical grooves arranged alternately on its opposite sides, and, between them, relatively wide flat spaces, each such space being directly opposite a groove on the other side.

Storage Battery Plate.—William L. Silvey, 824,828, July 3. Application filed March 5, 1906.

The construction is indicated by the first claim: "In a secondary battery a perforated battery-plate, consisting of two lead plates having holes in them larger on the inside, perforated metal cups fitted into the holes in the perforated lead plates, the perforated lead plates fastened together forming perforated cavities for active material, and connecting ears on the plate for connecting it to an external circuit."

Storage Battery.—A. Mueller, 813,730, Feb. 27, 1906. Application filed July 30, 1904.

The battery electrode consists of an envelope, comprising two perforated covers or two sheets of metal gauze, within which are contained cakes of active material. At their margins these covers are held together by U-shaped reinforcing strips. At suitable intervals throughout the covers there are loops through which pins are inserted, which firmly bind the two covers together and stiffen the plate.

Miscellaneous

Ozone Ventilator. F. de Marc, 820,056, May 15 1906. Application filed Sept. 23, 1904.

The patent relates to a centrifugal fan, and the special feature of the device is that it sucks in, ozonizes and delivers the air in a single operation and in the same apparatus. A porcelain or glass casing of the usual form of the shell of centrifugal fans is divided into two identical halves by means of a vertical glass plate in the center, perpendicular to the axle of the machine. The axle carries two metal sockets insulated from each other, one in one section, the other in the other section of the machine. Each metal socket carries a series of aluminium wings which are electrically connected to a source of high-tension currents. Briefly, the device is an electrifier and pump for the gas it electrifies.

Locating Metals Beneath the Earth's Surface.—E. R. Wolcott, 822,175, May 29. Application filed Sept. 5, 1905.

"This invention consists of a process of locating and also extracting metals beneath the earth's surface. The process consists in placing electrodes in the ground at a suitable distance from each other, moistening the earth between the electrodes, if necessary saturating the earth with acids adapted to dissolve the metals contained in the earth, and finally passing a current of electricity from the anode to the cathode. In this event the dissolved metals will be carried along with the electrical current and deposited at or upon the cathode." By changing the position of the electrodes, the location of the metal or ore in the earth is determined. The electrodes are enclosed in porous cups filled with electrolyte; the anode consists of carbon, the cathode of platinum. Nothing is said how the scheme works in the field and not on paper.

RECENT METALLURGICAL PATENTS.

Lead and Silver.

Desilverizing Lead Bullion.—In the old Parkes process the procedure is generally as follows: After the bullion has been freed of its copper, antimony and arsenic in a "softening" furnace, metallic zinc is added and stirred therein, to cause the contained gold and silver, for which it has an affinity, to rise to the surface in the form of a scum containing more or less lead. This scum or zinc skin is skimmed off from the liquid lead by means of perforated ladles, and undergoes further treatment by retorting of the zinc and cupelling to eliminate the lead, and thus obtain silver and gold. According to A. Raht (826,114, July 17) a small quantity of sal-ammoniac, when added shortly after the zinc has been introduced and thoroughly stirred with the molten metals, effects a more complete separation of the lead from the scum than is possible by mechanical means alone. This supplementary step in the smelting process not only thus reduces the bulk of scum to be operated upon, but likewise causes it to be in a "drier" state; *i. e.*, containing less lead, and consequently more easily and cheaply treated in the following refining process. The cost is said to be slight, 1 pound of sal-ammoniac being required per 10 tons of bullion.

Nickel.

Mond Process.—For the production of nickel from "nickel carbonyl" (our Vol. II., p. 291), the vapor of the latter may be passed through nickel shot or pellets kept at a temperature of about 200° C. The nickel deposits from the vapor on the shot, which gradually increase in size. Carl Langer, of the Mond Nickel Co. (825,844, July 10), points out that for the proper working of the apparatus it is essential that the pellets should be as nearly as possible at the same temperature at all parts of the cylinder containing them. If this is not the case, the nickel carbonyl vapor may pass among the colder pellets without being decomposed, and nickel will be deposited on the hot surface of the cylinder. Moreover, the hotter pellets may decompose the carbon monoxide, causing a deposit of carbon on the nickel. There is also the danger that undecomposed vapor may arrive at the outlets, and, becoming heated in the neighborhood thereof, blocks them. To control the temperature better, Langer heats the cylinder by a number of gas flames, evenly distributed around the outside of the cylinder. The latter is provided with projections or ribs, to form, with the outer casing, chambers to contain the gas-jets. The part of the cylinder where the escape of gases occurs is formed as an annular chamber through which a cooling medium is circulated, the gas-escape pipe passing through this chamber.

Treating Nickel and Copper Matte.—A patent of J. A. Gilman (826,099, July 17) relates to a modification of Ins and Everett's process, whereby the nickel and copper matte while in the molten state is finely sub-divided into particles by impingement against a mechanical surface, like a cooled screen, the stream of particles being then subjected to the action of the oxygen of the air, so as to remove the sulphur from the particles. In the new process the descending particles are met by a lateral air blast, and thereby deflected, oxidized and desulphurized. The object is to decrease the height from which the

particles must be projected and allowed. An account of a new process of W. McA. Johnson for separating copper and nickel may be found in the Analysis of Current Electrochemical Patents in this issue.

Copper.

Copper Castings.—J. A. Yunck (825,100, July 3) proposes to deoxidize copper by means of nascent sodium vapor, to prepare it for the casting of sound and flawless ingots. He prepares a mixture of 30 parts of powdered carbon, 5 of coarser or granular carbon, 5 of carbonate of lime, and 60 of sodium carbonate, thoroughly mixes in water until the sodium carbonate dissolves, then evaporates to dryness and crushes the residue. This mixture is mixed with copper in the proportion of 1 to 10 parts, and is fused in a crucible sealed except for a small vent. When the temperature is raised to or above the melting point of copper, the sodium carbonate and carbon combine, forming sodium vapor and carbonic acid. The sodium combines with the oxygen of the oxides in the metal, and sodium oxide raises to the surface of the bath and forms a crust. The carbonate of lime in the mixture is intended to act as a retarder, to give the sodium vapor time to do its work.

Tin.

Detinning Tin Scrap.—On account of the comparative scarcity of tin ore, and of the immense quantities of tin used for tinning sheet iron, the electrolytic detinning of tin scrap has proven quite economical in recent years. In this development the firm of Theodor Goldschmidt, in Germany, has performed decisive pioneer work. It is noteworthy that in spite of their flourishing electrolytic works this firm has experimented in recent years on a purely chemical process of detinning the scrap, as is shown by a patent recently granted (822,115, May 29) to one partner of the firm, Dr. Karl Goldschmidt, and to Mr. Josef Weber. The process consists in subjecting tin scrap in a compressed state to the action of chlorine in combination with an "inert anhydrous fluid," in which the chlorine gas is dissolved. By the term "inert anhydrous fluid" an anhydrous fluid is meant which does not attack the scrap or chlorine or the products thereof during the process of detinning. Tetrachloride of tin is stated to be suitable for this purpose.

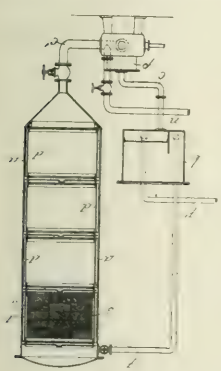


FIG. 1.—DETINNING PLANT.

The process seems to be extremely simple in operation, only few precautions being necessary. It is important to avoid any great increase of the temperature during the process. Further, since the tin scrap is used in firmly compressed bundles, care must be taken that the chlorine enters even into the narrowest and most firmly closed spaces. This can be accomplished by alternating the pressure in the closed vessel during the process, and it is of advantage to guide the process in such a manner that the pressure in the vessel is so augmented that at the end of the operation the over-pressure is, say, one atmosphere above the ordinary pressure.

The bundles of tin scrap may conveniently be 40 to 60 centimeters in width and length, and 8 to 14 centimeters in height, so that they weigh from 50 to 60 kg. The detinned iron bundles may then be directly used in steel works. The iron waste is obtained perfectly detinned with a fine smooth gray surface.

The apparatus is shown in Fig. 1; *a* is the iron receptacle for the compressed tin scrap bundles *s*. In order to facilitate the filling of the vessel the packets are brought in baskets *d*, and these baskets, filled with the bundles of compressed tin scrap, are then introduced into the vessel. Fluid chloride of tin, in which chlorine gas is dissolved, is drawn from the vessel through the pipe *t*. The chlorine gas enters the receptacle *b* through the pipe *t'*, and the chloride of tin comes into the receptacle *b* through the pipes *n* and *c*. The fluid chloride of tin comes in contact with the surfaces of the tin scrap, and the chlorine dissolved in the fluid chloride of tin acts upon the tin of the tin scrap, and chloride of tin is formed. The fluid chloride of tin flows down and new fluid chloride of tin, in which chlorine gas is dissolved, is led from above into the receptacle. The fluid chloride of tin is gathered at the bottom of the receptacle and is lifted by a pump *p* through the tubes *c* and *c'* into the receptacle *b* into which, by the tube *t'*, chlorine gas is led so that chlorine is dissolved in the fluid chloride of tin. This fluid chloride of tin, in which chlorine gas is dissolved, is brought anew through the tube *t* from above into the receptacle *a*, and the reaction of the chlorine begins again.

If the operation is as simple as described in the specification the process looks very promising, and is presumably cheaper than electrolytic detinning. It might represent an important outlet for electrolytic chlorine, in view of the fact that our electrolytic caustic and chlorine plants have troubles in working their chlorine into profitable products. While electrolytic detinning yields metallic iron and metallic tin as end products (the value of both metals being in general about equal), the final products of the new process are iron and tin tetrachloride. There is a good market for the latter product, which by treatment with ammonium chloride may be easily changed to pink salt, which is extensively used in cotton printing and silk dyeing. The tin tetrachloride could also be treated with water and afterwards with metallic zinc, resulting in the precipitation of metallic tin and formation of zinc chloride. See our Vol. II., p. 339.

Lixiviation Process.

Sulphatizing Ores.—E. Enke (826,925, July 24) patents the following process for the sulfatization of ores: The ores, their by-products and the like, are mixed with such quantities of acid alkali sulfates or ferrous sulfate, or mixtures of both, that the alkalies which, at the fusion, form sulfates, receive the required quantity of sulphuric acid from the added compositions of sulphuric acid. In preference sodium bisulphate is mixed with the ferrous sulphate. It is essential that the air be excluded during the melting, and that the temperature be raised to such a degree that the ferrous sulphate, which is formed at the beginning, is separated into oxide of the iron and sulphuric acid, so that the latter can combine as soon as it is formed with any alkali present in the charge and adapted to form a sulphate, so that this sulphuric acid is not lost and a considerable saving in bisulphate to be added is obtained. The temperature at which the other metallic sulphates decompose is, however, not reached, so that upon lixivation a solution is obtained which is free from iron and contains as little metallic sulphate as possible. Ferrous sulphate is employed particularly for the purpose of enriching the charge to be lixiviated so much with iron that the lixiviated residue forms a marketable product.

Treating Matte.

Removal of Iron from Matte.—To treat matte for the object of removing iron, the matte may be blown, in order to cause the iron to enter the slag. But in this method practically all the sulphur contained in the matte is turned to sulphur dioxide, which escapes into the air; further, the silica is taken from the lining of the Bessemer converter, whereby the lining is destroyed. Of course, the matte may first be roasted and then smelted with sand in a shaft furnace, in order to slag the iron oxidized by the roasting. Besides being very costly, due to the large fuel consumption, the roasted product is in a pulverulent condition and requires briquetting before smelting. Moreover, the roasting must proceed with care, in order to leave sufficient sulphur in the mass for the subsequent smelting

J. Savelsberg, of Papenburg, Germany (825,983, July 17), endeavors to avoid these objections in the following manner: The matte is finely divided and mixed with the necessary percentage of slag-forming ingredients, such as sand and lime, requisite for the subsequent slagging of the iron. This mixture is then brought onto a bed of glowing material, as coal, coke, ore, etc., in a suitable apparatus, and a blast is passed through the mixture from below. Due to the heat of the reaction, the mass gets red-hot without completely melting, so that the mass sinters together. The blowing is completed when the red-hot mass cools down, which it does very rapidly. During the blowing, practically all the sulphur, combined with the iron, has combined with nickel, copper, etc., while practically all the iron has been oxidized by the air-blast. The mixture after cooling is broken into pieces of suitable size for subsequent smelting in a shaft-furnace, which will yield a matte practically free from iron, yet containing substantially all the sulphur of the previous matte, while the slag is quite fluid, and contains only a small percentage of nickel, copper, etc., and nearly all the iron in the matte previously employed. A matte containing 50 per cent nickel, 14 per cent sulphur and 36 per cent iron, contained after such a treatment (*i. e.*, blasting and subsequent smelting) 77 per cent nickel, 20 to 21 per cent sulphur, and ½ per cent iron. Thus it will be seen that hardly any sulphur was lost, since the proportion of nickel to sulphur in the resulting matte is the same as in the matte treated.

Roasting Furnaces.

McDougall Furnace with Helical Hearth.—The well-known McDougall furnace has a number of successive hearths, from one to the other, of which the material is dropped in the process of roasting. This results in the production of much flue-dust. R. L. Lloyd and P. Thill (12,511, July 24) propose to overcome this objection by providing the McDougall furnace with a helical continuous hearth (like Fig. 1, in our Vol. III., p. 126), making as many turns of the helix as to get sufficient surface for the proper roasting of the material passing through the furnace. So far, the arrangement is undoubtedly sound. But now arises the question how to arrange the stirrer arms between the turns of the helix, for it is manifest that the operation of arms in the helical spaces by continuous or rotary motion is impossible. The inventors, therefore, provide for a rotary reciprocation of the arms and arrange for their following the pitch of the helix in their downward movement (where the arms are angularly placed 90 apart) for a little more than half a revolution. At the end of their stroke they are first raised vertically a sufficient distance to clear the body of ore on the hearth, and are then rotated backward, being depressed at the beginning of the stroke until they again engage the ore on the hearth. But if we consider that the central shaft and stirrer arms of a commercial furnace weigh several tons, it is clear that a very considerable amount of power would be required to operate the stirrer mechanism. The inventors endeavor to overcome this difficulty in a very interesting manner by the expedient of counterbalancing the rabble shaft and arms so that as these are lowered the counterweight is raised. They carry out this idea by supporting the mechanism upon hydraulic cylinders and connecting the latter by pipes of sufficient capacity. Thus as one sinks the other rises, and the amount of power required to operate the furnace will be, as far as the lifting is concerned, greatly reduced. The design is certainly interesting, but it is questionable whether the advantages of the continuous helical hearth are not outweighed by the complications of the stirring mechanism. This question must be decided in practice.

Cooling of Shaft and Raffles.—Another patent (825,326, July 10) has been granted to Frank Klepetko for a method of cooling the rabble-shaft and rabble-arms of a McDougall furnace. Shaft and arms are hollow, and within the shaft is located a feed-pipe, through which a limited quantity of water is conveyed into one compartment of the shaft, the latter being divided into a series of compartments. By evaporation the steam fills the compartment into which the water is introduced, passes then into the next rabble-arms and back through special conduits provided within the arms to the next compartment of the shaft, etc. The cooling effect is due to the superheating of the steam in its passage through shaft and arms.

Rake Construction.—A patent of G. S. Crouse (825,519, July 10) relates to mechanical details of rake construction of McDougall furnaces, the object being to provide improved means for attaching the rakes or blades to the arms, consisting essentially of improved split boxes or frames held together on the arms by the rakes or blades.

Tilting Roasting Furnace.—In the old form of "Edwards tilting" furnace—which is a reverberatory furnace of elongated form with revolving rabbles—the capacity is limited, because the width of the furnace is limited by the circle swept by the rabble-arms. Long arms are impracticable for mechanical reasons. T. Edwards (825,446, July 10) now provides several rows of rabbles in the ore chamber with intersections of each rabble-hearth area by other rabble-hearth areas. This permits to make the furnace of any width and capacity, although short-armed rabbles are employed. The whole furnace consists of two longitudinal chambers, one above the other, with a connection between them. The upper chamber is the ore chamber, as already described, while the lower chamber is sub-divided by a longitudinal partition, with compartments for the entrance and exit of highly heated gases and furnace products. Dampers control the outlets of the chambers.

Regenerative Gas Furnace.

Heat Regulating Device.—Archibald Johnston (824,723, July 3), the general superintendent of the Bethlehem Steel Co., improves the regulation of heat in a regenerative furnace as follows: Fig. 2 shows a transverse vertical section with the furnace chamber A, along each side of which extends the longitudinal regenerating gas chamber B and the regenerating air chamber C, both being connected to sources of gas and air respectively, and the connections being controlled by the usual reversing valves in the well-known manner. The new feature is that the "heating fluid" (produced by the combustion of gas and air) may be caused to change its path through the furnace chamber. For this purpose a plurality of entrance ports are provided for the gas as well as for the air. The illustration shows the two entrance ports 4 and 5 from the air chamber to the furnace chamber. In the same way upper and lower entrance ports lead from the gas chamber to the furnace chamber. If valve 10 is open and 12 closed, the air is forced through 4, if 12 is opened the air will pass upwards through 5. The air and gas supply is regulated in quantity by the lower valves (like 10), while by means of the upper valves (like 12) the air is directed to the upper or lower outlets, or both, as desired. If the material to be heated is placed on the floor of the furnace chamber, and if the gas and air are passed through the lower ports (like 4), the heating fluid will be in direct contact with the material to be heated. If the gas and air are introduced through the upper ports (like 5), the heating fluid will not come into direct contact with the material to be heated, and the latter will be heated by radiation. In this case

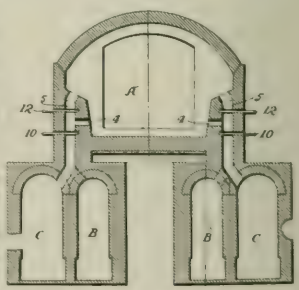


FIG. 2.—HEAT REGULATION.

oxidation is prevented and the formation of scale on the surface of the material is avoided.

Iron and Steel.

Reverberatory Steel Furnace.—V. Defays (825,522, July 10) patents details of construction of reverberatory furnaces for the production of steel, the object being the possibility of using poor gases such as blast furnace gas, and to obtain perfect control of the flame. The chief feature is the arrangement of the gas flues and air flues. At each extremity of the furnace a gas-flue opens through a suitable aperture into a combustion chamber between the apertures of two air flues, in such a manner that a sheet or layer of gas is arranged between the sheets of air, the axes of these three flues being preferably convergent, so that the jet of gas may be attacked, for example, from left to right at its upper part by one of the air jets, and from right to left at the lower part by another jet, so that it assumes a gyratory movement, insuring intimate mixture between the combustible and the carrier for oxygen. The combustion chamber constitutes a removable part independent of the metal chamber of the furnace and of each of the air and gas supply flues, which are also removable.

Attachment for Blast Furnaces.—C. Abercrombie (826,819, July 24) patents an attachment for removing the furnace-stock, carried away with the gases of blast furnaces, so that these gases are freed from any fine dust; the free discharge of the fine ore from the mouth of the furnace is taken care of so as to prevent a discharge of the fine ore at the mouth of the furnace in clouds. For this purpose the inventor provides several "primary" conducting pipes leading to the "furnace-stock catcher," which is in communication with a "furnace-stock collector." "Secondary pipes" communicate between the collector and the furnace; they serve only in cases of violent explosions or slips. The collector and catcher contain a body of water of the same level, and a gas-conducting pipe communicates with the catcher near its top. The water in the catcher and collector removes the dust and the moisture contained in the gases, and the matter which is caught in the furnace-stock catcher flows by gravity into the collector from where it can be easily removed. A spray pipe in the catcher near the top insures quick removal of the fine dust from the gases.

Iron-Ore Reduction.—G. L. Fogler (826,557, July 24) tries to reduce iron from its ore in a stack in such a manner that the reducing pressure by the action of a highly heated reducing gas made in a gas producer. Each tuyer is supplied with superheated producer gases by a pipe projecting into its tuyer end, and leaving between the gas pipe and tuyer an annular space in communication with the atmospheric air of such size as to admit under the siphoning action of the gas flow, and against the resistance offered by the charge a proportion of air sufficient to effect complete combustion. This is regulated so as to produce the greatest increase in the temperature of the gases without impairing their action in reducing the ores. The incompletely-burning gases reduce the ore, rapidly leave the reduction zone, expand and flow quickly upwards, both because of the greater porosity of the upper charge, and because of the partial vacuum which is maintained at the upper end of the furnace.

Miscellaneous.

T. Evans (826,715, July 24) patents mechanical details of construction of an air-tight blast-furnace tuyer, with automatic valve.

J. Levey (825,004, July 3) patents details of a valve for gas generators, the special object being to control the flow of hot or otherwise destructive vapors.

M. Hoopes (825,537, July 10) patents the construction of an oil furnace for heating metal blanks, the latter being automatically conducted through the furnace and thereby heated.

W. L. Painter (825,003, July 3) patents construction of a heater with twyers and a forced air blast, specially adapted for use in heating tires.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

Fixation of Atmospheric Nitrogen (Calcium Cyanamide).—A most interesting lecture on this subject was presented at the recent Congress for Applied Chemistry in Rome, by Prof. Dr. Adolph Frank, of Charlottenburg, Germany, who fifty years ago was instrumental in making the Stassfurt potassium salts available for fertilizing purposes, and who during the last ten years has devoted much time and thought to the problem of fixation of atmospheric nitrogen.

Together with Dr. N. Caro, Prof. Frank observed that barium carbide BaC_2, when heated to a high temperature, binds the nitrogen almost quantitatively, according to the equation $BaC_2 + N_2 = Ba(CN)_2$, thus yielding barium cyanide. Their first patents were taken out in 1895. The Siemens & Halske Co. then became interested in this work, and a new company, the "Cyanid" Co., was formed.

The production of cyanide appeared at that time most profitable, in view of the rapid increase of consumption in the cyanide process for gold production. But, with the Boer war and the stopping of gold mining in Transvaal, the price of potassium cyanide went down suddenly and Frank and Caro were forced to try and cheapen their method. They now took up again older experiments, tending towards the substitution of calcium carbide for barium carbide as starting material in their process.

Here they found, however, that the reaction was different, and that calcium cyanamide $CaCN_2$ was formed in preference to calcium cyanide CaC_2N_2, the former according to the equation $CaC_2 + 2N = CaCN_2 + C$. But the calcium cyanamide which is formed in this way, together with the cyanide, can be made to bind again the second atom of carbon (see last equation), by melting it with alkali salts, so that in this way pure cyanide may be produced.

More important, however, was the establishment of the fact that the raw calcium cyanamide, when heated with water under high pressure, reacted according to the following equation: $CaCN_2 + 3H_2O = CaCO_3 + 2NH_3$. This represented the solution of the old problem of making ammonia from atmospheric nitrogen.

The fact that this simple reaction takes place led the son of the author, Dr. Albert Frank, to the suggestion that the raw calcium cyanamide might be used directly as a fertilizer. Practical tests, extending over several years, have shown, indeed, that raw calcium cyanamide with a content of 20 per cent of nitrogen is fully equivalent to ammonium sulphate with the same content of nitrogen.

Of the raw materials required in the process, lime, carbon and the atmospheric nitrogen are everywhere available, but the problem of getting pure nitrogen from the atmosphere had to be solved. The author found that a satisfactory manner of way for getting pure nitrogen on a large scale is by means of

distillation of liquid air (the pure oxygen which is hereby obtained as a by-product is to be used in a later stage of the process, which will be mentioned below).

Of fundamental importance for the commercial success is, however, the supply of sufficient electrical energy at a reasonable price. This is not available now in Germany. When Prof. Angelo Menozzi, in Milan, became interested in the process, the Societa Generale per la Cianamide was formed in Rome, which acquired all the patents for making calcium cyanamide and its derivatives. This company sold the Italian and Austrian patents to the Societa Italiana per la Fabbricazione di Prodotti Azotati, which has already started a large plant in Piano d'Orte. Since in this plant the process proved satisfactory in every respect, it was decided to greatly enlarge the works by developing the large water powers of the Pescara River, which are owned by the mother firm, and also to erect a large factory in Fiume. A number of other plants will be

CALCIUM CYANAMIDE PLANT IN PIANO D'ORTE.

erected in France, Spain, Switzerland and Norway, at places where cheap water power is available.

It is hoped that in Germany cheap energy may be obtained from peat, and that the large electricity works which are operating at full load only during 20 or 25 per cent of the whole time might take up the manufacture of carbide during hours of low load.

Calcium carbide is one of the best energy accumulators. With respect to the binding of nitrogen by means of carbide, the theoretical figures are that an electric horse-power-year produces the quantity of calcium carbide required for binding 77.2 kilograms of nitrogen (corresponding to about 5,000 kg. of Chili saltpeter). In practice, however, this theoretical figure is not reached by any means, since 1 hp-year suffices only for the binding of 300 or 330 kg. of nitrogen; anyhow, this corresponds already to 2,000 kg. of Chili saltpeter, or 1,000 kg. of ammonium sulphate.

As mentioned above the reaction of calcium cyanamide with steam at high pressure proceeds almost quantitatively. It yields nitrogen in form of ammonia, and the efficiency of this process when operated on a large scale is 95 to 97 per cent.

Ammonia obtained in this way can be oxidized by means of the oxygen obtained as a by-product from the distillation of liquid air to nitric acid, and experiments which Dr. Frank and Caro have made lead them to the conclusion that this more roundabout way of making nitric acid from atmospheric air may give better results with respect to power consumption and output than the most direct process of Birkeland and Eyde by electric discharges through air.

Other interesting applications of the process are the preparation of organic compounds, for instance, the production of urea and indigo by Caro's process, already patented in Italy, in which the action of dicyanamide on phenol-glycine and its derivatives is used. More recently dicyandiamide and its salts have been used as an addition to explosives and powder for the purpose of lowering the temperature in the gun barrel. On account of the high content of inert nitrogen (66.6 per cent) in dicyandiamide, it yields strong pressures in the gun-barrel while it produces only a small quantity of heat. This is of importance, since it lengthens the life of the gun-barrel. In some mixture of powder the cooling action of the dicyandiamide makes the fire disappear at the end of the gun-barrel so that we have now powder which is not only smokeless but fireless.

Ludwig Loewe & Co. (large manufacturers of guns and machines in Berlin) found that calcium cyanamide may also be used for hardening steel. Various tests have given such good results with tool steels, ore drills, pieces of machinery and armor plates, that this new hardening material has been placed on the market under the name of "ferrodur," and has already found an extended application in fine mechanics' workshops as well as in the machine shop.

Copper.

Mining, Milling and Smelting in the Clifton-Morenci-Metcalf District.—A serial in the *Engineering and Mining Journal*, by D. E. Woodbridge, dealing with the general conditions of mining and metallurgy in Arizona and Sonora, contains in the July 21 issue of that paper some notes on the milling and smelting methods used at the properties of the Detroit Copper Co. The greater part of the ore mined requires concentration, the concentrating ore varying from 3 to 5 per cent of copper, and being mostly of the lower grade. The mill concentrates 7.5 tons into one, the product containing 18 per cent of copper, 23 per cent silica and alumina, and 30 per cent sulphur, the tailing loss amounting to 0.8 to 0.9 per cent of copper. The ore is crushed so that 50 per cent or more will pass through a 100-mesh, and 35 to 40 per cent of those fines through a 200-mesh screen. The consumption of water is about 225 gallons per ton of ore treated. The completed mill is expected to treat at least 1,400 tons per day. The mill is operated by Crossley-type producer gas engines, 600 hp. being installed for the full mill. The smelter contains in addition to smaller furnaces one large one of the usual water-jacket type, the dimensions being 204 x 48 inches; the distance from the charging floor to the tuyeres is 14 feet 10 inches. The daily charge passed through this furnace amounts to 400 tons, mainly of sulphide concentrates. A reverberatory furnace, 40 feet long and 9 feet wide, is stated to have been built for experimenting with flue dust and slag. The furnace is running successfully, and takes about 15 per cent of its charge in flue dust daily. Slag flows into the furnace from the settler, passes through it, is tapped out at the lower end into cars and sent to the slag dump. The slag tapped out contains about 0.2 per cent less copper than it did when the tapping was done directly from settlers. The flue dust, fed into hopper bins located above the upper end of the reverberatory furnace, is sprayed over the stream of slag by means of an oil-burner blast. It is distributed over the slag in the upper end of the furnace, and runs along with the stream, disappearing at about the center of the furnace. The cost of operation of the furnace, which is, however, built as an experiment, and, therefore, rather crude, is stated to be about $80 per day, including oil, blast and labor. About 350 tons of slag is stated to pass daily through the furnace.

The Washoe Plant of the Anaconda Copper Mining Co. in 1905. A lengthy paper by L. S. Austin in the *Bi-Monthly Bulletin* of the American Institute of Mining Engineers gives a detailed description of the equipment of the large Washoe plant of the above company in Anaconda, Mont. The company is controlled by the Amalgamated Copper Co. The plant in 1905 handled 7,000 tons of ore daily, with an output of copper of 500,000 pounds. The consumption of coal was 600 tons of coke 400 tons, and of lime-rock 1,000 tons, while 190 tons of flue dust and 9,000 tons of slag and tailings were produced. The blast furnace plant formerly contained seven furnaces,

each 56 x 180 inches in area at the tuyeres. Three pairs of them, including the 21 feet space between each pair, were united, so that there are now three large furnaces, each 51 feet long, and one furnace 180 inches long. At the slag floor level are seven fore-hearths or settlers, two fore-hearths for each of the large furnaces. On a matte track, at a lower level, stand the matte ladles into which the fore-hearths are tapped. The furnaces have side-boshes of 8 inches, making the width at the throat 6 feet. They have no end boshes. The bottoms of the crucibles slope towards the two tapping holes. The top of the furnace is closed, the gases being carried off by three downtakes to a large dust chamber adjoining the blast-furnace building. The number of tuyeres is 88, set at 14-inch center, 46 of them being arranged at the back and 42 at the front side of the furnace. Usually the four tuyeres nearest the ends of the furnace are plugged, as it is found that by this means secretions are somewhat prevented. The bustle pipe extends completely around the furnace, with two branches from the blast main, so that two valves must be closed in order to take the blast off the furnace. The tuyere pipes are 6 inches in diameter, the nozzle having a 1-inch hole. The fore-hearths are 16 feet in diameter and 5 feet high. The dust chamber is 40 feet wide, 280 feet long, and 20 feet high, and is provided with sheet-steel pyramidal hoppers for the purpose of drawing off the dust into the charge cars. The grade of the matte produced is 40 per cent, and the loss in flue dust amounts to 3 per cent. The large furnaces are stated to have the following advantages: Saving in fuel, saving in jacket water, quick and large discharge of matte and slag, decrease of incrustation, elasticity of operation, large flow through fore-hearths, the possibility of varying the slag in the different parts of the furnace, lesser labor cost. The working plant contains 56 McDougall roasting furnaces, as improved by Evans and Klepetko, arranged in 14 rows of 4 furnaces each. The reverberatory furnace plant contains 7 reverberatories, which are the largest in the world. Their dimensions are as follows: Length of hearth of No. 1, 115.8 feet; Nos. 2, 4 and 5, 102.5 feet; Nos. 3, 6 and 7, 112.5 feet. All hearths are 19 feet wide, and the fire-boxes are 16 feet x 7 feet, or 112 square feet area. The construction and operation of the furnaces are described in detail. The furnaces can smelt from 250 to 325 tons of ore daily, and burn from 55 to 60 tons of coal per 24 hours, or 21 per cent of the charge. About 10 per cent of this is recovered from the cinders as coke. A typical charge consists of calcines, limestone (put into the charge at the McDougall roasting furnaces) and flue dust, containing SiO_2, 26.1 per cent; FeO, 31.3 per cent; CaO, 2.9 per cent; S, 8.1 per cent, and Cu, 9 per cent. The slag contains SiO_2, 37.8 per cent; FeO, 38.6 per cent; CaO, 4.6 per cent; Al_2O_3, 6 per cent, and Cu, 0.37 per cent. The loss of sulphur by volatilization is 33 per cent. The converter plant contains 10 converter stands for converters of the barrel type, the converters being 8 feet in diameter and 121 feet 6 inches long, of 10 tons capacity. The lining consists of siliceous ore, which contains from 70 to 85 per cent of silica, with values in copper, gold and silver. The operation of the converters is also described at length. The last part of the paper gives a brief description of the arsenic plant, the coke washing plant, the slime ponds and the brick plant.

Pyritic Smelting.—As a contribution to the discussion of a paper before the British Institution of Mining and Metallurgy, by Alabaster and Wintle, which was abstracted in the June issue of Electrochemical and Metallurgical Industry, G. N. Blakemore gives his experiences in the practice of the process. He claims that hot blast is not of the least commercial use in smelting copper ores which carry up to 20 per cent of sulphur, or any other class of sulphide copper ores. He has saved 7d. per ton of ore by discontinuing the use of hot blast at the furnaces of the Great Cobar Copper Co. Moreover, the furnaces produced a better grade of matte and ran faster on cold than on hot blast. The ore from the Great Cobar mines contains on an average about 12 per cent of copper. The hot blast had a temperature of 500 to 600° F. By stopping the use of the hot blast on the furnace at the Nymagee [illegible] increasing the depth of the furnace from 5 feet to 6 inches of charge to 11 feet 6 inches, he obtained a first matte running from 10 to 15 per cent of copper and equal to that produced with the hot blast, while the furnace smelts 250 tons per week more than when hot blast was used. This ore carries up to 18 per cent of sulphur. He advocates large furnaces, a deep ore column and a large volume of air from a positive source such as, for instance, blowing engines, as they are used on iron-blast furnaces. He recommends the following type of furnace where large ore reserves are available. Twenty feet long by 4 feet at the tuyeres, 16 to 18 feet depth of ore column, water-jacket from feed floor to crucible, volume of wind per minute 30,000 cubic feet of free air, from a blowing engine capable of delivering this quantity against any pressure up to 10 pounds per square inch. A furnace of this kind, he claims, will smelt daily 400 long tons of ore with not more than 20 per cent of sulphur, while the tonnage ought to run up to 550 per 24 hours with ores carrying 30 to 40 per cent of sulphur. He believes that within a limit of 20 per cent silica it is of no importance what the slag carries in silica, as long as it remains fluid enough to run out of the furnace and is low enough in copper. He has produced slags with from 21 to 50 per cent silica, the copper contents of which were practically the same.

Gold.

Tube Milling Practice.—In a paper read before the Chemical, Metallurgical and Mining Society, of South Africa, published in the April issue of the journal of that society, W. R. Dowling describes in detail the plant of the Robinson Deep Gold Mining Co., Johannesburg, and the method of operating the tube mills. The plant comprises eleven classifiers, each 2 feet x 2 feet, of the ordinary inverted pyramid form with sides sloping at 60°. They receive the tailings from 200 stamps, crushing through a 700-mesh screen, as well as the return pulp from the tube mill. The underflow pulp goes to two unwatering spitzkasten, one of which is located at the head of each of the two tube mills. The latter are 22 feet long, with central feed and discharge, the internal diameter of the shell being 5 feet. They discharge onto shaking amalgamating tables, which receive 190 1½-inch shakes per minute, and have a fall of 10 per cent. One mill is provided with four of these tables, each 12 x 5½ feet, while the other has eight small tables of half the length of the first-mentioned ones, but the same width. In the operation of the tube mills, according to the author, a most important point is the separation from the battery pulp of that portion which it is proposed to regrind, and the plant is now being increased by half in order to effect a still more complete separation. He also lays great stress upon the necessity of careful sampling for grading, the samples being taken from the overflow of the classifier for the purpose of controling the working. The tube mills are run with the object of reducing the percentage of material remaining on the 60-mesh in the classifier overflow to a minimum. The dewatering spitzkasten thicken the pulp to, say, 40 per cent of water and 60 per cent of solids, and they also exclude a great deal of slime, which is led around the mill in the overflow, to join the effluent material. Another important point in the running of tube mills is the constant care and attention needed to meet any variation in the volume or nature of the battery pulp, in order to run the mills to their full capacity. It is quite easy to run them below that capacity by feeding thin pulp and not much of it. The mills at the Robinson plant are lined with silex, and the time of stoppage for relining has gradually been decreased from a week to three days, four hours, the latter figure including the shoveling out of pebbles and sand after stopping, the hammering out of the worn linings and the introduction of the pebbles and sand after the mill has been relined. The pebbles used are imported from the Danish and French coasts, and have a diameter of from 2 to 3 inches. It has been found that considerable economy could

be effected by using a certain amount of selected banket ore. The pulp after leaving the mill is evenly distributed over the plates by means of a launder, along the heads of the tables, with holes pierced through the bottom. These holes act like spitzkasten, so that the first shaking table gets the richest and heaviest portion of the pulp.

The Homestake Slime Plant.—An article in the *Mining Reporter*, June 21, by H. J. Baron, contains a brief description of the plant at present under construction for the treatment of gold-carrying slimes of the Homestake Mining Co. The pulp from the 4,000 tons of ore daily treated at this plant is, after amalgamation, separated into 2,400 tons of sand and 1,600 tons of slime. The latter carries average values of 80 cents per ton, and is at present allowed to go to waste. The plant is being equipped with filter presses constructed by Mr. C. W. Merrill, the cyanide manager of the company. The presses are constructed so that they can be readily and rapidly discharged automatically, so as to reduce the labor of discharging to a minimum. The following results are stated to have been obtained on a 10-ton capacity press, of 35 tons weight, and fitted with frames of 4 feet x 6 feet in cross-section, which press has been in operation for upwards of eighteen months on slimes carrying values of from 80 cents to $1.20 per ton. On a 131-charge run of 1,291 tons, the charges being all discharged without opening the press, the average assay value of the slime before treatment was 91 cents per ton. The average value of the slimes after treatment was 10 cents per ton, and the extraction by assays per ton treated was 90 per cent. The recovery in precipitate was 83 per cent, or 83 cents per ton. The treatment is simply lixiviation in the press and no agitation. The amount of solution used per ton of slime was 0.73 ton, the amount of water used in sluicing being 4 tons to 1 ton of slime. The thickness of the slime cakes was 4 inches. Twenty-four presses are being installed at the plant, each of which weighs 65 tons and has a capacity of 26 tons to a charge. The strength of solution used will be 1 pound of cyanide to 1 ton of solution. A recovery of 92 per cent of the assay gold values in the slimes is expected to be made, at a total cost of not exceeding 25 cents per ton of slimes treated. The new plant is so arranged that the flow of material throughout is by gravity. It is run by electric power, the power required for operation at a capacity of 1,600 tons daily being estimated at approximately 1/10 hp. per ton of material treated.

Rand Metallurgical Practice.—E. M. Weston, in a paper in the *Australian Mining Standard*, gives a brief review of the results obtained by the Denny Bros. at the new plants erected on the New Gooch and the Meyer & Charlton mines, Transvaal, and discusses the adaptability of a similar process to Western Australian practice. The slimes plant at the above mines consists of three or four conical steel vats, provided with a slow-agitating device, for the purpose of preventing the settlement of the slimes on the sides to too great an extent. The slime is fed by gravity from one vat to another, and the vats automatically thicken the pulp, so that the bulk of the solution is delivered clear enough to be sent to the zinc boxes, while the thickened slime is sent to the filter presses to have the remaining solution pressed out and then to be washed. Cyanide solution is also used in the battery, and most of the gold is dissolved in the boxes and launders before reaching the plant for sands and slimes. A great saving in water consumption is thus claimed to be effected, as compared with the method of using decantation slimes plants and slimes dams, which is quite a factor at plants where water is expensive. The cyanide consumption is low, being 5 pounds per ton treated. The total amount of solution in circulation for an 80-stamp mill is 2,133 tons, which carries 2 dwt. of gold per ton in solution and 0.08 per cent average cyanide. In March the screen assay at the Meyer & Charlton mill showed 10.871 dwt. The average value of the residues from the sands was 14.4 grains, and from the slimes 6.91 grains. There were 70.7 per cent of sands and 29.3 per cent of slimes. The average assay value of the residues was, therefore, 12.39 grains, which is equivalent to an extraction on screen assay of 95.302 per cent. In discussing the applicability of this method in West Australian plants the author points out that the ore in the Transvaal is comparatively easy to be treated, whereas, in West Australia as well as in Cripple Creek the difficulties are much greater. Agitation of the slimes before filter pressing has nearly always been found necessary, and it has been necessary to pass a great proportion of the slimes solution through presses. A plant, such as that designed by the Messrs. Denny might, in cases where the ore was refractory, not allow of sufficient agitation to promote rapid solution.

Notes on Stamp Mill Practice.—In the course of a paper read before the recent meeting of the Canadian Mining Institute, Mr. C. de Kalb gives some interesting remarks about various details connected with stamp milling practice. He deals with screens, foundations of the battery, the stamp duty, mortar liners, shoes and dies, inside amalgamation, the dressing of the outside plates, the use of salts with coppery ores and outside amalgamation. In regard to stamp duty he is of the opinion that this term is relative, depending not only on the character of the ore but on the rate of discharge which has been found to give the most economical results. The metallurgist who has not advanced to the point of tolerating the low-crushing efficiency per horse-power of the stamp mill, for the sake of the extraction by amalgamation which he can make it yield, is in error at the very foundation of the trade. If a large recovery of the gold values inside of the batteries is not obtained, there is something wrong with the methods, or else the stamp mill should not be used, and the whole process needs study and revision. He emphasizes the fact that it should be borne in mind that it is not necessarily the mill man who can put through the largest tonnage, who is earning the most money. Thus, during a recent experience in California the author found that he could obtain a recovery of 91.5 per cent when crushing 0.121 ton per stamp per hour, while by using the same screen (with 0.028 inch-diameter of opening), but readjusting the mill in regard to drop, discharge, etc., the stamp duty could be run up to 0.188 ton per stamp per hour, but the recovery was reduced to 77 per cent. As far as the author's experience with concrete mortar foundations is concerned, it has led him to regard them with disfavor. He has in each instance found the ratio of broken stems to be as 1 to 3 in favor of built-up wooden blocks. To reduce the wear of concrete blocks he has employed, with marked success, a facing of ¼-inch wrought iron sheet, at least 4 inches wider than the width of the mortar base and 2 inches larger. This effectually stops the wear of the sharp edge of the mortar on the concrete if any nuts loosen and furnishes a suitable surface for a rubber cushion.

Nickel.

Conversion of Nickel Matte into Nickel by Means of Blast Rich in Oxygen. In *Metallurgy*, Nos. 10 and 11, R. Hesse gives the result of laboratory experiments made in Prof. Borcher's laboratory at the Technical University of Aachen, to convert nickel matte into nickel by blowing air rich in oxygen into the matte. The contents of oxygen in the air were run up to 63 per cent, in order to get as high a temperature as possible in the converter as well as to make the blast more effective to carry out the reaction intended. Incidentally the action of nickel monoxide on nickel sulphide was studied, by heating a mixture under different conditions in the electric furnace. The author found, with regard to the latter question, that even when the temperature is increased considerably, it is not possible to get metallic nickel by the action of NiO on NiS. The experiments have, however, shown that an action of NiO on NiS takes place, notwithstanding statements at other places to the contrary. It begins only at temperatures exceeding 1,400°, but it was not possible to obtain a quantitative result according to the chemical reaction $NiS + 2NiO = 3Ni + SO_2$. In regard to converting nickel matte by blast containing oxygen it

was established by the experiments that it is not possible to perform that operation, on account of the oxydation of the nickel, which in the presence of silica is slagged off as nickel oxide. Whether nickel is set free by reaction between NiS and 2NiO, or by dissociation, it is not liberated as such as long as there is undecomposed NiS present, as Ni and NiS (or Ni_2S?) alloy in all proportions. The richer the alloy is in Ni, the more free Ni is oxidized besides NiS. NiO and Ni, however, alloy also, and therefore it is out of the question to obtain commercial Ni by blowing.

Zinc.

Electrometallurgy of Zinc.—In a paper read before the Sixth International Congress of Applied Chemistry at Rome, E. Ferraris discusses the possibility of zinc smelting in the electric furnace. He gives a brief outline of the various types of electric furnaces proposed, including his own, which has been experimented with for some time at Monteponi. In ordinary furnaces the zinc is distilled and the cinders removed once a day, whereas, in the electric furnace a slag must be formed which can be run off frequently. For that reason an excess of reducing carbon must be avoided, which would prevent fusion of the gangue of the ore. The author states that this absence of an excess of reducing coal is the weak point in the electric furnace, as the zinc fume is partially reoxidized by the carbonic acid from the reaction of carbon monoxide on the zinc oxide, and the oxidized parts impede condensation of the zinc in the liquid state. The composition of the ore should be corrected so as to produce a readily fusible slag and which does not dissolve the zinc oxide too rapidly. The best slag for this purpose is stated to be the monosilicate of iron and calcium. In order to assure continuity of the work and to make the composition of the slag uniform, at least 25 per cent of normal slag must be added to the ore. The author gives the following heat calculation for the treatment of calcined or roasted ore, containing 50 per cent of zinc, to which 50 per cent of fluxes and slag is added, presupposing that a continuous resistance current is used, passing, as in the Monteponi furnace, through a bath of slag of regulatable thickness, upon which descends a closely-mixed conical charge. The slag and the furnace must be kept at about 1,200° C., in order to extract the zinc and smelt the gangue and fluxes. The heat needed to treat 1 kilogram of ore is stated to be as follows: Reduction of 500 grams of zinc, 665 Cal.; loss by oxidation of coal, 222 Cal.; heat to bring the charge to about 1,200° C., 450 Cal., a total of 893 Cal., or about 1,000 Cal., reckoning loss by radiation and secondary reactions. One steam horse-power develops in 24 hours an energy corresponding to 15,176 Cal. If the efficiency of the horse-power transformed into current and heat is 80 per cent. and there is one steam horse-power applied to the dynamo shaft, this will suffice to treat 12 kilograms of zinc ore per day, or 3,600 per year of 300 days. The author estimates the cost of the effective steam horse-power from a hydro-electric works at 100 lire per year, and, therefore, the treatment of a ton of calamine would cost 28 lire for motive power. The author calculates the following cost of treatment per ton of calamine with an average plant: Motive power, 28 lire; reduction coal, 128 kgs. at 50 lire per ton, 6 lire; electrodes, refractory material, repairs for furnace, etc., 2 lire; labor, etc., 4 lire; a total of 40 lire. As the cost of treatment in ordinary Silesian and Belgian furnaces varies from 40 to 50 francs, according to the cost of coal, it is greater, or at least equal to that with electric furnaces, on the basis of 100 lire per year and steam horse-power. The author is convinced that experiments made on a large scale would solve the question in details. The efficiency of the electric furnace might be increased by filling it with hot calamine after calcination or roasting.

Electrogalvanizing.—The importance of detecting flaws in boiler tubes at an early stage, that is, before any machining or fitting is done on them, was early recognized by the English admiralty. In this work it was found that zinc plating greatly facilitates the process of inspection, showing up defects and flaws which cannot otherwise be detected. An illustrated article in the London *Electrician*, of July 20, describes a large electrogalvanizing plant installed at the Thames Iron Works, Ship Building & Engineering Co. for galvanizing a variety of work, principally tubes [illegible]. The Cowper-Coles regenerative process is used, in which the zinc is replenished by means of zinc dust, which is a by-product from zinc-distillation furnaces, instead of by rolled zinc anodes. This is effected by [illegible] of zinc dust and coke arranged in a separate tank, through which the solution is continuously passed by means of a centrifugal pump, the anodes being of lead. When zinc anodes are employed it is found that only a small [illegible] solution, the greater part crumbling away; the adoption of a filter bed prevents the waste that would [illegible] as the zinc is caught by the filter and gradually dissolved, and the solution kept free from particles in suspension, which have a very detrimental effect on the deposited zinc.

Iron and Steel.

Utilization of the Blast Furnace Gases for Power Purposes.—In our Vol. III., p. 95, F. du P. Thomson, who was connected with the pioneer gas power plant of the Lackawanna Steel Co., gave a review and very conservative estimate of the commercial possibilities of blast-furnace gas for the development of power. In our Vol. III., pp. 150 and 190, A. J. Rossi gave further figures on the same subject, on the basis of the results obtained in practice by various engineers. A further valuable contribution to this subject, and one which has not yet been mentioned in these columns, is a paper read by H. Freyn at the end of last year, and published in full, with the discussion, in the February issue of the *Journal of the Western Society of Engineers*. Since this subject is at present of great interest, in view of the activity of the United States Steel Corporation in this field, the conclusions of Freyn shall here be given. He finds that a power plant of about 10,500 hp. capacity, complete in every detail and installed in connection with a blast-furnace plant, would be capable, when running at full-load capacity, of producing 1 hp. per year at the cost of $17.88, no value being placed on the blast-furnace gas. The enormous saving as compared with the production of power in a steam engine plant is still more striking, when the cost of generation of electric current is considered. According to Freyn's tables, 1 kw-hour at full-load capacity of the plant could be produced at 2.95 mills, which is away below the best figure ever reached with a steam engine power plant. Even under worse conditions, that is, when the power plant is running on an average of only 50 per cent of its total capacity, the cost of generation of 1 kw-hour is but 5.50 mills [illegible]. By comparison, F. du P. Thomson's figures may be given. In a plant with four 1,000-kw units, with 50 per cent overload capacity, a load factor of 50 per cent and 24 hours power, the total cost per kw-hour is 1.475 cent, in a plant with four 1,500-kw units, no overload capacity, with a load factor of 85 per cent, 24 hours power, the total cost per kw-hour is 0.821 cent. An eventual increase in the capacity of the power plant would still tend to reduce the cost of the generation of power per unit, as certain expenditures for the power plant of 10,500 hp. would remain unchanged for additional power units. Figures are finally given of the actual results obtained in the works of the John Cockerill Co., in Belgium, which operates at present seven blast furnaces of about 1,200 tons [illegible]. To produce the power for the electrical service, they had installed 1,000 kw in steam engines and [illegible] and 800 kw in steam engines and 2,000 kw in gas engines in 1905. The total operating cost was 137,163 francs [illegible] 206,328 francs in 1905; i. e. the increase in operating cost amounted to 31 per cent only, whereas, the [illegible] power plant had been increased 370 per cent. The cost per kw-hour fell from 0.088 franc in 1902 to [illegible] in 1905.

so that the cost of 1 kw-hour in 1905 was but 25 per cent of the corresponding cost in 1900.

Corrosion of Iron Wire.—*Farmers' Bulletin*, 239, of the Department of Agriculture, contains an account of an investigation by A. S. Cushman, on the corrosion of fence wire. It seems certain that modern Bessemer and open-hearth steel wire rusts much more rapidly than the iron wire made twenty or more years ago. The author produces some evidence to show that manganese, especially if it is unevenly distributed in the steel, is at least in part cause of the trouble.

Cleveland Convention of American Foundrymen's Association.—A very full and profusely illustrated account of the proceedings of the recent convention of the American Foundrymen's Association may be found in the July issue of *The Foundry*, from which we abstract the following papers which are metallurgical interest:

Fluxes in the Cupola.—N. W. Shed pointed out that the value of fluxes in the cupola is not generally appreciated by foundrymen. There are two available fluxes for the cupola, namely, limestone and fluorspar, but limestone is far cheaper and far better than fluorspar as a flux. The most practical and easily fusible slag is a monosilicate, which means equal amounts of silica and alkaline bases. It is a good rule to figure the limestone on the weight of the coke, using 25 per cent limestone. This amount will flux any ordinary coke ash with the average amount of sand on pig and scrap. If the amount of sand on the pig is excessive, figure 30 per cent limestone on the weight of coke. With a low coke ash, machine pig and clean scrap, the limestone may be reduced to 20 per cent and make a good cinder.

High-Grade Ferro-Silicon and Cast Iron.—A. E. Outerbridge, Jr., discussed the beneficial effects of adding high-grade ferro-silicon to cast iron. He first referred to an important problem which comes up daily in foundry practice, namely, to provide metal melted in one cupola and one heat, suitable for a great variety of castings and requiring wide variations in physical properties and chemical composition. The author's practice is to group all the small work for the day, needing soft ductile iron, so that it will be cast in the beginning of the run of iron from the cupola which is, of course, charged with the required amount of the softest grades of iron; the proportion of silicon in such mixtures approximating 2½ per cent. This is followed by a medium grade of iron, calculated to contain about 1.9 per cent silicon, suitable for miscellaneous castings of medium weight and thickness. Finally, come strong iron mixtures for large and heavy work. They may contain as little as 0.5 per cent silicon, and may possess high chilling property. The author refers to some special castings which have been made in the foundry of Wm. Sellers & Co., and which have united the two antagonistic properties of high tensile strength with softness and ductility in a remarkable degree. Tensile tests of bars 0.5 inch square section formed in the same mold with the castings and attached thereto, have shown, when turned and pulled on the 100,000 pound Emery hydraulic testing machine, extraordinary tensile, viz., 40,535 to 44,505 pounds per square inch. The element silicon is practically the governor which determines the degree of hardness of cast iron. The author made a long series of experiments, both with commercially pure metallic silicon and with ferro-silicons of various concentrations. Pure metallic silicon was found impracticable, partly on account of the high melting point and partly on account of its brittleness, which causes the powder to float to the surface of the molten iron if sprinkled into the ladle, or to simply cake if fastened to the bottom. Very good results were obtained with a ferro-silicon containing 50 per cent silicon, which in the form of a coarse powder was sprinkled into the molten iron as it was tapped into the ladle. The author speaks well of the chill cup test, which is in effect a physical analysis of practical value, which enables the foundry foreman to know the character of every ladle of iron, small or large, before he pours his metal. The chill cup tests of the treated and untreated iron showed at once that the small amount of 50 per cent ferro-silicon added (half a pound in a 200-pound ladle) has dissolved in the iron, wiping out the chill in the metal almost completely. The metal of the treated bars was much softer in appearance, as was already indicated by the chill cup tests, but, instead of being weaker, the treated bars were all stronger than the untreated bars. In other words, increased softness and increased strength are here combined. The rationale of the increase in strength with increased softness is probably to be found in the deoxidation of the iron by the silicon added in its nascent condition.

Thermit in the Foundry.—W. M. Carr discussed new applications of the thermit process in foundry practice. In a number of cases where it was important to pour light, thin sections, and where the metal was rather dull, the desired increase in temperature and fluidity was gained by an addition of thermit placed in the bottom of each hand ladle, into which the iron was poured from the bull ladle. In this way the castings were free from cold sheets. In another case a foundry was equipped with a single 2-ton Tropenas converter and only one 5-ton ladle. It was desired to pour some ingots and castings requiring twice the capacity of the converter, so that it was necessary to make two heats separately and pour them both into one ladle. The evident difficulties of this method of making "double blows" were overcome by keeping the first charge of steel hot and liquid by the insertion of thermit cartridges attached to iron bars and plunged to the bottom of the ladle at regular intervals during the progress of the second heat, which occupied about 45 minutes. The result was entirely satisfactory. In the treatment of large headers—both iron and steel—the application of thermit in suitably sized cartridges has kept them open longer and produced a better feeding effect than any other means. In the manufacture of semi-steel castings, two uses of thermit have been developed. First, when the necessary steel is melted in the cupola with regular charge, the insertion of a semi-steel cartridge in the ladle produces a poling action, which thoroughly mixes the metal with no loss in temperature. Second, if it is desired to make but a few semi-steel castings among a larger number of ordinary castings, steel borings or chips can be put in the ladle bottom, the iron tapped in, and by the insertion of a semi-steel cartridge the steel will be melted and thoroughly and completely dissolved in the iron. This obviates the necessity of making special heats for occasional semi-steel castings. The author then spoke briefly on repairs to gray-iron and steel castings.

Electric Smelting of Iron Ore at Sault Ste. Marie.—A paper by Dr. E. Haanel, in the August issue of the *Journal of the American Chemical Society*, contains some details which were not covered in his preliminary report, which was the subject of various articles in this journal. These details relate to the construction of the contact between the electrodes and the steel shoe and to some observations made during the experiments. Thus some of the runs made showed occasionally a rapid temporary decrease of ohmic resistance shortly after the current was put on, when the charge consisted of conductors in the form of small pieces, which phenomenon has recently been studied by Broun (see p. 148 of our April issue). This trouble occurred chiefly at the beginning of a new experiment, before the furnace had acquired its normal temperature. By adding a few shovels of iron ore, omitting flux and charcoal, it was sometimes possible to cause the electrode, which to keep the current constant, had, by hand regulation, been elevated to return to its normal position. In a few instances, however, this method failed, especially when the furnace was choked with fines and the gases evolved escaped under great pressure. If the explanation offered by Broun is correct, that the de-

crease of the ohmic resistance during the "Anheizungsphase" is occasioned by the more intimate contact of the conducting pieces of the charge, due to pressure of escaping gases from the pores of the carbon and the carbon dioxide evolved from the limestone, it is evident that preheating the charge, which might be effected by utilizing the carbon monoxide resulting from reduction, would entirely overcome this difficulty, if the charge were sufficiently porous to permit the gases evolved to escape at low pressure. Under such conditions the electrode would maintain its normal position throughout the operation, requiring to be lowered only to keep step with its consumption. In a general comparison with the blast furnace, Dr. Haanel points out the following advantages of the electric furnace: Original small cost of furnace, absence of bulky or costly charging machinery, small expense involved through breakdown, small cost and ease with which repairs may be made, no serious complications arising from scaffolding, loss due to wrong composition of charge reduced to a minimum, perfect control of the temperature in the reducing and melting zone.

BOOK REVIEWS.

Handbuch der Anorganischen Chemie. In four volumes. With the coöperation of others, edited by Prof. R. Abegg. Vol. III., part 1. Leipzig: S. Hirzel. 466 pages. Price (in New York) $5.00.

Among the special features of the program of this new, voluminous German handbook of inorganic chemistry are the endeavor to cover fully the results of physico-chemical research; to state not only facts, but to give at the same time the theoretical connection between chemical phenomena; to exercise critical judgment in giving figures and other data. The editor of the handbook is Dr. R. Abegg, professor of the University of Breslau, and editor of the *Zeitschrift für Elektrochemie*, the organ of the German Bunsen Society.

The chemical elements will be covered in three volumes, divided into eight parts, each of which relates to one group of the periodic system. In addition, a special volume will cover general principles. The programme is, therefore, as follows: Vol. I., general principles; Vol. II., part 1 and 2, elements of the first and second group of the periodic system; Vol. III., part 1, 2, 3, elements of the third, fourth and fifth groups; Vol. IV., part 1, 2, 3, elements of the sixth, seventh and eighth groups.

Vol. II., part 2, and Vol. III., part 1, have so far been published. Vol. III., part 1, covers boron, aluminium, the rare earths, etc. For each element a critical discussion of determinations of its atomic weight is first given. This is followed by a concise summary of the preparation and properties of the element and its compounds, with a rather full bibliography on the subject. When completed this new handbook will be a very useful reference work, since the endeavor to give only reliable information is plainly evident.

Thermodynamik technischer Gasreaktionen. Seven lectures by Prof. F. Haber. Munich and Berlin: R. Oldenbourg. 296 pages. Price (in New York) $3.35.

This book is a treatise of thermodynamics of reactions between gases. "It has been written not for the sake of the theorist, but for the sake of the engineer." Knowledge of chemical technology is presupposed, also knowledge of the first elements of calculus. But the principles of thermodynamics are developed, explained and illustrated, beginning with the very foundation.

The first lecture deals with the "latent heat" of chemical reactions and its relation to the energy of reaction. Prof. Haber adopts Helmholtz' conception that a chemical reaction involves a certain latent heat; the two terms of the "Gibbs Helmholtz equation" for the total energy are, therefore, not called free and bound energy, but energy of reaction and latent heat. It would appear that in this way Haber succeeds in facilitating the fundamental conception.

The second lecture deals with entropy and its role in reactions of gases, the third lecture gives another derivation of the fundamental formulas and a discussion of their relation to reactions between solid substances. In the fourth and fifth lectures examples are given of reactions taking place without a change of the number of molecules and with such a change. The sixth lecture deals with the determination of the specific heats of gases, the seventh with the determination of the equilibrium of gases, with a discussion of the theory of various engineering problems.

In the presentation of the principles of thermodynamics, Dr. Haber manifests much individuality in details, but he follows in principle the safe and sane orthodox methods of Clausius, Helmholtz, Van't Hoff and Planck, and his book is free from the heresies of modern "energetics." This is most fortunate. While Planck's work will remain the classical treatise of thermodynamics for physical chemists, Haber's book is worthy a similar place of honor in the library—or rather on the desk—of the chemical engineer. His book is not a dry presentation of principles, since the principles are constantly illustrated in a most interesting manner by applications to engineering problems. It cannot be too often said that the only way of getting a working knowledge of principles is by means of applications—not qualitative but quantitative applications. For the engineer a general knowledge of a principle is good for almost nothing, if it does not enable him to base on it a numerical calculation.

Some of the subjects dealt with in Haber's book are of great actual interest, like his calculations on the fixation of atmospheric nitrogen. Other engineering problems which are fully discussed, are the Deacon chlorine process, the contact process of making sulphuric acid, the manufacture of water gas, etc.

The book is dedicated by the author to his wife, Mrs. Clara Haber, Ph. D., "*zum Dank für stille Mitarbeit.*"

Pebble Mills.

Practice has established the fact that pebble mills are the only apparatus whereby all kinds of drugs, chemicals, colors, enamels, glazes, ores, etc., can be reduced to an absolutely impalpable powder without bolting. The pebble mill pulverizes all of the material charged into it and delivers a uniform product. There are no contaminating surfaces; there is no dust in the work room, while the saving in power, labor, space and time is considerable without any loss of the material.

While other mills either crush, twist or cut the material, pebble mills grind principally by friction, the effect being produced by the sliding, tumbling and rolling inside of the mill of a great number of flint pebbles or porcelain balls, which are mixed with the substance to be ground, and the movement being caused by revolving the mill at a regulated speed. However, there is also the grinding effect due to the direct impact of the pebbles falling down upon the material after they have been carried up by the revolution of the mill. Whether the friction or the impact of the falling pebbles will prevail depends on various circumstances, especially on the speed of revolution, the height to which the mill is filled with the pebbles, and whether dry or wet grinding is employed.

In any special case the best conditions for the working of the pebble mill must be found by experiment.

The Abbe Engineering Co. of New York City, who have done, during the past years, a great deal towards the present perfection of the pebble mill, have for this reason installed a complete testing laboratory where they have at hands of a dozen of their different mills constantly set up ready to cut, grind or mix either dry or wet product. In this testing

laboratory the company is enabled to determine the best conditions of operation.

While the impact of the falling pebbles has some effect, yet in most cases the grinding by friction is prevailing. Since a great number of pebbles or balls are used in a minute, every two of which can be considered an individual small grinding mill, there is an enormous grinding surface within a limited space. The simplicity of the operation of the mill results in the necessity of using only unskilled labor during ordinary

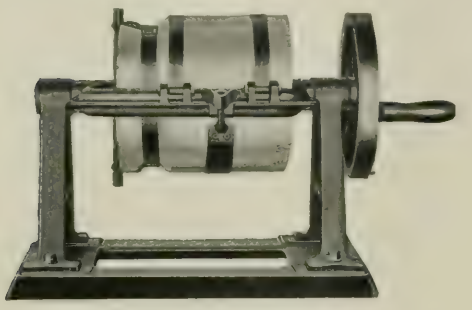


FIG. 1.—"THE LITTLE TROJAN."

operation, attention being needed only while charging and discharging.

No parts require dressing or sharpening, as in all other grinding mills, as practically there is scarcely any wear; consequently the mills are always in good working order.

In the following we describe some types of pebble mills made by the Abbé Engineering Company. All their pebble mills are lined with the finest quality of vitrified porcelain, thus presenting a grinding surface which will neither contaminate nor discolor the material being pulverized. In the smaller sizes the lining is made in one piece, but in the larger pulverizing cylinders the lining is furnished in the form of blocks, which are fitted perfectly into the inner circumference of the cylinders, and cemented in place with the finest quality of cement.

Instead of vitrified porcelain, the Abbé Engineering Co. furnishes, if desired, either natural flint (silex) or wood linings. When the pulverizing cylinders are to be used for the grinding of white enamel, glaze, etc., they furnish a colorless solution to be mixed with the enamel and used in the same way as regular cement when lining the mill.

Porcelain presents the best possible grinding surface, and with ordinary usage and proper care such linings will wear for many years, the exact life of a lining being determined by the variety of the material being pulverized.

These machines will pulverize either wet or dry material, and will pulverize all of same, delivering a perfectly uniform product without bolting. As a mixer, the pebble mill is, without any exception, the best thorough dry or wet mixer in the world, and is, therefore, largely used in the drug business and in the cement manufacturing trade. For mixing colors it is unsurpassed.

Besides the linings there are few things in a pebble mill of greater importance than the quality of the pebbles used in the cylinders. Soft or brittle pebbles not only wear out very rapidly in the machine, but also deteriorate the quality of the material being pulverized, thus making them costly at any price. Pebbles of uniform shape (round or oval) are also far preferable to those of irregular shape, and greatly increase the grinding capacity.

The great simplicity of operation may be seen from the following rules which cover the full directions for operating the Abbe pebble mills.

The dry grinding mills, after having been mounted on brick, stone or iron frame, and encased in such a manner that the bearings are outside of the casing, should be cleaned out thoroughly. This can be accomplished in the best manner by putting in a charge of pebbles (sufficient to half fill the cylinder), and with them a charge of sand (enough to fill all the space between the pebbles). Then close the manhole with the tight cover and revolve the mill at its rated speed. In this way all the cement on top of the lining will be ground off and the pebbles are cleaned at the same time.

The mill should be run for several hours, but as soon as it is clean, which can be determined by looking into it once in a while, can be emptied by replacing the tight cover with the grate discharge cover and the cylinder set running, by which operation the sand is sifted through the openings in the grate and the pebbles or balls are retained. It will take from 2 to 5 minutes to discharge, after which the grate is taken out and the machine is ready for a charge of the material that is to be reduced. Now the mill must be run until the product is of the fineness desired, at which time the same method must be followed to discharge, as stated before, and when the mill is empty it is ready for the second charge.

With wet grinding mills the same directions must be followed, only they require no casing and the operation is the same as above, except that when discharging, a tight cover, provided with a valve, is used instead of the grate discharge cover. The cylinder is turned with the valve downward, and the latter opened so as to allow the material to run into any receptacle placed to receive it.

Another point to which attention may be called is that the pebbles should not be shoveled into the mill, but put in carefully by hand. Figs. 1 and 2 show two types of jar mills of the Abbé Engineering Co. The porcelain jars of these machines are imported. They are manufactured from the finest raw materials obtainable, made in the plastic state, thus forming jars that are impervious to the action of even such materials as ink. These jars are of so superior a quality that constant use of them has never worn them out.

The only protection necesary is that in charging the balls

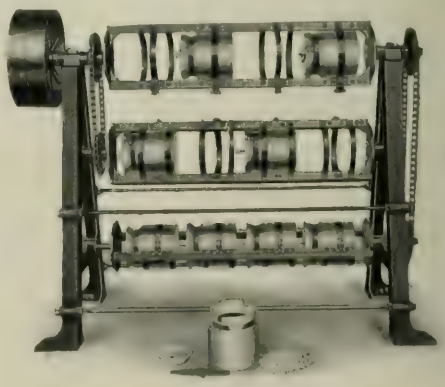


FIG. 2.—TWELVE-JAR LABORATORY MILL.

or pebbles should not be thrown in, since this might result in a break of jars. The balls or pebbles should be carefully put in by hand.

Fig. 1 shows the type called by the company "The Little Trojan," which has a shipping weight of 90 pounds, and requires a floor space of 10½ x 25 inches. The normal speed of the mill is 60 r. p. m. The jars measure outside 8.75 x 9.65 inches (in any case of giving dimensions in this article the diameter is given first and then the length). This jar

is capable of handling from a few ounces up to 5 pounds at a time.

Of course, it is possible to use a battery of such jars and revolve all the jars from the same driving shaft or motor. The advantage of using a battery of several jars is that a different material can be ground or mixed in each jar at the one operation, thus four, six or a dozen products can be handled at a time. The same quantity of material need not be put into every jar, so that one can reduce an ounce of one, a pound of another and 5 pounds of a third.

There is one ceramic color and chemical manufacturing firm in the United States using more than 250 of these mills. This alone will go far towards showing the usefulness of these laboratory mills.

The Abbé Engineering Co. builds a battery of jars up to the number of six in one row, while for a greater number the arrangement is shown in Fig. 2. This shows a twelve-jar laboratory mill, the dimensions of all the jars being outside, 8.75 x 9.65 inches. The floor space required is 6 feet 6 inches by 2 feet 6 inches. The normal speed is 60 r. p. m., and the shipping weight of the whole mill is 1,000 pounds.

Fig. 3 shows a pebble mill, the outside dimensions of the cylinder being 30 x 30 inches. Its normal charge, taking sand as unit, is 200 pounds. Its shipping weight is 2,500 pounds, the floor space required 5½ x 3¾ feet, the normal speed, 38 to 44 r. p. m. With dry grinding this mill requires 1½ hp. With wet grinding ¾ hp.

Fig. 4 shows a pebble mill with the latest improved manhole frame and covers, made by the Abbé Engineering Co., with swing bolts and thumb nuts, as used on all mills designed to handle from 500 to 4,000 pounds of charge when sand is taken as unit. The manhole frames are made with detachable flanges, so that they can be easily replaced with new pieces of flat sheet iron when the inside lining is worn out. These are the flanges that hold the lining in position, and gradually wear down with same. To users of these mills this improvement will readily appeal as an important factor, as it avoids the riveting in of new manhole frames when a machine is to be relined.

The illustration also shows how it is possible to attach a stuffing box, with pipe connection, for steam or air inlet on one

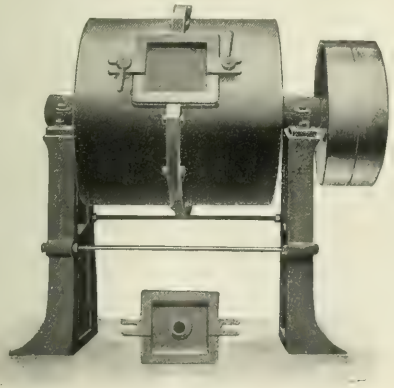


FIG. 3.—PEBBLE MILL.

shaft and a way-cock on the other shaft, to enable grinding or mixing material under pressure.

From the preceding description it will be evident that the simplicity in construction will result in a considerable saving of power, labor, space and time. Concerning power and space required by the Abbé pebble mills the following table will give interesting notes:

Outside Dimensions of Cylinder	Charge, Taking Sand as Unit	Floor Space Required	Do	Do
6 x 8 ft.	30 000 lbs.	12 x [illegible] ft	1[illegible]	[illegible]
6 x 5 ft.	2880 lbs.	1[illegible] x [illegible] ft	12	[illegible]
5 x 4 ft.	1200 lbs.	10½ x 7 ft	[illegible]	[illegible]
4½ x 3½ ft	800 lbs.	9 x 6 ft	[illegible]	2½
3½ x 3½ ft	500 lbs.	7½ x 6 ft	[illegible]	[illegible]
3 x 3½ ft.	360 lbs.	6¼ x 5¼ ft	2	[illegible]
30 x 30 in.	200 lbs.	5½ x 3¾ ft	1½	[illegible]
30 x 19 in.	120 lbs.	3¾ x 3¾ ft	[illegible]	½

These pebble mills are undoubtedly destined to play a very important part in chemical and metallurgical arts. In the cement industry as well as in the cyanide process for winning

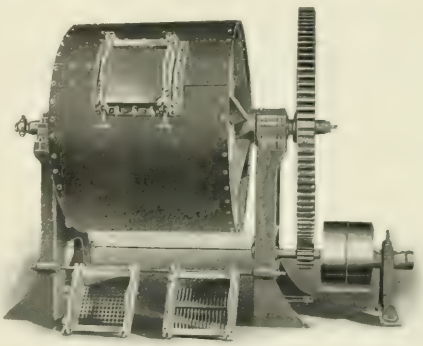


FIG. 4.—LATEST TYPE OF PEBBLE MILL.

gold, the pebble tube mill has found many applications in the past, and is constantly extending its sphere of usefulness. A description of Abbé pebble tube mills paying particular attention to their special spiral feed and discharge arrangement may be found in our Vol. III., page 41.

The Electrochemical Department of the Siemens & Halske Company.

Last year the old well-known Berlin works of the Siemens & Halske Co., in the Markgratenstrasse, were removed to a magnificent large new plant outside of Berlin, at the Nonnendamm, midway between Spandau and Berlin, West. This new plant is officially named the Wernerwerk, in honor of the late Werner Siemens, and is mainly devoted to what the Germans call Schwachstromtechnik, namely, telephone and telegraph engineering, construction of instruments, etc. The Wernerwerk, however, also contains the electrochemical department. The following description is taken from a recent profusely illustrated book on the Wernerwerk, issued by the Siemens & Halske Co., Engineer Hans Dominik being the author.

For many years the Siemens & Halske Co. have paid special attention to electrochemical developments. They took part in the first development of electrolytic copper refining in Germany, and reference may also be made to the well-known Siemens process for winning copper from ores (our Vol. II., p. 225), which, while not commercially successful itself on a large scale, was instrumental in influencing the later work of Hoepfner, which again started in this country the successful nickel process of the Canadian Copper Co. by its chief chemist, Mr. David H. Browne.

Successful on a very large scale has been the Siemens & Halske cyanide process with electrolytic precipitation in South Africa. This has been the subject of numerous articles in these columns.

Electrolytic Bleaching.

The Siemens & Halske Co. has done considerable work in connection with the late Dr. K. Kellner on bleaching with electrolytic hypochlorite. They developed successfully three arrangements, all of which are based on a connection of a larger number of electrolytic cells in series in one apparatus, so as to get the required small voltage at the terminals of each cell from a given electrical supply network at a higher voltage. In all these arrangements the series connection is obtained by means of bipolar electrodes.

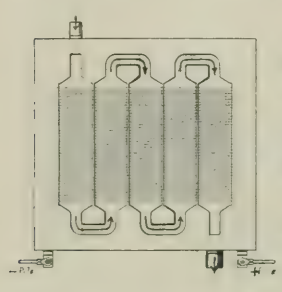


FIG. 1.—ELECTROLYTIC BLEACHING.

The first system was the so-called point electrode. A thin platinum wire was stitched through a rubber plate, which was then vulcanized, and the wire was cut through at the different stitches and the ends of the wire bent straight. In this way the rubber plate was pierced through by a great number of short pieces of platinum wire ending on both sides of the plate in a point, one side acting as anode the other as cathode. However, the rubber plates were not very durable, and were especially affected by the product of electrolysis on the positive pole.

A modification of this method was therefore devised, which consisted in glass plates around which a helix of platinum-iridium wire was wound. These plates were placed tight into recesses of the walls of the electrolytic cell, so that the wire on one side acted again as anode and the wire on the other side as cathode (the American patent of Kellner for this construction was abstracted in our Vol. III., p. 395). This arrangement was, until recently, the normal construction, and has been used in many fiber paper and textile plants, the aggregate electric power used in cells of this type being 2,000 hp. A power consumption of 180 hp. is sufficient for bleaching 10,000 kg. of sulphite cellulose per day.

The third system is characterized by the use of horizontal electrodes, and represents a further increase of efficiency. The arrangement is shown in Fig. 1, the electrolyte passing through the cell along a narrow zigzag path, as shown by the arrows in the illustration. The walls between the zigzag paths divide the horizontal platinum-iridium wire-net electrodes into two halves of opposite polarity. This last system has already found many applications in the textile industry.

In connection with this subject the electrolytic production of hydrogen and oxygen from water may be mentioned, since here also bipolar electrodes are used, to get the required low voltage of 2.5 at the terminals of each cell from the higher voltage of the current supply. The Siemens & Halske Co. uses the apparatus of Dr O. Schmidt, in Zurich, the mechanical form being similar to a filter press.

The products of water electrolysis, oxygen and hydrogen gases are used for lighting, for filling balloons and for autogenous welding with the oxyhydrogen flame.

Electric Furnaces.

The Siemens & Halske Co. has bought the Kjellin patents for the electric induction furnace for making steel. Since this process has been described in detail repeatedly in our columns this notice must here suffice.

The Siemens & Halske Co. also has developed several types of calcium carbide furnaces, but the calcium carbide industry has not attained in Europe the importance which had been hoped for. Much is expected, however, from the use of calcium carbide as a starting material for making calcium cyanamide, which can be used directly as fertilizer or may be worked up into cyanide.

For this purpose calcium carbide is used in powder form, and is heated in hermetically closed red-hot iron retorts in an atmosphere of nitrogen. (See the paper by Frank on another page of this issue.)

Discharges Through Air.

While the production of calcium cyanamide represents one method of binding the atmospheric nitrogen in useful form, the company has also carried out extended experiments on the direct oxidation of the atmospheric nitrogen by means of arc discharges. These experiments have not yet reached finality, but the results are stated to be promising.

On the other hand, the production of ozone by the silent discharge through air has already reached industrial importance. If ozone can be made cheap enough it may be used for many purposes.

At present the strongly oxidizing and sterilizing properties of ozone have been used in practice especially for two purposes. The treatment of ordinary starch with ozone yields a peculiar product, the "ozone starch," which, with its properties, stands midway between starch and dextrine. Like true starch, ozone starch is insoluble in cold water, but dissolves in hot water without forming a paste like dextrine. For this property ozone starch is used extensively as a binding material for paints and colors and for printing on cloth, linen and all kinds of fabrics. For the production of ozone starch there are now in operation two large plants in Kyritz and Fuerstenwalde, in the Province of Brandenburg, in Prussia.

The second important application of ozone is for the sterilization of drinking water, which has often been referred to in these columns. Two plants erected by Siemens & Halske in connection with large municipal water works, were described in our Vol. I., page 179; these are the plants of the cities of Wiesbaden and Paderborn. (It was recently reported that the ozone plant of Wiesbaden had been abandoned, but from a note of Mr. H. P. M. Halbertsma in *Engineering News* of July 12, this is not the case, but the plant is operated each month during one day only, so as to keep it in running order and to preserve it as a reserve supply in case of emergency. For general use the ozonizing process is said to be too expensive, and

FIG. 2.—PORTABLE OZONE PLANT IN RUSSIAN WAR.

is made so principally by the many bacteriological tests which are necessary for the control of the proper working of the plant.)

The municipal ozone plant of Paderborn was designed for sterilizing 60 to 90 cubic meters of water per hour, and has furnished in continuous operation during day and night the whole drinking water for the city for a number of years. The result has been that the city of Paderborn, which formerly had an epidemic of typhoid fever almost every year, is absolutely free of it now since the ozone plant was started.

Ozone apparatus in smaller size for sterilizing of water in hospitals or in private residences are in use in various cities. Apparatus of this kind have been installed, for instance, in all police stations in Petersburg in the autumn of 1905, when there was the danger of an epidemic of cholera morbus, in order to be able to provide the poor people in case of emergency with pure water.

Fig. 2 shows a transportable ozonizing plant in the form used in the Russian war in order to have always good water for the soldiers.

Portable Voltmeters and Ammeters.

In addition to the line of direct-current switchboard instruments made by the American Instrument Co., attention may be called here to a series of accurate portable direct-current voltmeters and ammeters.

These instruments are built on the permanent-magnet moving-coil type, although very different in important respects from instruments of the same class made heretofore.

By the use of cylindrical steel pivots journaled in the best grade of watch jewels, friction errors which so often occur because of damage in shipment, or from jars during continued service, are entirely overcome. This method of pivot control is far more difficult to manufacture than the usual conical pivots working in conical jewels; but when once accomplished instruments of this kind will remain practically frictionless for many years.

The permanent magnets in these instruments are made of the best magnet steel obtainable, hardened and aged in accordance with the most improved methods. The moving systems combine light weight with extreme stiffness, and with the method of pivoting already noted it is impossible for the moving coil to come in contact with the pole pieces, or with the core of the magnetic system.

By the selection of a proper winding for the moving coils, "American" instruments have a somewhat larger torque than usual, which permits the use of very strong controlling springs. Because of this fact, and also on account of the unique manner in which current is taken into and out of the moving coils, "zero errors" are completely avoided in ammeters and milli-voltmeters as well as in voltmeters.

PORTABLE INSTRUMENT.

The series resistance in American portable voltmeters is wound so as to be perfectly non-inductive, and each instrument has a negligible temperature coefficient. All voltmeters are adjusted to a uniform resistance of 100 ohms per volt, and, therefore, when multipliers are used they can be interchanged. Also the measurement of insulation resistance and the location of faults and grounds are much facilitated as the necessary calculations are greatly reduced.

The portable voltmeters are self-contained up to and including 750 volts, but any range beyond that can be provided for by the use of external multipliers.

Ammeters are self-contained up to and including 200 amps. When it is desired to use one ammeter with several ranges, the plan adopted is to employ a milli-voltmeter with separate interchangeable shunts. Each milli-voltmeter with its leads has a fixed resistance of 10 ohms, and is designed to give full scale deflection with a potential difference of 50 millivolts. All [illegible] shunts are adjusted for a potential drop of 50 millivolts at full load. Consequently, any instrument can be used with any shunt or any number of shunts with one instrument. This is a new feature that will be highly appreciated by engineers who wish to make current measurements through a wide range of values.

"American" portable instruments are made in two types, designated Nos. 4 and 5. Type 4 instruments are provided with special iron shields made of the wood case, and may be used close to large generators or under other conditions where an unshielded instrument would be useless. Type 5 instruments lack the iron shield, but in consequence are lighter in weight and somewhat smaller in size.

The new factory of the American Instrument Co. is located in Newark, N. J., but the general sales office is at No. 1114 Chestnut Street, Philadelphia, where Mr. James G. Biddle, President, as well as General Sales Agent of the company, makes his headquarters.

Notes.

Electrical Books.—The Pratt Institute, of Brooklyn, has issued a list of books on electricity, a special feature being concise annotations concerning the contents and character of the books mentioned.

Hydro-Electric Installations.—At the Reliance Works of the Allis-Chalmers Co., of Milwaukee, there are at present under construction and in various stages nearing completion, forty-three hydraulic turbine units, aggregating 166,000 hp., for shipment to various portions of the United States, Canada and Mexico.

Laboratory Apparatus.—The Arthur H. Thomas Co., of Philadelphia, has just issued a second edition of their Catalogue F on laboratory apparatus, especially selcted for laboratories of chemistry and the biological sciences. The catalogue —which is a book of 444 pages—is alphabetically arranged according to subject matters and profusely illustrated. A very full index at the end of the book greatly enhances its usefulness.

Engineers' Society of Western Pennsylvania.—The membership list of the Engineers' Society of Western Pennsylvania, corrected to June 30, 1906, has just been issued. The thoroughness with which it has been prepared makes it a model of its kind. The Society has now 892 members; the large majority is located in or around Pittsburg, but the Society has members in other States all over this country and even abroad. This gradual abandonment of the local character of the Society is undoubtedly the reason why, while the absolute number of members attending the monthly meetings has remained about the same, yet when expressed in per cents of total memberships, there is a decrease from 27.8 per cent in 1880 to 4.7 per cent in 1905. The present president is Mr. Julian Kennedy, of Pittsburg.

Porous Diaphragms and Cells.—The Deutsche Steinzeugwaarenfabrik für Canalisation und Chemische Industrie, of Friedrichsfeld, in Baden, manufacture for electrochemists' use a refractory and acid proof ceramic product known as "Dr. Buchner's Patent Material." Besides the adaptability of this material for evaporating dishes, stills, cooling worms, refractory tubes, etc., it is also made in the form of porous diaphragms. The manufacturers claim that these diaphragms, although somewhat thicker than the ordinary ones, offer exceedingly small resistance to the passage of electric current, and are not easily penetrated by liquids. Furthermore, that their thickness is advantageous from the fact that they thereby resist mechanical influences much better. The resistance of every diaphragm in ordinary electrolytes is determined before

leaving the works, and the determination is guaranteed. They are constructed in any shape desired; *e. g.*, in the form of cylindrical cells, square cells or plates. It is stated that this product is remarkable for its excellent thermo-conductivity, its minute contraction and expansion under the influence of changes of temperature, and lastly, for the fact that it may be made either porous or dense according to requirements. The electric resistance of the diaphragms is claimed to be smaller than that of Pukall and Mettlach cells of the same dimensions. The manufacturers are represented in this country by Messrs. F. Bertuch & Co., Temple Court Building, New York City.

Autogenous Welding of Aluminium.—While the difficulties of soldering aluminium have been overcome by the invention of suitable solders, containing foreign metals (zinc, tin, etc.), yet it has been found impossible in the past to devise a method of autogenous welding of aluminium, by which the two pieces could be so firmly united as to form one solitary unit. Autogenous welding of lead with the oxy-hydrogen flame has long been used to advantage in practice. But all endeavors to develop an analogous process for aluminium failed on account of the well-known property of aluminium to form a fine oxide film on its surface in contact with air; this film may be invisible, but prevents the welding together of aluminium. Mr. M. U. Schoop, in Paris (who contributed an article on autogenous lead welding to our Vol. III., p. 260), endeavored to solve this difficulty by finding a substance which, on the one hand, would protect the heated aluminium surfaces from the air, and which, on the other hand, would dissolve immediately any oxide film, so that an intimate and direct contact of the two surfaces to be welded together is rendered possible. These experiments of Mr. Schoop were successful, so that it is no longer difficult to weld autogenously in a perfect way aluminium of any form, whether sheets, rods, wire, etc., just as it is done with lead with the air-hydrogen or the oxy-hydrogen flame. A burner of special construction is used in which an intimate mixture of the two gases is accomplished before they leave the burner tip. During May last official tensile tests were made by the Laboratoire du Conservatoire des Arts et Métiers, in Paris, which showed that the mechanical properties and resistance of the joints are exactly equal to those of pure aluminium. A specially simple and effective application of this method of welding is the manufacture of aluminium tubes and pipes in any thickness and size, and a company has already been formed in Paris for the production of such tubes for which there is a large demand in connection with automobiles, balloons, autocanoes and on board ships in general. It is to be expected that the aluminium industry, the growth of which in recent years has been so phenomenal, will thereby find further fields of usefulness.

Perkin Jubilee of the Coal Tar Industry.—We have already alluded in these columns to the fiftieth anniversary, to be celebrated this year, of the epoch making discovery, by Dr. William Henry Perkin, of the dyestuff "Mauve," by which the foundation was laid for the coal-tar industry. The American "Committee of Fifteen," with Dr C. F. Chandler as chairman, Mr. A. Kuttroff as treasurer, and Dr. H. Schweitzer as secretary, has arranged a banquet and symposium on the coal-tar industry for October 6. A personal token will be presented to Dr Perkin, who will attend the celebration. The program also includes the foundation of a Perkin medal to be awarded annually to an American chemist for distinguished work in applied chemistry, and the establishment of a nucleus of a fund for the foundation, at the Chemists' Club in New York City, of a reference and circulating library, covering the entire field of theoretical and applied chemistry, which is to be in charge of a salaried librarian, and to contain duplicate sets, one of them to be used for circulation among American chemists. It is estimated that $5,000 will be amply sufficient to cover the expenses of the personal token and Perkin medal, while the sum of at least $50,000 will be necessary to place the library on a permanent basis.

Personal.

Mr. Max Meyer, who was connected with Messrs. Eimer & Amend for a long series of years, has recently entered the firm of Ernst Leitz manufacturers of microscopes and optical instruments, in New York City.

Mr. O. F. Saugstad, formerly connected with the H. & M. Automatic Regulator Co., as master mechanic and superintendent, and also with the Davis & Roesch Temperature Controlling Co., has established offices at 21 Park Row, New York City. He will practice as consulting and advising engineer, making a specialty of temperature and pressure regulation in all their branches.

Digest of U. S. Patents.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

Ozone.

(*Continued from page 290.*)

No. 527,320, Oct. 9, 1894, John T. Donovan and Henry L. Gardner, Springfield, Mass.

Produces ozone, "as much as thirty-eight and five-tenths per cent" at the positive electrode of an electrolytic cell, having separate but communicating chambers containing the electrodes. The electrolyte is a solution of a permanganate of any solid metallic base, such as sodium, silver or barium, but is preferably a saturated solution of potassium permanganate containing crystals of permanganate and 5 to 10 per cent of sulfuric acid.

No. 563,288, July 7, 1896, Walter Lobach, Chicago, Ill.

Oxidizes substances such as oils which it is desired to bleach, by subjecting them to the action of nascent oxygen. The apparatus may consist of spaced electrodes with an intermediate dielectric plate. A high-frequency current is passed transversely between the electrodes, and oxygen or air and the oil to be treated longitudinally between them. In another apparatus, the ozonizer is vertical and comprises a cylindrical electrode surrounded by a dielectric layer, concentrically arranged within a tubular electrode spaced therefrom. The liquid flows down between the two electrodes, which are placed in a chamber communicating with a gas reservoir. Another apparatus comprises a horizontal revoluble chamber having a perforated hollow shaft communicating with the gas reservoir. The liquid to be treated is contained in the cylinder. A series of concentric electrodes, one a rod, the other a perforated tube, extend between the ends of the cylinder and are electrically connected to rings upon which bear brush terminals.

No. 565,952, Aug. 18, 1896, Emile Andreoli, London, England.

A vacuum tube, or one containing rarified gases is provided internally with an electrode consisting of a carbon filament, a wire or several wires, or a bundle of small serrated wire, the points of which face the glass. Outside of, but spaced from the tube, is arranged, longitudinal or spirally, another serrated wire electrode.

No. 568,177, Sept. 22, 1896, Nikola Tesla, New York.

An electric motor of high self-induction is placed in the primary of an induction coil. One terminal of a lighting circuit is connected to one binding post of the motor, and thereby through one field coil, the brushes and commutator and the other field coil, to the brush of an interrupter consisting of a rotating disc with insulating sections. The other terminal of the circuit is connected to another brush bearing on this controller. Around the controller is a circuit of low self induction, which includes a condenser and the primary of the transformer. The terminals of the secondary are connected to spaced electrodes between which air to be ozonized is driven by a fan.

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Reinforced Concrete and Cement in Furnace Construction.

The great San Francisco disaster showed well that the fire-proof qualities of any structure in which cement entered [illegible] been greatly underestimated. The great mechanical hardness of cement when once well "set" shows that there is a strong chemical bond between the water and the complex lime-aluminate-silicate formed by the hydration. This fact reduces the vapor pressure as regards water to a low figure even at a low red heat and preserves mechanical hardness. Mr. David H. Browne, in his description of concrete electrolytic vats, published in our first volume, stated that it was possible to slowly heat these vats to a low red heat. Cement is also used as a bond for the hearth of silver cupelling furnaces. It has even been used for the same purposes for lining the long rotary kilns. Here it is subjected to a terrific heat, and acts simply to hold particles of ganister or silica sand together at low heat.

"Cement mortar" has also been used, we are reliably informed, in building the brick walls of metallurgical furnaces that are put to an inside heat approximating 1,600° C. It has the useful effect of making the outside wall one piece mechanically, and prevents cracks if properly reinforced by external buckstaves or other iron. By such a construction the buckstaves can be spaced farther apart, and the arrangement is far superior to iron plates and besides considerably cheaper. The same engineer proposes to use reinforced concrete for the external shell, and no good reason against its use here can be induced, provided walls are thick enough for the temperature gradient to be low, and not subject the concrete to an excessive heat. Reinforced concrete is, of course, used for furnace foundations, dust chambers and stacks. We hope to be able to publish some time an article describing this interesting phase in the development of metallurgical engineering. The use of cement as an ingredient in "semi-refractory" construction is extremely promising.

Electric Steel and Sociology

The study of sociology is hardly within the province of this journal, nevertheless we feel it our duty to refer to some recent work of the United States Government that is of great sociological interest, because of an extraordinary misapprehension of the object of the investigation that seems to have arisen in the mind of the public. It has been asserted that the barbarous people inhabiting the Far West have made great progress in their knowledge of the arts and sciences of civilization, and that it is no longer possible for the adventurous traveler to that interesting region to astound the natives by means of the childish toys with which he was formerly wont to excite their wonder and admiration. Following the policy

of enlightened paternalism, which so brilliantly distinguishes our Government, Dr. Day, last year, undertook the investigation of this important subject among the native tribes inhabiting Portland, Ore. For this purpose he hit on the ingenious idea of testing their knowledge of the metallurgy of steel by exhibiting to them a simple electrical experiment. Incredible as it may seem this interesting and instructive experiment on the state of the scientific knowledge of a Western people, has been described in some of the daily papers as though it were a real metallurgical investigation.

* * *

That there is absolutely no excuse for such an astonishing blunder may be readily seen by referring to an article in the *Technology Quarterly* of June, 1906. Here Prof. Robert H. Richards, of the Massachusetts Institute, describes the electric furnace shown to the inhabitants of Portland under Dr. Day's auspices. "It consists of a hollow cylinder standing on end, 12 inches in diameter and 30 inches high inside." The remarkable effect of the experiment on the natives conclusively demonstrates the untrustworthy nature of the traveler's tales mentioned above. "There was boundless enthusiasm among the Portland people who had gathered to see it, when * * * 200 pounds of white-hot steel was tapped from it." And then Prof. R. H. Richards adds, with subtle humor: "It (the furnace) is still in the experimental stage as to the means of controlling the contents of the steel in silicon, carbon, manganese, sulphur and phosphorous. This control, as well as the means for obtaining sound ingots free from bubbles, will have to come by experiment. At the time of writing it does not look as if any of these would give serious trouble." He concludes with an ironical estimation of the cost of producing a ton of steel. To anyone but superficially familiar with the work of Héroult and others, it is obvious that the highly trained men of science employed by the United States Government would not waste money on a crude experiment of this kind, except for the sociological purpose described. But it is a matter of no small concern to us that some of the daily papers of the East, and presumably their readers, are in a state of intellectual development but little more advanced than that of the barbarians of the Pacific Coast.

Pre-Heaters and Electric Furnaces.

The cost of a heat unit of electrical energy is from two to forty times the cost of the same unit in the combustion value of coal. The relative ratio, of course, depends on the price of the two sources of energy in any one locality. With electrical power at 3 mills per horse-power hour and coal at $4.00 per ton, the ratio is about fifteen. With cheap electric power at 1 mills, and coal at $8.00, which happens in several localities, as in the French Alps, and to a less extent in inland Norway, the ratio is only two. While at a coal field with slack at a very low figure, not much over cost of mining, the ratio is forty. The use of electric heating comes from the high thermal efficiency of electric furnaces. The efficiency of an electric furnace is usually over 75 per cent and often as high as 95 per cent, whereas the efficiency of heating by coal is in evaporating or drying or in a boiler rarely as high as 75 per cent, and is sometimes as little as 3 to 7 per cent in indirect heating, crucible melting or in muffle heating. By a careful use of "metallurgical calculations" the cost of electrical heating can be figured exactly, and the able metallurgist can reckon thus the relative value of two methods for any given set of conditions.

* * *

Now, the efficiency of a coal or gas furnace varies greatly in the operation. If a pyrometer be placed in such a furnace in the charge, the rise in temperature is at first a straight line, but finally curves, and at the end reaches a straight line parallel to the horizontal co-ordinate. In other words, the first efficiency is high, but is gradually reduced to zero, for radiation and serviceable heat in chimney gases absorb all the energy of combustion, and at 1,400°—1,900° C. in most instances all the heat of the fuel is lost. Now, if the furnace were designed to use a combination of gas or coal heating for the first part of the metallurgical work and electric heating for the last part of the work, an average high efficiency would be attained, while the "commercial efficiency" or cost of each useful calory would be reduced to lowest possible point. Such is the function, indeed, of "pre-heaters," either two furnaces combined or a fuel-heated furnace separate but directing its heated product into the electrical furnace. A somewhat broader use of this principle is seen in the great "direct" steel mill where heat of pig iron never leaves the material until the finished product is allowed to cool so that it can be loaded on the railroad car.

The new steel plant at Syracuse, using Dr. Héroult's process, uses the open-hearth furnace for the first and easy part of the work, while the difficult high-temperature refining is effected by the Héroult electric furnace. The practical combined pre-heater as a shaft furnace directly on top of an electric furnace, so that the two are really one metallurgical instrument, has not been yet proved publicly practical, at any rate commercial, although we have private information of several interesting attempts that are likely to eventually be successful. Most electric furnaces are used for operations only possible by electrical heating, as reduction of alumina, graphitizing articles and the manufacture of carborundum. When the practical solution of the "pre-heater" is worked out and made successful, it will open large vistas of successful electrometallurgical operations that are most alluring and wonderful, and the demand from this source on the manufacturers of prime-movers and electric generators cannot be estimated, because of its possible magnificence.

Simplified Spelling.

President Roosevelt's order to the Public Printer that all messages and documents from the White House shall be printed according to the recommendations of the Mathews committee, has been the subject of much comment in the daily press. Some twenty years ago or so Fürst Bismarck gave out an ukase that all reports, etc., of the different departments of the German Government should be written or printed in the so-called German type, instead of the international Roman type. His order was severely criticised, but his will prevailed, and has undoubtedly done much towards the preservation of the "German alphabet." Not so very long ago we noticed an excellent book, brought out by a prominent German publisher, on a scientific subject of world-wide interest, namely, on

systems of physical and electrical units and measurements—printed in the German type. It is almost incredible that Germans should fail to realize that to many men in other countries who understand the German language, and who would be interested in the subject, a book printed in German letters looks very much like one printed in Russian or Chinese characters. If Fürst Bismarck's well-meant, but undoubtedly retrogressive order could have so much success, it seems certain that President Roosevelt's progressive and broad-minded order will be far more effective in the end. Those who might expect a very speedy development in this direction should consider that in journalism there is a distinct inter-relation between editor and printer, with enough existing trouble and no necessity to borrow any.

* * *

We have referred to this whole matter because in the discussion of President Roosevelt's order it seems to have been generally overlooked that a single branch of the system of our Government has already done pioneer work for simplified spelling in a special field, namely, the United States Patent Office, in the spelling of chemical words. While others spell sulphide or sulfide, they spell sulfid, and this method will undoubtedly prevail in the end, on account of its simplicity. For the present every chemist and metallurgist uses his own way of spelling chemical terms, and this would seem to be part of his chemical individuality. For this reason we have given the widest latitude to our contributors in this respect. The real international language of chemists as well as engineers employs formulas and drawings. A single chemical equation is immediately understood all over the world. Some improvements, though, would be desirable; thus France should adopt N instead of Az. In the symbols used by physical chemists, on the other hand, there reigns a chaos which is even worse than that which still exists in the symbols employed by electrical engineers. There is at least a desire and an endeavor on the part of electrical engineers to get together and devise a uniform system of electrical notation. But how far we are still from the realization of such a plan is indicated, for instance, by the fact that the Germans, almost to a man, insist on retaining the letter w (widerstand) for electrical resistance, while everybody else uses r.

Extension of the Senses by Scientifc Instruments

Principal Griffiths, in his Olla Podrida on scientific progress, in his presidential address before the British Association for the Advancement of Science, touched on ions, chemical thermodynamics, scientific education of the British youth, university extension and other things too divers to mention. At the end of this recital from his mental scrap book, he discussed the broad proposition of the growing multiplicity of our sensible knowledge due to the advance in scientific experimenting. The eye is a continuous photographic camera, and the camera preserves the record of its vision in the negative plate. Here we can distinguish between our natural senses and the artificial senses,—the creations of experimental science. The natural senses are those of touch, smell, taste, seeing and hearing in the main. They have simple extensions as the act of "hefting a stone" to judge roughly of its weight against that of a dimly remembered standard, is, of course, an adjunct sense, that of muscular co-ordination. But to the electrical engineer,—especially to those who have grown with batteries and motors from their childhood up,—the indications of the ammeter, of the "gasing" on the plates of a storage battery are not unseen phenomena (a contradictory pair of words) but as real to him as is the blowing of the wind to the average mortal. In short, the ammeter is his electrical eye. The layman is apt to deny the reality of electricity, as the Christian Scientist is apt to deny the reality of the entire external world. But the Christian Scientists of our acquaintance do not give an imaginary yell when pricked by an actual pin, as a general rule, and a shock of "200 volts to the ground" ought to convince the most skeptical of the actuality of electricity. The difference between the expert and the ordinary man lies in the fact that the phenomena in one case are seen every day and understood every day, while in the other they are neither seen nor understood. The difference, let it be stated, is in the "subject" and not in the "object," to use convenient metaphysical terms.

* * *

There is a further difference between our natural senses and our artificial and extended senses, and that difference is found in the fact that man can reject old pieces of apparatus and install improved ones. On the other hand he can only train and educate his natural senses within the limitations of their physical being as developed by the course of evolution for the use of savage man. Marconi can tear up a coherer and put in a new one, but no one would care at the present time to have a new ear grafted in his head. It should be noted that both the ear and the coherer are detectors of two kinds of wave-motion. The ear detects the wave-motion due to the change in pressure in the air, while the coherer detects the difference in the electro-magnetic displacements, which take the form of true wave-motion. A second difference is due to the permanence and accuracy of the indications of our artificial senses. The best illustration of this is seen in the photographic maps of the heavens which the Harvard University observatory sends out to astronomers in different parts of the world to study. These are permanently correct and enduring. The eye of the furnace-man working the "Ziervogel process" at the Argo smelter at Denver, can err in his judgment of the heats from copious indulgence in colored aqueous solutions of ethyl alcohol, but Prof. Howe's determinations of the physical constants are correct, lasting, and in fact give the essential details of this metallurgical operation.

* * *

This wonderful mechanical age, too mechanical as many think, perhaps is due to the magnified and projected extension of our senses by the advance in experimental science. The effect on the faculties of the growing generations who are accustomed to lisp at the age of two years on the long distance telephone, will undoubtedly be greater than upon those who now occupy the stage and have seen the advance at a later period of their lives. Immanuel Kant gives in his Critic of Pure Reason a dictum concerning our natural knowledge. This was to the effect that the indications of our senses are always imperfect, and sometimes misleading, but that if care and discrimination are used, the true can be sifted from the false, and an approximation of the truth can be secured. This saying of a metaphysician, who was also a physicist, is justified more and more by the march of scientific advance.

The Iron and Steel Market.

The feature of August has been the excitement in the pig iron market. Purchases of pig iron have been made freely for delivery through the first quarter and first half of next year, which is almost unprecedented for August, particularly as regards Bessemer and basic. Prices have been advancing sharply, the advance beginning in July, before there were any large purchases, except of Southern iron, and altogether the advances have presented the quite unusual feature of preceding rather than following the heavy buying. Ordinarily the large buyers at least succeed in getting in at near the bottom, but in this case they have freely covered at advances.

The total sales of pig iron during August have totaled many hundred thousand tons, a large total for any month, and particularly large for August, which is usually a dull month. Advances have ranged from 50 cents to more than a dollar, the month closing with prices from $1 to $2 above the low point reached in the late spring or early summer.

There has been a complete reversal of sentiment among buyers of pig iron as regards the prospective supply. The pendulum of sentiment had swung too far in one direction, and it will probably be found later to have swung in August too far in the opposite direction. During the second quarter the prevailing sentiment among buyers evidently was that pig iron production would be greatly increased in the second half, with perhaps a decline in demand. Production decreased towards the end of the first half, but merely through weather conditions and the necessity for relining an unusual proportion of merchant furnaces. Labor conditions in the South were a contributory cause. Buyers suddenly lost faith in an increase in production through the completion of new furnaces, and the foundries, but not the steel works, started accumulating stocks against a scarcity, thus precipitating the condition which they feared.

As a matter of fact, production is bound to increase materially. The furnaces which were operative Jan. 1, 1906, will make as much pig iron in the second half, taken as a whole, as they did in the first half, while the new furnaces completed during the first half, having a rated capacity of 880,000 tons a year, will naturally contribute more than in the first half. There are scheduled for completion in the second half additional furnaces having a rating of 975,000 tons a year, and in the first half of 1907 furnaces having a rating of 818,000 tons. Invariably some such plans miscarry, and the actual results will be less than the anticipation, but still quite large. Production of pig iron in 1902 and 1903 was about 18,000,000 tons per year. Following the off-year of 1904 it jumped to 23,000,000 tons in 1905, and in the first half of this year was at the rate of 25,000,000 tons. It is safe to predict that in the second half it will be at a rate exceeding 26,000,000 tons, and that available capacity will be equal to a rate of 27,000,000 tons during the first half of 1907. It is true that the present rate of increase is less than that from 1903 to 1905, but the iron trade cannot continually stand so great an increase as that was. There was undue excitement as to pig iron during August, and it will probably be found some time in the first half of next year that buying and price advancing was overdone, and this will result in a decline, which is likely to reach serious proportions if assisted by financial and political uncertainties.

From surface indications the iron and steel markets are regarded as remarkably strong. The visible future of the market is usually taken as covering only six or nine months at most, and it is extremely bright. The great bulk of the merchant pig iron production is already sold, and contracts and definite orders are on books for the various steel products, covering in most cases the great bulk of the prospective production.

The Carnegie Steel Co. has definitely withdrawn from the market on standard rails for this year's delivery, being already oversold. It has advanced prices on light rails a dollar a ton being sold up for two or three months, and has advanced its price on sheet bars a dollar a ton.

Sales of wire products have been exceptionally heavy, and would be large for any month, while August is usually quiet. The programme of restriction of output for July and August was abandoned, being found unnecessary. In the past two years it was followed, and with excellent results.

Pig Iron.

Against a steady market of $17.25, valley, for Bessemer pig during the first half, sales were made early in August at $17.75, valley, including 30,000 tons for first quarter of next year to the Jones & Laughlin Steel Co. and 70,000 tons for first half to the Youngstown Sheet & Tube Co. Concurrently sales were made for the fourth quarter of this year at $18.00. Later sales were made for first quarter and first half at $18.25, valley, and for early delivery and fourth quarter at $18.25 on up to $18.75. The month closed with $18.25 as the absolute minimum, and $18.50 to $18.75 being paid for early delivery, according to circumstances. Outright sales of Bessemer and sales subject to monthly adjustment amounted in August to 250,000 tons or more. There is still a large unsatisfied demand, and it is certain that large steel producers will be compelled this winter to regulate their steel output to the iron supply, which they have been occasionally forced to do in the past.

Basic has been in active inquiry, but has moved less freely, because furnaces have held off. Sales have been made for first quarter at $18.00, valley, and the market is well established at that figure, with prospects that $18.25 may be obtained.

Southern iron has advanced sharply, with heavy purchases, the month closing with the market practically at $15.00, Birmingham, for No. 2, for delivery next year, and $15.50 to $16.00 being obtained for earlier delivery. The coal strike in Alabama was settled during the month, relined furnaces are coming in, and weather conditions becoming more favorable, indicating that the advance in Southern iron may have been overdone, from the minimum of $13.00 in June and the market of $13.50 at the time of our last report.

Northern foundry iron has advanced freely, with liberal buying through the first quarter and first half. The market for forward delivery of No. 2 stands at $18.85 to $19.10, Pittsburg; $19.50 to $19.75, Chicago, and $19.25 to $19.50, Philadelphia.

Billets and Sheet Bars.

There is only a nominal market in billets, at $28.00 to $29.00, Pittsburg, for Bessemer, and $29.00 to $30 for open-hearth. Only occasional lots can be picked up. Small billets command merchant bar prices, equal to $33.60 per gross ton. The Carnegie Steel Co., on Aug. 20, announced its price for September for sheet bars at $30, Pittsburg, for long bars, with the usual extra of 50 cents for bars cut to specifications, on contracts subject to monthly price adjustment. Such contracts are unimportant, but it is certain that on the more important quarterly adjustment contracts, the price will be named at this or a higher figure, for fourth quarter. The third quarter price was $29. The open market on odd lots has been $30 to $33.

A new Bessemer steel producer was added to the ranks during August, the Youngstown Sheet & Tube Co., Youngstown, Ohio, having completed a standard two 10-ton vessel plant in the record time of a little over a year. For several years it has operated skelp mills, sheet mills and a very complete merchant pipe plant. The addition just completed includes a large blooming mill, which makes no billets, but will supply blooms and slabs for the old skelp mills, a new mill for billets and small billets and a new sheet bar mill. The management hopes shortly to reach 2,000 tons of ingots a day, but the billet, sheet bar and skelp capacity is in excess of this, so that the plant has great flexibility as to character of output. There is a surplus of billets and sheet bars for the open market, but this has been practically all sold for the balance of the year. The completion of the plant releases a large tonnage of crude steel,

which the Youngstown company had been receiving from the Republic Iron & Steel Co., but the advent of the new producer has been scarcely felt in the market. Never before has the starting of a new Bessemer plant made so little impression

FINISHED MATERIALS.

Mills are sold up practically full for the remainder of this year, and well into next year, on plates and shapes. Some rail mills have their entire 1907 production sold, and others from a fourth to a half of their production. All the important producers of steel bars have contracted for substantially their entire output to the middle of 1907, and specifications thus far have been much beyond output. Sheets and tin plates, wire products and pipe have been sold on an average until nearly the close of the year. Booking has been heavy during August, but attended with no excitement. In some quarters advances are expected in sheets, tin plates and wire products.

The following prices are f. o. b. Pittsburg, plus regular freight to destination, and are regular mill prices on carload and larger lots to large buyers for forward delivery:

Beams and channels, 15 inches and under, $1.70 per 100 pounds; tees, $1.75; beams and channels over 15 inch, $1.80.

Plates, tank quality, ¼ inch and heavier, to 100 inches wide, $1.60.

Sheets, 28 gauge, box annealed, one pass cold rolled, $2.50.

Galvanized sheets, 28 gauge, $3.55.

Plain wire, $1.70.

Wire nails, $1.85.

Tin plates, 100-pound cokes, $3.75 per box.

Artifical Soft Graphite.

From Niagara Falls comes the important announcement that Mr. E. G. Acheson has succeeded in devising a method of making soft graphite in the electric furnace. The process is stated to be simple and cheap.

The artificial graphite, made up to the present by the International Acheson Graphite Co., has been hard graphite, and has found a large field of usefulness of its own, especially for electric furnace electrodes, electrolytic-cell electrodes, as a paint pigment, etc. The possibility of making also soft graphite in the electric furnace opens to artificial graphite a vastly greater field of applications, and is expected to make it a very serious competitor of natural graphite, especially since the artificial product is of greatest uniformity.

Artifical Diamonds.

Some time ago we referred to the presidential address of R. THRELFALL before the Birmingham section of the (British) Institution of Electrical Engineers on "some problems of electro and electrothermal chemistry." One of the subjects discussed in his address and not yet alluded to in these columns is the production of artificial diamonds or, what is the same, the artificial crystallization of carbon.

The artificial production of diamonds presents many problems of great interest. As is well known, Moissan succeeded in producing minute diamonds by saturating iron and other metals with carbon at electric furnace temperatures, and then suddenly cooling the molten mass by dropping the crucible and its contents into water, mercury or melted lead. These experiments have been followed up by Mayorana and others, ending with Sir William Crookes, who produces some evidence of the formation of microscopic diamonds by the explosion of cordite in a closed vessel.

By far the most important investigation hitherto published on this subject is that of Dr. A. Ludwig in the *Zeitschrift für Elecktrochemie*, Vol. VII., p. 273 (May 8, 1902), and in several patents; *e. g.*, British 16,908 and 17,119 of 1900, and 5,482 of 1902. Dr. Ludwig has made a brave but unsuccessful attempt to produce diamonds, or rather cubic carbon, on the large scale.

The apparatus consists of compression vessels and compression pumps, enabling carbon to be kept under high pressure. The carbon was heated electrically in compressed hydrogen, and it was found necessary to cool the product of the operation very suddenly; this was accomplished by injecting water at enormous pressure, so that the carbon under treatment was struck by jets of water moving at immense velocities. The result of a successful experiment was to produce a sort of carbon glass, for obviously the rapid cooling does not afford time for the formation of crystals of appreciable size, and the glass tended to resemble black diamond rather than clear cubic crystals. Dr. Ludwig does not state how much of this material was obtained, so that it is not possible to form an opinion as to whether the process has any commercial interest. There can, however, be no question as to the interest and importance of some of the facts disclosed, and as this work does not seem to have received the notice to which it is entitled, Mr. Threlfall gives a summary of some of the experiments.

First in regard to the heating. At a hydrogen pressure of 1,500 atmospheres it was found impossible to form an arc between carbon terminals, even at 70 volts p.d.; while with metal electrodes the formation of an arc required practically no pressure at all, so that heat could not be developed conveniently in this way. According to the old experiments of Dulong and Petit, the cooling effect of different gases depends on the pressure in such a way that the effect is proportional to the pressure raised to the 0.45 power for air and the 0.38 power for hydrogen. These experiments of Dulong and Petit were only made at comparatively low temperatures, and may have been superseded, but it is probably safe to assume that the cooling effect of a gas (particularly hydrogen) does not increase very rapidly with the pressure, so that to produce a state of high pressure by means of hydrogen does not seem an unreasonable proposal, nor does it seem likely that we are to ascribe the non-formation of an arc in hydrogen to excessive cooling.

By placing two carbon rods in contact and in series with a source of current at the hydrogen pressure mentioned (1,500 atmosphere = 9.84 tons per square inch), Ludwig encountered a very instructive phenomenon. The current flowed for a few seconds and then stopped; after a few seconds more it rose again—again to fall to zero. The alternations continue indefinitely so long as the rods are kept in contact. The points of contact are found to have been converted into graphite at the close of the experiment. This is a unique experience, and most probably we may accept Ludwig's explanation as correct, viz.: that the first result is to transform the heated carbon into a solid or liquid non-conducting variety. This cuts off the current, and after a few seconds is itself transformed into graphite as the temperature falls; this enables contact to be re-established and the cycle recurs.

Unfortunately, Dr. Ludwig does not give any data as to the temperature at which this or other phenomena take place, and excuses this omission on the ground of the undoubtedly great, but not invincible, difficulty of making such temperature determinations. It is, nevertheless, stated that the melting point of carbon falls rapidly with rising pressure, and that the non-conducting variety was stable down to a red heat when "under pressure"—presumably about 1,500 atmospheres. The current density necessary to convert carbon into the non-conducting variety varies between 14 and 18 amps per square millimeter sectional area according to the pressure. If a small rod of carbon or a pair of carbon points is heated in an atmosphere of hydrogen at 3,000 atmospheres pressure and allowed to cool, it is found that the resulting graphite does not differ from that formed at 1,500 atmospheres. All attempts to cool the transformed carbon by a jet of hydrogen compressed to 3000 atmospheres, and blowing into the pressure vessel at presumably 1,500 atmospheres) did not result in the formation of diamond, though in this case the graphite crystals were of "regular"

form, and one of them measured 5 mm. at its greatest length. Suddenly discharging the hydrogen from the pressure vessel so as to drop the pressure as quickly as possible only resulted in the formation of graphite.

The most successful attempt was made at a pressure of about 1,200 atmospheres, a carbon rod being arranged in the pressure vessel for resistance heating, and was provided with a lagging of some refractory non-conducting substance. The current is gradually raised till the interruption occurs, and this is allowed to take place two or three times. If no cooling is applied, soft, pure graphite results. If, on the other hand, water at 2,000 to 3,000 atmospheres is squirted in, a product consisting of carbon in non-conducting transparent form is obtained, but Dr. Ludwig does not give the yield.

A fair inference from this work is that carbon can be converted "into the non-conducting variety" at pressures not necessarily exceeding the moderate amount of 1,200 atmospheres, and at temperatures which are known to be quite easily attainable by resistance heating. The difficulty consists in preventing the reconversion into graphite as the temperature or pressure falls—the most rapid cooling attainable apparently only operating successfully on very small fragments of material.

Mr. Threlfall then adds some theoretical considerations of his own which are very interesting and suggestive, though they are essentially based on analogies, not fully proven. He gives the equilibrium diagram of octahedral and monoclinic sulphur, with the pressure in atmospheres and the temperature in degrees C. as coordinates (see, for instance, Van't Hoff's Lectures on Theoretical and Physical Chemistry, translated by Lehfeldt, Vol. I., p. 27). On the basis of certain analogies he then takes this sulphur diagram and writes "diamond" for octahedral sulphur and "graphite" for monoclinic sulphur. This method enables him then, of course, to draw theoretical conclusions as to the possibility of producing artificial diamonds.

He also points out that it should be possible to make vessels which will withstand a pressure of 50 tons per square inch for long-continued pressure, while 75 tons per square inch is given as the practical maximum pressure, if we put the working vessel inside another vessel, also under high pressure; 50 tons per square inch is 7,500 atmospheres—more than double Ludwig's highest pressure; and according to Sir William Crookes, Sir Andrew Noble has reached a pressure of 95 tons per square inch for a fraction of a second. There is, therefore, obviously room and opportunity for an extension of Dr. Ludwig's work by any one who has sufficient time and money at his disposal.

CORRESPONDENCE.

Kryptol.

To the Editor of Electrochemical and Metallurgical Industry:

Sir:—On page 296 of your August issue, Mr. Richard Seligman calls attention to the matter of the alleged composition of "Kryptol." His results are almost identical with our own, although the specimens used for analysis were purchased in different countries.

We are also permitted to say that Mr. Bronn maintains in a private communication to these laboratories, that during the period of his management of the Kryptol Gesselschaft, "only the purest carbon was used in connection with the apparatus for high temperatures." This statement is published in justice to Mr. Bronn.

In view of the facts as disclosed in the several communications that have appeared in your columns, it seems pertinent to repeat our inquiry as to where kryptol of the complex composition claimed may be had?

P. M. N. Bennd.
Fitzgerald & Bennee Laboratories.

Niagara Falls, N. Y.

Detinning.

To the Editor of Electrochemical and Metallurgical Industry:

Sir:—On page 325 of your August issue is printed a description of the detinning process disclosed in United States Patent No. 822,115, granted to Goldschmidt and Weber, with comments thereon.

The claims in this patent cover unimportant modifications of the process of removing tin from tin scrap by a solution of chlorine in an anhydrous liquid, as broadly covered by United States Patent No. 783,725, dated Feb. 28, 1905, granted to Mr. F. von Kugelgen and myself.

The only novelty claimed for the Goldschmidt-Weber disclosure appears to be the compressing of the tin scrap and the variation of pressure inside the detinning vessel, neither of which details properly constitute invention.

It may be interesting to know that, in company with Mr. Chas. E. Acker, we have formed the Tin Products Co., a corporation of the State of New York, for the purpose of exploiting our process above referred to.

George O. Seward,
Holcomb's Rock, Va. *Virginia Electrolytic Co.*

Fluxes in the Cupola.

To the Editor of Electrochemical and Metallurgical Industry:

Sir:—With reference to the paper of Mr. N. W. Shed, read at the Cleveland convention of the American Foundrymen's Association, and abstracted on page 332 of your August issue, we wish to make some remarks. Mr. Shed takes a very strong line against the use of fluorspar as a flux, and it is to this we wish to take exception.

In the first place, fluorspar as a flux, not only in cupola practice, but generally, is much more valuable than limestone. It has the power of liquefying the silicious and gangue minerals to a remarkable extent, to a very much greater extent than limestone, and on that account its use in metallurgy is very rapidly increasing. The production of our mines is now over 50,000 tons per annum, and is steadily increasing.

In the second place, a large number of the founders in this country admit that they do find a reduction of sulphur contents in the iron when fluorspar is used. Its action on sulphur is also borne out by the fact of its extensive use in the basic open-hearth furnace for the elimination of sulphur in the basic steel process.

As to the improvement, or otherwise of iron when fluorspar is used in the cupola, we have evidence of a large number of the best founders in this country that they do find the iron improved. Fluorspar undoubtedly assists in eliminating slag, the slag is thinned, and the metal is kept hotter, consequently sharper castings will be the result. In addition to this the charge in the cupola is brought down more rapidly by the use of fluorspar.

As to the contention that limestone is far cheaper than fluorspar, this, of course, goes without saying, but it should be made clear that the intention is not to use fluorspar alone as a flux, or as a substitute for limestone, but that the fluorspar should be used in conjunction with the limestone, the usual mixture being about 20 pounds per ton of iron melted.

It will be obvious, therefore, as the percentage of fluorspar used is so small, the additional cost of using it is exceedingly trifling, and is far more than outweighed by the many advantages obtained by using the spar, and also probably by the saving in fuel, due to the iron coming down more rapidly.

In conclusion, we cannot do better than refer to the very able papers and articles that have been read and written by Mr. J. S. Gruner, who has pretty conclusively proved the many advantages to be derived by using fluorspar in the cupola.

Geo. G. Blackwell, Sons & Co.

Liverpool, England.

A Metallurgical Revolution in Guanajuato.

BY JOSEPH W. RICHARDS.

One hundred years ago the largest city on the American continent was not New York, it was the City of Mexico, New York was not even the second largest, while third on the list

FIG. 1.—THE HACIENDA DE PURISMA, NEAR MARFIL, GUANAJUATO.

was the mountain city of *Guanajuato,* the most picturesque, historic and interesting city in the Mexican Republic. The Spaniards found a town here in 1534, but in 1548 commenced the wonderful mining development, when two Spanish muleteers discovered the silver mines of La Luz. Since then, by methods which extracted probably only about 50 per cent of the value of the ores, some fifteen hundred millions of dollars of gold and silver have been obtained from these contorted mountains.

Historically, this is the birthplace of Mexican independence. Near here was born the patriot priest Hidalgo, who on Sept. 15, 1810, proclaimed the independence of Mexico, which was finally achieved after eleven years of struggle, during which, however, poor Hidalgo lost his life, and the hook is still shown, on the castle walls, where his head hung for years as a warning to the patriots.

Scenically, the city resembles a town of the hill country of Judea, let us say Jerusalem or Bethlehem. With square, flat-roofed houses, tinted with most of the colors of the rainbow, spreading along several miles of narrow valley and straggling up the steep sides of the mountains, with church towers, plazas and old castello making a Damascus-like *ensemble*, the effect on the visitor is altogether one of striking and absorbing interest.

With but one tortuous principal street, like Mauch Chunk in the Switzerland of Pennsylvania, and mule cars for transit, one is struck dumb with surprise to be confronted, at a sudden turn of the road, with the beautiful Juarez Theater, which cost half a million dollars in gold, and which is architecturally the most beautiful one on the American continent. The city, now counting 40,000 inhabitants, is a strange medley of the old and the new, of the antique and the most modern.

Geologically, the lower parts of the valleys are filled with barren conglomerate of a rich red color, which [illegible] way up the sides of the mountains. The [illegible] the conglomerate in light colored masses of [illegible] a peon's towering straw tile above his enveloping red blanket. The Veta Madre, or mother vein, is plainly traceable across the country for miles, being punctuated by the [illegible] extracted from the workings.

Mineralogically, the vein material is quartz, calcite, with hematite, arsenopyrite, pyrite, some pyrolusite and alabandite (MnS). Gold occurs metallic both free and combined in the pyrites. Silver occurs as argentite (Ag^2S), proustite (Ag^3S^3As), and stephanite (Ag^5SbS^4). Average mine assays show $20 to $40 per ton, while tailings left on the mine dumps show $5 to $10, of which millions of tons are available for cheap treatment.

Metallurgically, the process of the past has been the *patio* process. Scattered along the narrow gulch, from Marfil at its entrance up to the head of the valley above Guanajuato, are the old *haciendas*, each named after some patron saint, but nevertheless provided with moats, drawbridges, double-barred gates and loop-holed towers at the corners. Only one of these is now running, the *Hacienda de Purisma*, near to Marfil.

In this three-century-old metallurgical plant is to be seen the old patio process just as we have seen it pictured by Percy and Schnabel. The Chilean mill with two mules and three workmen, doing about one-third as much work as a Griffin mill with one-sixth an attendant; the seventy *arrastras*, turned by blindfolded mules, grinding each about half a ton of ore per day; the broad *patio*, covered with ore mud 15 inches deep, and churned by the ever patient teams of each a dozen mules, in narrowing and enlarging circles, presided over by a giant peon with a wicked whip, the very impersonation of inquisitorial cruelty.

Besides these were the *lavadores*, or washing tubs, with queer old eighteenth century wooden cog-wheels; the mercury strainers and distilling apparatus, all as quaint as the pictures in Paracelsus' "Re Metallica." Such is the process which is passing away, together with the old methods of hand mining, packing of the ore on burros, and extraction of only half the precious metals in the ore after all.

FIG. 2.—RECEIVING AND SAMPLING FLOOR.

The Hon. Dwight Furness, American Consul at Guanajuato, is the personification of United States enterprise and during his sixteen years residence there has probably done more for the development of this camp than any other one individual. In an address before the Saturday Night Club of Guanajuato, published in the July number of the *Pan-American Magazine*,

he summarized so ably and completely the present metallurgical situation, and described so clearly the metallurgical revolution now taking place, from the patio to the cyanide process, that we cannot do better than extract freely from his address:

"Up to a recent date mining in Guanajuato was carried on under rather difficult conditions. Except for hoisting purposes, machinery was very little used, and even when so used, was very costly, owing to the lack of skilled mechanics and the high cost of fuel and repairs. The excessive cost of lumber

FIG. 8.—THE CHILEAN MILL.

made the timbering of the big circular rock shafts prohibitive, and, therefore, the larger mines used cable guides and hoisted water in galvanized iron buckets, fitted to the cable with ear guides, while the ore was handled in hide sacks, or 'mantas.' In the smaller mines the horse whims, or 'malacates,' were found cheap and nearly as efficient as the steam hoists, both ore and water being hoisted in hide bags without guides. Except when mining out the bonanza ore-chutes, practically all the ore was dumped on the 'patio' on the surface, and hand-sorted at a cost varying from $1.50 to $3.00 per ton. The ore from the stopes was carried on men's backs to the shafts and dumped and rehandled there. Barring the richer bonanza pay streaks, the mining of the ore cost from $15.00 to $18.00 per metric ton. Careful revision of the pay rolls of a number of the leading mines shows that a mine producing an average of 300 tons of ore per week was exceptional, and a mine of this capacity usually had a pay roll of from $4,000 to $5,000 per week. Even in the mines worked during the past ten years, and under comparatively modern conditions, like the Esperanza and Cedro, the cost of mining averaged over $12.00 per ton of ore sold to the mills.

"The mining by the 'patio' process was very efficient in the recovery of silver and equally deficient in gold recovery. There is no question but what a saving of 90 per cent of the silver was usual, while the gold saving varied from 30 to 40 per cent only from the mother-lode, and from 50 to 60 per cent from the mines like Peregrina and La Luz, from which the gold was more easily amalgamated. The cost of treatment varied greatly, being influenced chiefly by the price of quicksilver; the loss of quicksilver rapidly increasing with the grade of the mineral. The cost of treatment, therefore, varied from $12.00 to $17.00 per metric ton. The mills were nearly all located along the Guanajuato River, partly on account of water supply and partly for reasons of security, causing a heavy freight charge on the ores from the mines to the mills, never less than $1.50 per ton, and often reaching $3.50 per ton.

"I remember seeing a very interesting table compiled by Engineer Pablo Orozco, then in charge of the Rul Estate mines. The Concordia Co. had been working the Mellado and Rayas mines for ten years, and had charged up against the mines a shortage, or loss, of nearly $500,000. This loss, however, was less than the amount of freight paid on the shipments of ore from these mines to the mills during the same period. The mines were almost universally worked under the old 'avio' or lease, and all profits, except in time of bonanza, were made in the mills, and in the course of time the mines were looked upon merely as feeders to supply ore to the mills, and worked for that purpose as economically as possible, so that the future of the mines and the exploration work for opening new ore bodies was greatly neglected.

"Free coinage of gold and silver obtained, the price fixed at the mint per kilo of silver was $40.915, from which were deducted Federal and State taxes, leaving net value paid to the mill owner by the mint of $37.68 per kilo. The assaying and melting charges reduced this to about $37.50 per kilo, which was the valuation universally used by the mill owners as net value of their silver. After deducting a treatment charge equivalent to from 500 to 600 grams of silver from the assay, the standard price paid by the mills on the remainder was at the rate of $29.80 per kilo. The gold in the ore was considered of little importance. The assay was made in grains per mark of silver. The average assay on the Veta Madre ores was about 20 grains per mark, of which ten grains for a so-called 'parting charge' were first deducted, in this case equivalent to 50 per cent of assay, the balance being paid for at the rate of 1¼ cents per grain, equal to about $8.00 Mexican currency per Troy ounce.

"The mint figured the gold at $675.416 per kilo, which, after deducting Federal and State taxes, was equivalent to 64.3 cents Mexican gold currency per gram of gold. There was an additional parting charge of $1.25 per kilo gross weight of bullion, making it impossible to part the gold unless assaying over 1½ one-thousandths fine in the bullion, while on the average gold fineness of the 'arrastro' bullion—20 to 30 thousandths—the cost of parting was about 5 cents per gram of gold. The mint paid for the gold with Mexican gold coin, which did not circulate, but was shipped out of the country, remelted and refined, these costs amounting to a little over 3 cents per gram Mexican currency on original gold value. The exchange rate during the greatest activity in the Guanajuato mines was at

FIG. 4.—AN ARRASTRA.

par, but up to recent years ranged from 30 to 40 per cent premium. This made the net value of the gold recovered a little less than 90 cents Mexican currency per gram.

"Taking the foregoing net gold and silver values in bullion and using the most favorable mining and treatment costs at hand, viz.: $12.00 per ton for mining, or a total of $26.00 per ton, the average grade of ore mined and milled shows results as follows. (All figures are in Mexican currency.)

800 grs. Ag 90" equals 720 grs. at $37.50 equals	$27.00
5 grs. Au 40 per cent equals 2 grs. at $0.90 equals	1.80
Total recovery	$28.80

Mining		$12.00
Milling		12.00
Freight		2.00
		$26.00
Net profit per metric ton		$2.80

"Under the foregoing conditions the decadence of mining in Guanajuato is readily understood.

"During the past three years a wonderful transformation has taken place. First, a shrewd, level-headed lawyer from Colorado Springs visited the camp on legal business for some clients. The proverbial—and in this case we might say Providential—slowness of legal matters in Mexico left him with

FIG. 5.—THE PATIO—GENERAL VIEW.

ample leisure. He cast his eyes over the brown bare hills, denuded long since of fuel, over the vast dumps and masonry walls, ruins of a glorious industry in a mining camp that held the world's record. He stepped aside in the streets to let pass the long trains of mules conveying in a crude way the ores from mines to mills, and he visited the 'patio' mills and found dozens of men and hundreds of mules treating the ores in a most laborious and primitive fashion. Looking on these things he was convinced that if great things could be accomplished under such conditions, even greater things could be accomplished by modern methods, and he had the courage of his convictions.

"Cheap power he rightly conceived to be the first essential, and he found at hand two hustling, shrewd young men, with more brains than capital, and through them, at a hundred miles away, the power was found. We old-timers smiled indulgently. We had seen a dozen American companies try to revolutionize Guanajuato methods and miserably fail. We signed contracts for the power because it didn't cost anything to sign them and couldn't do any harm. The first electric shock came a year later. The power was here and clamoring for payment, and only one company ready to use it.

"In the meanwhile, one of our pioneer American operators had been struggling along with one of the big old mines of the Veta Madre. He had developed large bodies of low-grade ores, but their values were absorbed by costly and inefficient milling processes. Then began a long series of costly experiments, which indicated that the Guanajuato silver ores could be successfully treated by the cyanide process, something heretofore not deemed possible. But capital is timid and wants something more than the work in laboratory or experimental plant,

FIG. 6.—THE PATIO—NEARER VIEW.

and not until this persevering and courageous man built a 150-ton per day cyanide plant—the first of its kind in Mexico—and demonstrated on this large working scale the absolute practicability of this process, were the doubters and scoffers finally convinced. To-day all the companies are erecting these plants.

"The next important stage of development was the placing of the mills in close proximity of the mines, avoiding all freight charges on the ores, and at the same time opening up and

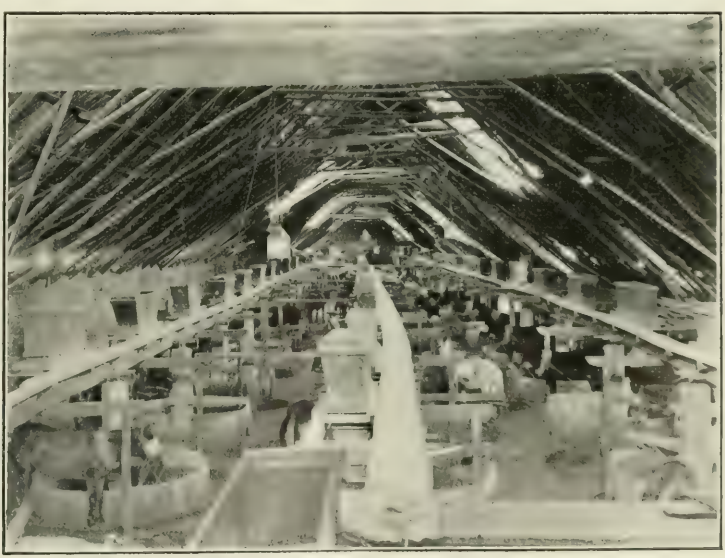


FIG. 7.—THE ARRASTRA MILL.

equipping the mines with mechanical contrivances for the rapid and cheap handling of large tonnage. The result of these changes is comprehensively shown by the following figures, taking the same assay values used in our first table, silver at 61½ cents, U. S. currency, per ounce and exchange at 200:

800 grs. silver 85 per cent equals 680 grs. at $37.50 equals		$25.50
5 grs. gold 90 per cent equals 4.5 grs. at $1.25 equals		5.62
		$31.12
Cost of mining	$4.50	
Cost of milling	4.00	
Administration	0.50	
		$9.00
Profit		$22.12

as against $2.80 under the old conditions. These recoveries of gold and silver, as well as costs of mining and treatment, are considered as average costs for the whole district, and are conservative. They are already bettered in several mines and mills, and no doubt some especially favorable ores will show a saving of 90 per cent of the silver and 95 per cent of the gold. Mining and milling costs will also in some cases be reduced to as low as $7.00 Mexican currency per ton. It requires no great effort to grasp the significance of these figures, but it does require a lively imagination to awake to the possibilities they convey.

"Nine dollars Mexican currency per ton from the mine's deepest workings to the bullion market. This means that ores assaying only 300 grams of silver can be mined and milled at a small profit; that ores containing only 10 grams of gold leave a profit of over $2; that Veta Madre ore assaying 400 grams silver and 2 grams gold leave a profit of $6 per ton. Recent developments in the larger mines seem to make it clear that nothing less than this value need ever be mined, and the indications are that the values saved in the larger mills will be fully $20.00 Mexican currency per ton.

"The awakening is already at hand. At the beginning of the year a careful tabulation showed that the approximate monthly

FIG. 8.—THE RETORTING PLANT.

expenditures of the mining companies of this district were only $80,000. By the close of the present year it will be over $200,000, and estimating only the additional projected work it will, by the end of the year 1909, reach a total of $400,000 monthly.

"Another comparison. On Jan. 1, 1905, there were only ninety stamps dropping, and inclusive of 'patio' mills the total milling capacity was only 300 tons of ore per day.

"At the close of 1905 there will be 200 stamps with a capacity of 600 tons per day—over 200,000 tons per year, producing a gross value of $20 per ton, equal to $4,000,000 per year. The next year's developments promise to be far more startling. Counting only the mills in course of erection, those already ordered and those whose erection during the coming year are practically assured, we have a total of nearly 700 stamps, a capacity of 2,000 tons per day, or 700,000 tons of ore per year, and to which corresponds a gross production of $14,000,000, with a net profit of at least $5,000,000 yearly. This production is one-half more than was produced in Guanajuato in her most

FIG. 9.—THE AMALGAM ROOM.

flourishing bonanza days. In the above figures I have not included the mining districts of Pozos, San Felipe or San Anton de las Minas—all important camps of this State and of great promise.

"But this is not all. Nothing succeeds like success, and capital rushes in when the pioneers have scored their first victories. The probabilities of the succeeding years are bewildering. It is certainly safe to say that the production for 1908 will be 50 per cent greater than that of 1907, and this on the low-grade ores only, which were until recently considered waste."

Manufacture of Carborundum in Europe.

For quite some time past the Carborundum Co., of Niagara Falls, intended to erect a plant in Continental Europe, to meet more directly the rapid increase in the demand of carborundum for grinding purposes in Europe. The European plant will now become a reality, the works being expected to open with the beginning of next year.

The official name of the German company will be the Deutsche Carborundum Werke, Ges. mit beschr. Haftpflicht. The plant will be located at Duesseldorf, in Rhineland, Germany. The engineers of the American company are to supervise the erection of the German plant, which will be equipped with the most up-to-date machinery for the production of abrasives.

While the Carborundum Co. of this country has been represented in the past in the principal cities of Germany, Great Britain, Denmark, Sweden, Norway and Russia, with stores in Berlin and London, it is evident that the completion of the German works will enable the company to push their European business with much greater energy.

The splendid success of the Carborundum Co., of Niagara Falls—evidenced by the rapid increase of the works—is well known to our readers. An illustrated description of the works as they were in 1902 may be found in our October issue of 1902. According to Mr. J. H. Pratt, of the United States Geological Survey, the production of carborundum in 1905 amounted to 5,596,000 pounds, the value varying from 7 to 10 cents per pound. Besides the very extended use of carborundum for abrasive purposes, it is now also finding very great favor for furnace linings in various cases.

JOINT MEETING OF THE IRON AND STEEL INSTITUTE, AND THE AMERICAN INSTITUTE OF MINING ENGINEERS.

The joint meeting of the Iron and Steel Institute and of the American Institute of Mining Engineers was held in London from July 23 to 28. Over fifty members of the American Institute, including the president, Mr. Robert Hunt, and the secretary, Dr. R. W. Raymond, attended the meeting.

Some notes on the proceedings of the first session on July 24 will be found in the regular monthly letter of our London correspondent on another page of this issue. It may be added here that at the suggestion of President Hadfield, of the Iron and Steel Institute, the following cablegram was sent to Mr John Fritz, in Bethlehem: "This great and memorable gathering of Englishmen and Americans, united, send you, Bessemer Medallist and Honorary Member of the Iron and Steel Institute, heartiest and most friendly greetings. We congratulate you upon the immense progress in metallurgy made during your life time, and in which you have played so honorable a part. Long may you be spared to us."

Mr. Hadfield announced that Sir Hugh Bell had been elected to succeed him as president of the Iron and Steel Institute on his retirement next year. The three papers presented during the first session dealt with large blast furnace gas engines. Abstracts will be found below.

The second session, on July 25, was devoted to a general meeting of the American Institute of Mining Engineers. Mr. Hunt delivered his presidential address on the development of steel rail mills in the United States. He also announced that Mr. John Edward Stead and Mr. R. A. Hadfield had been elected honorary members of the American Institute of Mining Engineers.

Then followed a long discussion, introduced by Mr. A. L. Colby, on "a comparison of American and foreign rail specifications, with a proposed standard specification to cover American rails rolled for export."

In the session of July 26 a paper by Mr. J. P. Roe on his mechanical puddling process, and a paper by Mr. J. E. York on improvements in rolling iron and steel, giving a description of the York universal rolling mill, were presented.

The afternoons were devoted to numerous excursions. At noon on July 27 the King received a deputation from the Iron and Steel Institute and from the American Institute of Mining Engineers at Buckingham Palace. The American Institute was represented by Captain Hunt, Dr. Raymond and Messrs. Dwight, Fackenthal, Hartshorne, Kirchhoff, Oberlin, Smith and Ward. Mr. Hadfield presented the Americans, and asked the King, on behalf of the Iron and Steel Institute, to receive the Bessemer gold medal (which had also been held by the late Queen).

After the conclusion of the London meeting the American visitors made a tour through England, and visited the Rhineland-Westphalia iron district on the invitation of the Society of German Iron Masters.

In the following we give concise abstracts of all papers which had been printed in advance, from advance copies kindly sent to us from the secretary of the Iron and Steel Institute:

Dry Blast.

A paper by Mr. J. E. Johnson, Jr. (Longdale, Va.), discussed "different modes of blast refrigeration and their power requirements." The object of the author is to give data for determining the plant cost and operating cost of the blast-refrigerating apparatus under any given conditions, and to point out means whereby this refrigeration may be accomplished with less first cost and less working expenses per cubic foot of blast treated than were required by Mr. Gayley's installation.

Some parts of Mr Johnson's paper are highly theoretical, but the following abstract is restricted to its general argument and his practical conclusions, and when necessary his mathematical analysis shall be translated into metallurgical argument. The standard temperature at which blast is measured is taken by the author as 70° F., at which temperature a cubic foot of air weighs .075 pounds, therefore all calculations are based on 75 pounds of air.

First, the "direct-expansion" system is compared with the "brine-circulation system." In the former the ammonia is expanded in the coils over which the air passes, in the latter the ammonia is expanded in coils immersed in a tank of brine, which is afterwards circulated through coils in the refrigeration chamber. It is obvious that the brine-circulation system requires twice the expense of coils or pipe surface, and also requires twice the temperature-interval between the temperature of the expanding ammonia and that of the air required by the direct-expansion system, since the heat requires to be transmitted through the walls of two sets of pipes instead of one. This means that "the substitution of the direct-expansion for the brine-circulation system leads to important economies in first cost, in power required, and in attendance, not offset by any necessary advantage on the side of the brine-circulation system."

Further, let us assume certain data corresponding to frequent summer conditions, namely, a temperature of 85° F. and a dew-point of 70° F., a condenser temperature of 85° F., and let the air be refrigerated down to 25° F. It is plain that the air leaving the refrigerating chamber at a temperature of 25° F. may be used for cooling the incoming air, with a saving of power to the incoming air. Therefore, "the regenerative system leads to important economies in first cost, only in minor degree offset by a loss of advantage to the blowing engine."

Now, concerning the refrigeration itself. Of all the moisture contained in the air at 70° F., half is precipitated when the temperature is lowered to 50° F., and of that removable down to 25° F. almost two-thirds. Yet the heat of all this is removed at the temperature of 25° F., if the one-stage refrigeration method is used. "This is precisely as if, in pumping from a shaft 45 feet deep, we put one pump at the bottom and let all the water run down to that level before pumping it out. It is really much worse than this, because in pumping heat the power required increases much faster than the heat, while with water this is not so."

"The remedy is as obvious in refrigeration as it is in pumping, to have two pumps catch the greater part of the heat near the top of its scale and pump it out from there, pumping from the lower level only the heat which comes in at or near that level, a much smaller quantity. This means the use of two ammonia compressor cylinders, one working from the 15° F. suction temperature as before, the other working from a much higher suction temperature, say, 36° F., both, of course working to the same condenser temperature." This two-stage method of refrigeration leads to important economies in first cost, in power requirements, and in absence of trouble with ice on the coils of the first stage not offset by any disadvantage worth considering.

By a combination of these methods (direct-expansion, double-stage, regenerative system), as compared with the brine-circulation, single-stage, non-regenerative system the author concludes that it is possible to dry the blast with less than half the first cost of plant, with much less attendance and with one-third the power.

Finally, as to the minimum temperature, the author thinks

that from a commercial point of view there is no gain in refrigerating the air to too low a temperature. In summer, all the uniformity possible will be reached by refrigerating to 32° F., and all trouble from ice formation will be avoided by not going below this point. As the weather becomes colder and the dew-point falls so as to be sometimes below this point, the diminished refrigeration required and the lower temperature of cooling water available, will enable the same plant to maintain a lower temperature, and still retain the uniformity desired by reducing the temperature of the air below the lowest natural dew-point probable. This change being entirely under control of the manager will lead to no sudden changes, and will give a coke consumption in summer so nearly the same as that in winter that the difference will probably be inappreciable. As between a refrigeration temperature of 32° F. and one of 22° F., with a blast temperature of 1000° F. and normal coke practice, the saving for the latter will be about 2 per cent.

Electric Induction Furnace.

A paper by E. C. Ibbotson (Sheffield) dealt with the Kjellin electric steel furnace, which has already been the subject of numerous articles in this journal.

At Gysinge, in Sweden, during the year ending May 31, 1906, from a fixed furnace giving 1 ton (2240 pounds) of steel per tap, there were produced 950 tons of tool steel and special steel ingots.

In carbon and iron tool steels all the usual tempers were made. The bulk of this steel was made from charges composed of about 80 per cent of Swedish pig iron and 20 per cent of steel scrap. The percentage of carbon was regulated by the addition of briquettes. Other charges were made from Swedish white iron and steel scrap.

The average time taken per charge, for the year, when adding briquettes, was 7⅛ hours, and the electrical energy consumed was 1,128 kw-hours per ton. The average time per charge for white iron and scrap charges was 5½ hours, and the electrical energy consumed was 880 kw-hours per ton. These consumptions include all time and energy lost from various causes, such as bad water supply, ice, etc.

It is quite ordinary practice to complete the charges with briquettes in 6½ hours, and without briquettes in 5 hours, as shown by the following typical charges:

White pig iron, 1,457 pounds; steel scrap, 439 pounds; briquettes, 220 pounds; ferro-silicon (50 per cent Si), 17 pounds; ferro-manganese (80 per cent Mn), 15 pounds; aluminium, 1 ounce.

The times, strength of current and energy consumed for this charge were as follows:

Time.	*Kw.*	*Kw-hours.*	
5.30	125	....	Charging two-thirds of the pig iron.
6.00	145	67.50	
6.30	160	76.25	
7.00	170	82.50	Charging the remaining one-third of the pig and the scrap.
7.30	170	85.00	
8.00	165	83.75	Clear melted.
8.30	165	82.50	Briquettes added.
9.00	165	82.50	
9.30	165	82.50	Briquettes added
10.00	165	82.50	
10.30	165	82.50	Briquettes added.
11.00	165	82.50	
11.30	165	82.50	
12.00	130	73.75	Ferro-silicon and ferro-manganese added. Tapped.
6½ h.	....	1046.25	

A second charge was made as follows:

Steel scrap, 1,372 pounds; white iron, 914 pounds; ferro-silicon, 4.5 pounds; ferro-manganese, 6.5 pounds; aluminium, 1 ounce.

The time was reduced, no briquettes being added. The results were as follows:

Time.	*Kw.*	*Kw-hours.*	
7.00	125	....	Charging all pig and one-half scrap.
7.30	145	67.50	
8.00	155	75.00	
8.30	160	78.75	Charging scrap.
9.00	165	81.25	
9.30	170	83.75	
10.00	165	83.75	Clear melted.
10.30	165	82.50	
11.00	165	82.50	
11.30	165	82.50	
12.00	135	75.50	Ferro-silicon and ferro-manganese added. Tapped.
5 h.	....	793.00	

The chemical composition of the materials charged and of the resulting steel were as follows:

White iron (Herräng): Total C. 4.00 per cent, Si 0.15, Mn 0.18, S 0.010, P 0.012.

Briquettes: Fe 59.0, S 0.010, P 0.006, SiO_2 11.00, CaO 2.50, Al_2O_3 0.50

Steel ingots: C 0.40 to 2.00, Si 0.12, Mn 0.34, S 0.012, P 0.014.

The lining of this furnace during the year was magnesite. When using briquettes the lining lasts, on an average, five weeks, and when charges are worked without briquettes the average life of the lining is seven weeks.

Various tests have been made of the steels produced in this furnace. During the year satisfactory results obtained in works practice have been reported, particularly in connection with the following steels: stamping dies, punches, cold chisels, screwing dies and taps, cutlery, drills and turning tools. The following special steels have also been produced: Tungsten

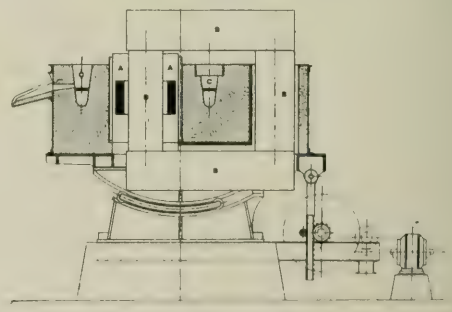


Fig. 1.—Electric Induction Furnace.

steel (including permanent magnet steel), chromium steel, nickel steel, nickel-chromium steel, self-hardening steel, and high-speed tool steel. High-speed tool steel, tested to one of the latest government specifications, gave satisfactory results, while 25 per cent unannealed 2-inch nickel steel bar, ½ inch in diameter, gave a yield point of 30.68 tons per square inch, a maximum stress of 50.44 tons per square inch, an elongation of 44.00 per cent, and a reduction of area of 60.00 per cent.

Electrical tests carried out have shown that No. 6 gauge rods had a specific resistance of 36 michroms per cubic inch; No. 16 gauge wire a specific resistance of 33.5 microhms per cubic inch.

The form of furnace adopted and recently erected is shown in Fig. 1. A is the primary coil, B iron coil, and C the bath

(secondary circuit). The advantages of this furnace are that it can be tipped over the lip, or from the tap-hole, and can easily be repaired and relined when necessary.

Silicon and Graphite in the Open-Hearth Process.

A paper by Alex. S. Thomas (Cardiff) discusses "the influence of silicon and graphite on the open-hearth process." The first part of the paper deals with a detailed discussion of the effects of using a high or low silicon pig. The author's experience is that when using pig with silicon of 2 per cent and over, the banks are seldom, if ever, fluxed, but that bad bottoms very often occur if the furnace man lags behind and does not add his oxides quickly enough to keep his slag from becoming viscous. When using pig with 1 per cent silicon or less, the banks are invariably fluxed, but the bottom of the furnace very seldom comes up. Therefore, in the acid open-hearth process a middle course must be steered by using an iron with the silicon neither very high nor yet very low, and the author's experience is that a silicon content from 1.25 to 2 per cent is an ideal one for the acid open-hearth melter.

There is no doubt that the up-to-date iron manufacturer is handicapped when making high silicon iron (above 1¼ per cent), and the acid steel maker is also handicapped when using iron under 1.25 per cent. To bring the steel maker into line with the iron maker, the author suggests that in furnaces making acid steel with low silicon iron, the acid bottoms now used should be replaced by basic bottoms, and the low silicon haematite iron worked on the basic bottom.

This could not be called a basic process, the essential aim and feature of the basic process (dephosphorization) being absent, and only enough lime (3 to 5 per cent) need be added to render the slag just sufficiently basic to prevent the banks and bottoms being fluxed. It would be an acid process on a basic bottom, and the quality of the resulting steel would be superior to the average quality made on acid bottoms. The output would also be increased 15 or 20 per cent, as so much more steel scrap could then be used in the charge.

The second part of Mr. Thomas' paper deals with the effect of carbon in the graphitic form, with special reference to the Talbot process. In a 160-ton furnace where haematite iron is used, silicon—contrary to one's expectation—is not a trouble to the same extent as graphite. In general, a high percentage of silicon in the iron is accompanied by a high percentage of graphite, so that high silicon is a disadvantage only because it is accompanied by a separation of the carbon in the graphitic form.

When a charge if iron containing a high percentage of graphite is worked, the whole bath in the furnace is covered with a layer of "kish," and sand shovels can be put into the furnace and the carbon actually ladled off the surface of the bath. When the iron is of this nature, the whole bath lies dormant until the carbon is partly burnt off and partly converted into the combined state. Further—whereas the average time taken in pouring a ladle of low-graphite iron into the furnace is only about 30 minutes—it takes 1½ to 2 hours to pour in an equal quantity of high graphitic iron. In this case no reaction takes place for some time, so that only 3 to 5 tons are poured in at a time. But when the reaction takes place, from 10 to 30 minutes afterwards, it is extremely sudden and violent. A few more tons of iron are then poured in, and usually another long wait ensues until the oxides and metalloids react. It is this waiting, interrupting the continuity of the process, lowering the output, and causing wear and tear of the furnace and a higher consumption of fuel, etc., which renders the presence of graphite objectionable.

Usually after emptying the ladle of high graphitic iron, the longest spell of inactivity takes place; this may last as long as 1 to 1½ hours before the bath begins to boil. The author has found that when iron with a high percentage of combined carbon was supplied, instead of highly graphitic iron, the output was immediately increased 25 per cent, and the wear and tear on the furnace was reduced [illegible]

"The foregoing applies to the effect of graphite [illegible] molten iron. But does not the same [illegible] in the ordinary process of charging cold material? When using pig iron containing a high percentage of silicon, is not the longer time taken in working such a charge—generally attributed to the high percentage of silicon—really due to the high percentage of graphite present? Why do some charges, using equal quantities of iron, with the same percentages of silicon, and practically the same percentages of total carbon, take so much longer to decarburize than others, under similar working conditions of heat, etc.? During the [illegible] the percentage of carbon when in the combined state is greatly reduced by being partly oxidized in the oxidizing [illegible] in the furnace, but when carbon is present mostly in the graphitic state, this reduction does not take place, as the carbon is being converted into the combined form, so that when the charge is melted a greater percentage of carbon is present to be decarburized by the oxides added. When using high graphitic iron, the author finds that less than 10 per cent of the estimated carbon contents are eliminated during the melting period. This differs greatly from the 30 to 45 per cent given by Harbord and Campbell in their books."

Treatment of Fine Ores.

A paper by Albert Ladd Colby (New York City) discusses "the nodulizing and desulphurization of fine iron ores and pyrites cinder." The author describes a process which has been in operation continuously since the Summer of 1905, and has successfully and economically converted over 24,000 tons of fine ferruginous ores and by-products into "nodules," which have been smelted in blast furnaces without the difficulties accompanying the use of large proportions of fine materials, and has, furthermore, economically reduced the sulphur present in objectionable quantities in the fine materials.

The invention consists mainly in purifying finely divided metalliferous materials, and forming them into nodules or lumps, by adding to the materials a binder, preferably reducing

Fig. 2.—Unloading at Newark Plant.

in character, adhesive at low temperatures (approximately below 600° F.), and volatile at [illegible] temperatures (approximately 1,200° F., which binder has no affinity for, and is capable of forming volatile compounds with such impurities as sulphur, arsenic, etc.

In practice, any adhesive hydrocarbon or carbohydrate compound such as tar, pitch, asphalt, petroleum residuums, dextrine, molasses, glucose, etc., may be used, provided it can be made liquid or plastic at comparatively low temperatures and can be readily and cheaply obtained.

The adhesive binder drops, or is sprayed, constantly into

the stream of fine ferruginous material as it enters the upper end of an inclined rotary kiln; the material is therefore at once cohered into masses, which by the rotary action break up into nodules or lumps. One per cent or less of an adhesive binder, such as tar, is sufficient.

When the lumps reach a zone of higher temperature the tar combines with the sulphur and both are volatilized, the binder acting as a fuel as well. When the nodules approach the lower and hottest end of the kiln (approximately 2,000° F.) incipient fusion takes place, so that the product is a nodule, permanently cohered without the presence of any extraneous binder reducing the percentage of iron or other valuable

Fig. 3.—Unloading at Newark Plant.

metallic constituents contained in the original fine material.

The process has already been successfully applied to a variety of fine materials fusing between a wide range of temperature, such as pyrites cinder, crude magnetite, magnetic concentrates, haematites (Mesaba), blast furnace flue dust, and franklinite (an iron-manganese-zinc ore).

The patents are now owned by the National Metallurgic Co. of New Jersey, which has erected a large plant at Newark, N. J. Other plants, using the process, are located at Aspinwall, near Pittsburg, Pa., at Steelton, Pa., at Hazard, Pa., and at Oxford, Pa. All these plants are described in the paper.

Several economic devices installed in these plants are, on account of their ingenuity, worthy of notice.

The appliances for the handling of the pyrites cinder, fine ores or coal, whether delivered to the Newark plant by water or by rail, are shown by Figs. 2 and 3.

At the Newark plant, dry bituminous coal is burned with a forced draught, the flame or flames entering the lower or discharge end of the kiln. Among the factors which cheapen

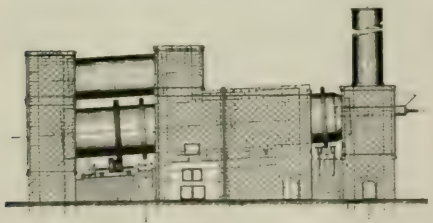


Fig. 4.—Coal-Dryer Plant.

the use of the pulverized coal is a coal dryer of special design and an automatic speeder and screw conveyor, with the accompanying storage bin for the dry pulverized coal, shown in Figs. 4 and 5.

With these devices the coal is used economically; an automatic and accurate record of the coal consumption is kept, and a perfect combustion maintained between the wide ranges of kiln temperatures used in nodulizing different materials.

At the Pittsburg plant natural gas will be used; in the plants

Fig. 5.—Automatic Devices at Newark Plant.

erecting, or in contemplation, other fuels, solid, liquid or gaseous, may be found more economical.

An automatic plunger feed, shown in Fig. 6, of very simple construction ensures a steady delivery, into the kiln, of any desired amount per hour of the material to be nodulized, independent of its fineness, ore the amount of moisture it may contain. Uniformity in desulphurization can be thus assured by changing the rate of feed and the quantity of tar entering the kiln, when using materials containing different sulphur percentages, and also by regulating the speed at which the kiln is revolved.

The storage tank, of 150-ton capacity, into which the hot nodules are delivered from the kiln by a link belt conveyor, is so arranged that the large steel trucks can be filled from the

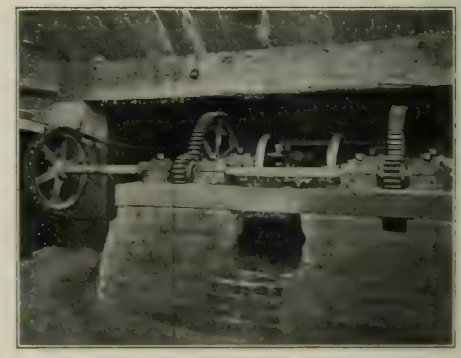


Fig. 6.—Automatic Devices at Newark Plant.

tank by the operation of a single lever, and if for any reason they are not at hand, the nodules, after the storage tank is full, are delivered on stock by an overhead shoot, into which the same link belt conveyor can also discharge. This is shown in Fig. 7.

The general construction of the kilns used in nodulizing may be seen by Fig. 8, representing kiln No. 3 of the Newark plant blocked up, and ready to be put in position.

During the development of the mines, supplying the Newark plant, it became evident that the pyrites, as shipped, would contain copper in paying quantities. The addition to the Newark plant of a modernized Henderson copper-leaching

plant was therefore decided upon. The most interesting feature of this plant is the adoption of the rotary kiln for the roasting of the ground mixture of copper pyrites, cinder and salt. This, it is expected, will prove a great economy over the styles of furnaces formerly used, for the kiln and its accessories are automatic in their operation, and the labor, especially in handling and rabbling the mixture, is greatly reduced.

The mixture will be handled by a belt conveyor from the grinding mills to the storage tank, from which, by a bucket elevator, it is fed automatically through a screw feed into the kiln.

Eventually the moist ore, free from copper, will be transferred by the overhead bucket crane to small tipping wagons that will be emptied into the storage tank, from which it will be delivered to the nodulizing kiln by the automatic plunger feed device, receiving its proper portion of tar as it enters the kiln.

Copper pyrites cinder, con- with the additional one of a sudden arrest at each end, by which the bath is forced through itself, and greatly increased thoroughness of mixture is obtained. This arrangement also utilizes the retardation of the lower stratum of the iron by friction on the bottom, producing the effect of [illegible]

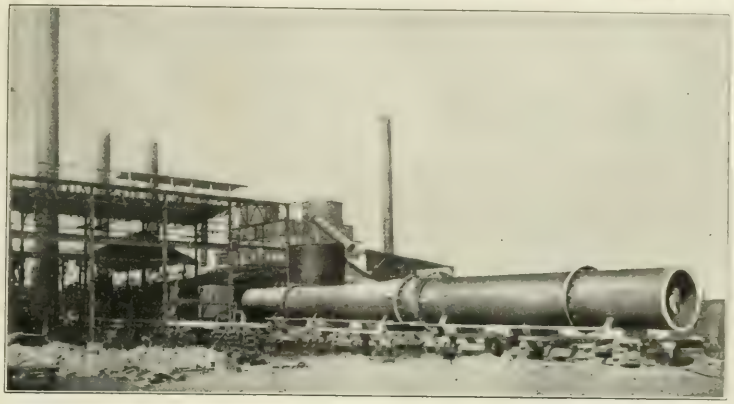


FIG. 8.—CONSTRUCTION OF NODULIZING KILN.

as low as 1.50 per cent of copper, can be handled with a profit in this modernized Henderson plant.,

MECHANICAL PUDDLING PROCESS.

A paper by JAMES P. ROE (Pottstown, Pa.) discourses the development of the Roe puddling process. The paper begins with a concise description of the ordinary puddling process; the author emphasizes that the arduous character of the work in puddling and its consequent high cost suggest the introduction of mechanical means. The author also felt that mechanical means should be associated with large units and the use of metal direct from the blast furnace; that is, that the same broad means that have made possible the low costs in the modern production of steel should be utilized as far as possible.

The author decided that the best means of agitation would be that of simply flowing by gravitation in a long, inclined

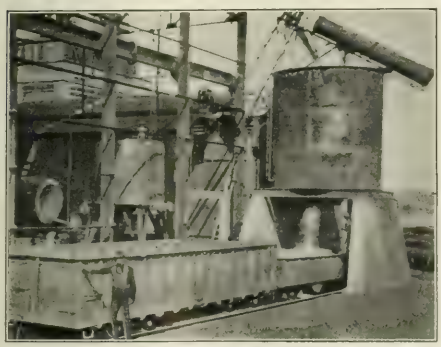


FIG. 7.—STORAGE TANK.

trough, but this idea was impracticable on account of the unreasonably great length which would have been required. The author then modified this idea by providing a trough of considerable length (but less than before), down which the bath could flow, first in one direction and then in the other. This possesses all the advantages of the long, fixed trough, together beach, while at the same time correcting it before it becomes excessive. In the first construction the trough was made 28 feet long.

Two long girders formed the sub-structure, upon which all the moving parts rested, and were suspended from two hollow trunnions, placed opposite each other at the middle of the length. The flames from two stationary coal fire-chambers passed through the trunnions into the furnace and out of the stacks at each end. The trough, forming the bottom, was made of steel plates, and was double, water passing through it to preserve the covering.

It was found necessary to modify this construction in various details, but the net result of this first series of experiments was that they proved conclusively that iron (coming in many heats directly from the blast furnace) could be puddled by causing a mixed bath of molten metal and slag to flow backwards and forwards in a heated trough, when the bath was arrested with sufficient suddenness at the end of each oscillation, and that the puddled iron could be massed into a compact ball by continuing this operation, and could then be shot out ready for further manipulation.

In deciding the question how to dispose of the puddle ball after it was made, the author concluded that in order to carry out the idea of using large units, the mass must be maintained in its integrity up to the point that is reached in modern steel practice, that is, up to the slab or billet. The puddling machine, therefore, must have the capacity to produce a ball which would approximate the weight of a steel ingot. The ball must be squeezed in one mass, rolled on a blooming mill, and cut to lengths in slabs or billets. When this was accomplished the low costs of the modern steel plant should be attainable.

The path marked out, therefore, was the use of molten metal from the blast furnace and mixer, puddling in a machine to be further developed in detail, squeezing in a machine to be designed, and rolling in a blooming mill to slabs or billets, or in a plate mill, directly to plates. By this means, there would be at least avoided the greater cost of handling [illegible] puddle bar. It will be noted that the course of procedure above noted is precisely that of steel making, mechanical puddling and squeezing being substituted for converting and casting into ingots.

The new machine, shown in prospective view in Fig. 9, has the side plates of the trough extended downwards, to form two

semi-circular racks, which gear into pinions on a countershaft. The four stacks are so placed as to produce as even a distribution of the flame and temperature in the furnace as possible. The door, forming the whole of one end, is operated by means of an hydraulic cylinder, placed horizontally under the furnace. The door is firmly locked by means of wedges, which pass through the side plates and are actuated by two small hydraulic cylinders. The sill and lintel of the door are water-cooled in order to chill the molten cinder when it strikes them, and thus to seal rapidly the joint and prevent the egress of iron.

The trunnions carry the whole weight of the machine, and are supported on roller bearings. The machine itself is so balanced that it has a slight tendency to always return to a

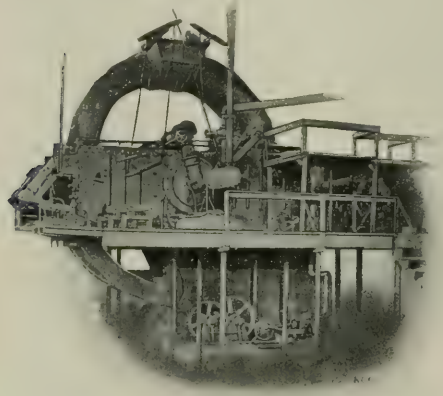


FIG. 9.—MECHANICAL PUDDLING MACHINE.

horizontal position. As a consequence, although the machine is heavy, but little power is required to operate it. The heating agents are admitted through the trunnions. In the present case the author uses oil as fuel.

The molten iron and cinder are brought to the machine in ladles, and are raised on a hoist, from which they pour directly into the machine near the end opposite to the door. There are seven observation holes, through one of which, near the door, the oxidizing agents are charged by means of a long spoon reaching across the furnace. There is also a tapping hole in the rear end, which is used for bleeding, if desired.

The machine is intended to move through an arc of 140°, and the 4,000-pound ball strikes the ends with considerable violence. The bricks, as well as everything else, had to be so secured as to meet these trying conditions of strain and shock in addition to the ordinary strains due to heat.

Coincident with the building of the new machine, a squeezer was built, operated by hydraulic power, applied by cylinders acting in three directions.

One of the important features in the Roe puddling process is the abandonment of all fettling, and the charging of all cinder and oxidizing agents. An auxiliary furnace for melting the cinder was, therefore, a necessary addition to the plant.

The author describes at length the troubles which he had to overcome in his work and which were mainly due to the use of part of an old Bessemer plant. Considerable trouble was also experienced with the design and construction of the working bottom, which were changed repeatedly.

The most important feature is the production and maintenance of the proper cinder, both chemically and physically. The conditions being the same in this respect as in the ordinary puddling process, the same kind of cinder is required. It is produced and regulated practically in the same way, except that, since there is no fettling to act as an automatic regulator, the proper composition must be effected and maintained by direct additions. The cinder which is charged just before the melted iron consists generally of remelted puddle tap cinder, although iron ore or any other oxide of about the same composition acts equally well. It is not strongly oxidizing in its effect, but fulfils all the other functions of ordinary puddling cinder. The oxidation of the metalloids is brought about by the addition of roll scale or any other similarly pure oxide of iron. The proper composition and condition of the cinder is also controlled by the regulation of the flame, as in the ordinary process.

The dephosphorization is chiefly determined by the amount of silica present in the cinder. It was definitely settled that with 25 per cent of silica in the final cinder the dephosphorization was practically nil, while with 12 per cent of silica in the cinder the phosphorus was reduced to 0.06 per cent in the iron, regardless of the amount originally in the pig, provided that the phosphorus in the cinder did not amount to much over 5 per cent. The amount of phosphorus in the finished iron can, therefore, be controlled very closely. The thorough elimination of sulphur was also completely established, but it was not settled that it could be as easily and completely controlled as the phosphorus.

The experiments of the author demonstrated in a general way the entire technical and commercial practicability of the process: "It can safely be affirmed that" in a well-designed plant with continuous running the commercial costs "would not exceed those of a well-equipped steel works." The iron is more thoroughly worked than in the ordinary puddling process, and the metalloids can be more completely eliminated. This permits either of starting with inferior pig irons or the production of better material from the same grades, thus widely extending the range of material entering into the blast furnace or produced from it.

[There was a long discussion of the paper, opened by Dr. Raymond, who referred to the fact that in recent years iron is being again favored to a greater extent, owing to the smaller degree of corrosion, compared with steel. Mr. A. Sahlin, Mr. Joseph Hartshorne and Mr. B. Talbot had seen the process in operation, and bore testimony to its practicability. Mr. Talbot thought the greatest difficulty would be to get suitable iron; in order to get a regular product, a mixer was desirable. Prof. Banerman and Prof. T. Turner questioned whether it was desirable to produce a ball of such a size as is done by the author. A smaller mass might give better results.]

Machine Moulding Practice.

A paper by Ph. Bonvillain (Paris) discusses "recent processes in machine moulding practice." After a historical introduction the author discusses present-day practice and describes in detail his designs of a set of universal moulding machinery, which adapts itself to almost all the requirements of foundry practice, no matter how intricate the castings to mould from it may be. The machine is essentially a combination of a moulding press and of a parting-off machine; *i. e.*, a machine to lift the finished mould from the pattern plate in a few seconds, without any special regard to the height of the casting.

High-Speed Steels.

Dr. H. C. H. Carpenter presented a report on "tempering and cutting tests of high-speed steels," made in the (British) National Physical Laboratory with nine different steels.

The research was undertaken in order to ascertain whether the temperature at which high speed steels soften can be pushed higher than 700° C. The only steel with which any success was reached was a sample which is not a recognized steel of this type. This alloy appears to resist softening by thermal influences even at 800° C., and if its behavior were judged from this standpoint only, better results might have been expected than have been actually obtained.

But it must be remembered that the tempering of such tools in practice is produced by a combination of mechanical stress and high temperature. The question has yet to be answered whether the influence of the former factor is direct or indirect. To the author it appears probable that its influence is indirect; *i. e.*, that the mechanical stress to which the tool is subjected in practice does not of itself tend to soften it, but rather the reverse, and that it is the heat generated by friction between the tool and the material it is cutting that is the main cause of tempering. The evidence at present available tends to show that mechanical work which is not accompanied by a great rise of temperature tends to *harden* metals and alloys.

However this may be, the author fully admits that further research in regard to the causes of the tempering of high-speed steels is necessary, and until these have been elucidated such steels will not reach their highest degree of efficiency in actual industrial practice.

Blast Furnace Gas Engines.

Three papers on this subject were presented, dealing with English, Belgian and German practice.

Tom Westgarth's (Middlesbrough) paper is entitled "Notes on large gas engines built in Great Britain and upon gas cleaning." He gives a list of gas engines of 500 hp. or more, built in England; there are six manufacturers of such sizes in England; the number of such engines actually built is 119, and their aggregate indicated horsepower 96,085, but it must be said that the author includes not only large gas engines, operated with blast furnaces gases, but also those supplied with producer gas. The latter are greatly in the majority. A few gas-cleaning apparatus are briefly described.

Prof. H. Hubert (Liège) discusses "the design of blast furnace gas engines in Belgium." The paper is essentially a historical review of the work of the Cockerill Co., with some notes on recent tests, made by the author, of the 1,400-hp. two-cylinder double-acting and tandem gas engine, installed in their electric service. The progress made during the last ten years by the Cockerill Co. is indicated by the following figures: In comparing the trials of 1906 with those of 1900, which gave results which were considered excellent at that time, there was found a diminution of 15 per cent of calories used per 1 hp. For the brake horsepower the reduction attains 31.4 per cent. Finally, the thermal efficiency came out as 15.77 per cent for an 8-hp. gas engine tested in 1896, as 25.2 per cent for a 600-hp. single-cylinder, single-acting, constant-admission engine, tested in 1900, and as 29.84 per cent for the 1,400-hp. engine, tested in 1906.

The advantage obtained, as compared with the employment of steam, is by no means the least interesting of facts brought out by this investigation. Admitting that a 70 per cent efficiency is obtained by burning the gases under boilers, which is considered satisfactory, and that the steam raised amounts, according to Carnot's cycle, to an efficiency of 40 per cent (between 200° and 10°, which the cycle of Rankine makes 77 per cent of that of Carnot), and finally that one could succeed in obtaining 80 per cent of the Rankine cycle, the thermal efficiency becomes 17.25 per cent, which is lower by 42 per cent than that of the 1,400-hp. gas engine mentioned above (29.84 per cent).

The author assumes that a blast furnace, producing 100 tons of pig iron per day, produces about 9,000 cubic meters of gas at 1,000 calories, available for the production of power. Under these conditions, by obtaining 2,450 hp. with the steam engine and 4,220 by using the gas direct in gas engines, there remains a difference of 1,770 hp. in favor of this new application over the results obtained by a first-class steam plant of the usual (boiler) type. "These figures explain the success obtained in metallurgy by the direct use of blast furnace gas."

The paper is concluded by a list of Cockerill blast furnace gas engines, "at work or building;" they number 55, and their aggregate horsepower is 53,505 i. h. p.; there are "additional orders" for nine more Cockerill gas engines aggregating 9,650 i. h. p., while 113 engines have been built by other companies, under licenses from the Cockerill Co., aggregating 103,550 hp.

A paper by K. Reinhardt (Dortmundt) on "the application of large gas engines in the German iron and steel industries," is quite voluminous and pretentious, covering 107 pages with numerous illustrations in the text and fifteen additional pages of drawings. It is impossible to give at this place more than an outline of the ground covered in the paper.

The first part of the paper gives exact information on the extent of the application of gas engines in iron works and collieries in Germany, on the basis of the answers received upon a systematic circular inquiry of the Society of German Ironmasters.

Of the forty-nine German smelting works questioned, thirty-two had in March, 1906, gas engines at work and nine had ordered such engines. The largest aggregate power of gas engines at a single works amounts to 35,000 effective horsepower, sixteen works possess over 10,000 hp., and twenty-seven works possess over 5,000 hp. in actual working.

In most German iron works the whole of the gas engines work continuously without any reserve; a few have up to 40 per cent reserve of gas engines, and a few have a similar reserve of older types of steam engines or steam turbines.

Nearly all engines in German iron works work with blast-furnace gases. Two plants use only coke-oven gases, three use blast-furnace gas and coke-oven gas separately, and one plant uses the two gases mixed. The Mansfield Co. utilizes the waste gases from the copper-smelting furnaces for driving gas engines. Producers employing coke as fuel are kept as a reserve at seven works. "They are really only of use in case of a strike, to assure the working of the most necessary part of the plant.

The application of gas engines in German collieries is much less important. This is due to the fact that the heat given off by the older type of coke oven can only be utilized under steam boilers, and, consequently, for these older plants, steam boilers are inevitable in collieries. Only the excess gas produced in the coke ovens is available, so that steam engines and gas engines will always be found in conjunction; and, indeed, in larger proportion as regards steam engines than is the case in iron works. In the new regenerative coke ovens the waste heat is utilized for preheating the oven itself, whereby there is an economy in gas, and a greater excess of gas is available for driving gas engines. In view of the irregular production of gas, however, the application for gas-engine driving can only be considered free from drawbacks when at least sixty coke ovens are in operation. The production of gas in coke ovens is much more irregular than in blast furnaces.

"Perhaps in the near future, in addition to the excess coke oven gases used for driving motors in collieries, producer gas, which will be obtained in the ring producer patented by Bergrat Jahns (*Zeit. d. Ver. Deutsch. Ing.*, 1904, p. 311), may also be used. The chief object of this producer is the utilization of the waste heaps or culm banks, and the production of gas, as far as possible free from tar. The gas from the producer mentioned is naturally also suitable for the driving of gas engines, which is demonstrated by the gas-engine plant at the Von der Heydt mine. The same object is effected by the Turk and other producers; but so far as the writer is at present aware, the utilization of the waste heaps and of low-grade coal in gas producers is only just being introduced, so that, for gas-engines in collieries, at present only coke-oven gas need be taken into consideration. So far as he is aware, in the beginning of March of this year, sixteen collieries possessed thirty-five gas engines at work, in course of erection or on order. The aggregate power of all these engines was 30,300 eff. h. p. Of these twenty-four engines were already working at 15,000 eff. h. p., nearly all for the production of electricity.

"The introduction of large gas engines in collieries was subsequent to their introduction into iron works, and, there-

fore, only engines of comparatively recent construction are to be found. Exception must, however, be made in respect to the smaller motors, which were early employed in plants for the recovery of the by-products in coke-oven gas."

The second part of Mr. Reinhardt's paper is entitled "practical experience obtained in working," and deals essentially with the methods of gas cleaning. This part will be dealt with in detail in our next issue.

The third part of the paper relates to "the present design of large gas engines in Germany." At the present time twenty-nine firms in Germany build large gas engines. Of these, twenty-one firms build double-acting four-cycle engines, five firms build two-cycle engines, and three firms build both systems.

The author first makes general remarks on details of construction of cylinder, exhaust-valve chest, valve gear, stuffing boxes, cooled pistons and piston rods, ignition and starting. The following modern types of German double-acting, four-cycle gas engines are then described and illustrated: Maschinenbau Gesellschaft, Nürnberg; Gasmotorenfabrik Deutz; Ehrhardt & Schmer, Schleisfmühle; Märkische Maschinenbau-Anstalt, Wetter-Ruhr; Elsässische Maschinenbau-Gesellschaft, Mühlhausen; Fr. Krupp Aktien Ges, Essen-Ruhr; Gutehoffnungshütte, Oberhausen; Schüchtermann & Kremer, Dortmund; Maschinenbau-Aktienges Union, Essen-Ruhr; Duisburger Maschinenbau-Aktienges. formerly Bechem & Keetman, Duisburg; Dingler'sche Maschinenfabrik A. G., Zweibrücken.

Of two-cycle engines the engines of the Oechelhaeuser system, constructed by the Aschersleben er Maschinenbau A. G., and by A. Borsig, Berlin, and the Koerting engines, built by Koerting Bros. and their concessionaries are described .

The question of two-cycle versus double-acting four-cycle engines is considered as an open question by the author. Up to March, 1906, German gas engines with 260,000-brake horsepower, were at work or on order for double-acting four-cycle, as against 91,000 for two-cycle. "If these figures prove that the competition of the two-cycle engine must not be neglected, on the other hand the importance and connection of the builders of the four-cycle types must be considered advantageous to the latter."

For driving high-speed dynamos, double-acting four-cycle engines are preferable. On the other hand, the two-cycle engine is, without any doubt, most suitable for driving blowing cylinders. The author thinks that theoretical discussions concerning the correct or the incorrect mechanical efficiency, which during the last year created such a stir in Germany, do not cut any figure in practice at present. The problems of security in working, in addition to its price, is all-important; and the quantity of gas consumed per brake-horsepower is the least important.

"Suitable trials concerning the consumption of gas in more recent engines are not available for comparison, therefore it is not known how far the two-cycle engine is at the present time, in this respect, still inferior to the four-cycle engine. Should the iron works now be compelled to consider an economy of gas, the author does not believe that the larger consumption of the two-cycle engines would for long have any great influence on the question of systems; for then the iron works would probably first consider a more thorough cleaning of the gas employed for heating the blast and for burning under the boilers, thereby increasing its value, and in this manner to save gas. So long as these conditions remain as they are, and so long as the four-cycle engine is not more secure, under average working conditions, than the two cycle engine, so long will the question of systems not be decided by general and theoretical considerations, but in such cases by the iron works and mining industries themselves.

"The situation was well summed up by a manager who, referring to this very point, told me personally that he himself preferred double-acting four-cycle, but his engine drivers preferred double-acting two-cycle.

"As regards other industries, they need not at present be considered to such an extent that they could assist in deciding the question.

"In conclusion, it is my duty to state that the present position of the application of gas engines in German iron works shows the value the managers of these undertakings attribute to the better and less dangerous utilization of the waste gases of their furnaces, and the successful efforts that have been made by the German engineers in order to meet the requirements suddenly demanded by iron works. It also proves that German iron works are obliged to utilize to the uttermost the sources of power at their disposal, in order to ensure their existence and their participation in the trade of the markets of the world. In other richer countries, with more favorable conditions, the matter is not so urgent."

[An account of the extended discussion which followed the reading of these three papers may be found in the London letter from our special correspondent on another page of this issue.]

Constitution of Iron-Carbon Alloys.

In a paper on "the constitution of iron-carbon alloys," Dr. Albert Sanveur (Cambridge, Mass.) discusses the meaning of the various curves in the classical Roberts-Austen Roozeboom diagram.

The chief result of his considerations is as follows: Microscopical evidences show conclusively that four distinct constituents are formed during the cooling of steel, namely: (1) austenite, (2) martensite, (3) troostite, and (4) pearlite, and martensite, and especially troostite, may be regarded as transition constituents in the transformation of austenite into pearlite. On the other hand, by looking into the meaning of the thermal critical points of steel, it is found that four constituents must necessarily be formed during the cooling of steel, and that these must be respectively: (1) solid solution of carbon in gamma iron, (2) solid solution of carbon in beta iron, (3) solid solution of carbon in alpha iron, and (4) the iron-carbide eutectoid. Dr. Sanveur's conclusion is that these four constituents must be those revealed by microscopical examination, that is, respectively, austenite, martensite, troostite and pearlite.

Crystallography of Iron.

An extended paper by F. Osmond and G. Cartand (Paris) on the crystallography of iron cannot be briefly abstracted. The only positive conclusion which the authors draw from their experimental researches is, that the three allotropic varieties of iron, although they all crystallize in the cubic system, present well-marked specific characters, and cannot have the same internal structure.

After the conclusion of the London meeting and a trip through England, a large number of the party of the American Institute of Mining Engineers followed an invitation of the Society of German Ironmasters to visit the Rhenish-Westphalian iron district.

The party arrived on Aug. 13 in Duesseldorf, and visited on Aug. 14, after a trip down the Rhine, the Krupp works at Rheinhausen, in which practically all the work of the Krupp concern is centered, with the exception of the war material, which is made at the Essen works.

On Aug. 15 three different parties visited either the Haniel colliery on the Rhine, near Homberg, or the two large iron and steel works at Ruhrort (the Phœnix and the Rheinische Stahlwerke), or the Gutehoffnungshütte at Oberhausen.

On Aug. 16 an excursion was made to Remscheid, with a visit of the Héroult electric steel plant at the Lindenberg works, while on Aug. 17 a trip was made on the Rhine, which brought the official visit to an end in the most delightful manner. All the American visitors were full of praise for the German hospitality showered on them. Private excursions were made on the following days to the Differdingen works at Luxemburg, and to Hannover and the Hartz Mountains.

Factory Scale Experiments With Fused Electrolytes—III.

BY EDGAR A. ASHCROFT.

In this, the last of the present series of articles, I shall describe a few, as yet untried processes of metal winning, the result of deductions and possibilities to which the formerly described work on lead and zinc, and the work with the sodium process described at the last meeting of the American Electrochemical Society, and printed in the June number of this journal, have led me.

The double electrolytic apparatus with magnetic stirring and circulation of the intermediate electrode has many such possibilities, and some of them at least should be a fruitful field for future experimenters to work in.

Thus, in the treatment of mixed ores, several processes suggest themselves. The diagram, Fig. 1, shows one such in which sulphide ore is electrolyzed directly into a bath of lead, all the metals being deposited by a current of high density, and the sulphur distilled off and collected.

In the second cell all the metals other than the precious metals and the metal which is employed as carrier (in most cases lead) are dissolved at the fluid anode and received fractionally upon a suitable cathode.

To render this possible without encountering too great electrical resistance and other difficulties, the author has devised a form of revolving cathode shown in Figs. 2 to 5. This apparatus is suitable for collection of metals either in solid or in liquid form. The action of the apparatus is as follows: A disc is supported on a vertical trunnion, the whole moving

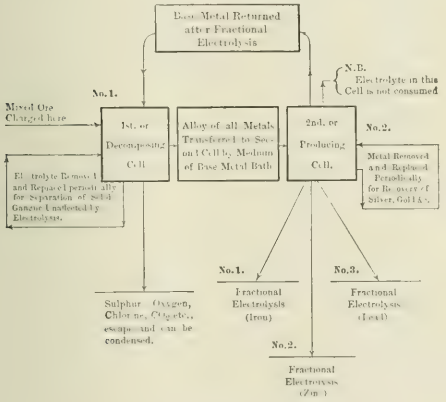

Base Metal Returned after Fractional Electrolysis
N.B. Electrolyte in this Cell is not consumed
No.1.
Mixed Ore Charged here
1st. or Decomposing Cell.
Alloy of all Metals Transferred to Second Cell by Medium of Base Metal Bath
2nd. or Producing Cell.
No.2.
Metal Removed and Replaced Periodically for Recovery of Silver, Gold &c.
Electrolyte Removed and Replaced periodically for Separation of Solid Gangue Undecomposed by Electrolysis.
Sulphur, Oxygen, Chlorine, Organics, escape and can be condensed.
No.1.
Fractional Electrolysis (Iron)
No.3.
Fractional Electrolysis (Lead)
No.2.
Fractional Electrolysis (Zinc)

FIG. 1.—SCHEME OF TREATMENT.

system being balanced so that it floats upright in a pocket of fused metal. When the current passes, the radial of lines of force distributed along the disc cut lines of magnetic force produced by a coil carrying the main current and placed behind the lining of the cell (exactly as in the formerly described apparatus wherein the liquid cathode is made to revolve).

From the rectangular cutting of these two sets of lines mechanical motion of the disc results. The disc spins violently, throwing up a wall of fused metal all round the trunnion, which keeps it automatically in the center. When a solid metal is being collected the friction of the disc on the electrolyte also helps to ensure a good deposit, and it is only necessary to slip out the old disc and replace it by a new one when the deposit gets too heavy.

A continuous adjustment may be made for the slow increase of weight by an automatic device whereby the quantity of floating medium in the pocket is varied, in order to maintain the distance between the cathode and the anode constant.

When a liquid metal is being received at the cathode this is thrown off the periphery of the disc by centrifugal action, and may be collected in an annular groove, as shown in Fig. 3. Again, the motion of the disc may be used for forcing the metal

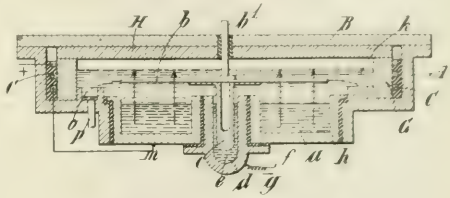


FIG. 2.—DETAILS OF REVOLVING CATHODE.

inwards along various arrangements of spiral grooves towards the center, when it may be drained off or ejected upwards by a screw pump device. The details of such devices will be self evident.

I should like to point out here that in all such devices the thermal efficiency of the apparatus, considered as a motor, is not only of minor importance on account of the relatively small energies we are dealing with, but it is absolutely unimportant to the net result, because in all such devices we are employing part of the electrical energy of the system as a heat producer anyhow (this is always the case in a minor degree even when the bath is heated externally), and any energy wasted in the motor system contributes directly to the heat of the bath. The objections which have been raised to this class of agitating device, therefore, do not stand at all on examination. The real questions which determine the value of such a device are durability and cheapness, both of first cost and of manipulation; in other words, convenience.

In this revolving solid cathode device (as in the case of the revolving fluid cathode) there is no kind of positive friction at all, the moving parts being in contact with fluid media, and so no destruction of parts due to friction results.

2. Very many modifications and possible applications of the revolving solid cathode device at once present themselves. Sodium deposited on such a cathode may be caused to collect in the center of the revolving disc in virtue of the centrifugal force, and thence be continuously or periodically removed by any convenient device. Magnesium might be similarly treated, and possibly aluminium, also, but of these metals I shall speak further. Iron and zinc can be separated by such means, and also lead from iron or lead from zinc. Zinc, lead or iron can be separated from copper, and it is for these four metals that I think the arrangement presents the greatest promise of commercial application.

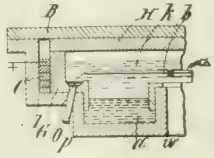


FIG. 3.—DETAILS OF REVOLVING CATHODE.

When any of the metals lower in the electropositive scale such as Au, Ag, Bi, etc., are present in the ore, these will accumulate in the lead or other metal employed as intermediate electrode, and may be for the most part readily separated out at suitable intervals. Gold and silver particularly may be readily removed from lead by cupeling. When antimony or arsenic are present they will be converted at once to chlorides in the first cell and will distil off. They may be readily collected for further treatment by condensers placed beyond the sulphur condensers.

In concluding this series of articles I am led to remark on the following points of interest to experimenters and investors, upon whose labor, enterprise, fortitude, wisdom and perse-

verance the future of our young industry depends. There is a very prevalent popular fallacy amongst the rank and file of engineers which greatly overestimates the importance and value of isolated new ideas. There is, also, especially in England, an equally exaggerated but conversely acting bias of mind towards conservatism and against all forms of experiment. Now, I think the true wisdom for the advancing industry of the future will be found half-way.

Ideas in themselves are as common as blackberries in a hedge row. Certain minds run to them naturally, and I have known men who could fill an encyclopædia with novel ideas in a year. Experiment of some kind also is absolutely necessary for every kind of industrial business. Many experiments, too, cannot be conducted on a small scale and be effective. But most people of any experience in such matters know how hard it is to draw the line between what is necessary research and what is mere desultory groping in the dark. Experiments, too, on a large enough scale to secure reliable results are always costly, and if indulged in too freely by any industry may easily be disastrous.

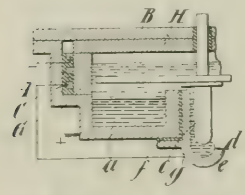


FIG. 4.—DETAILS OF REVOLVING CATHODE.

What is really needed to safely make the passage between the scylla and charybdis of ruin or stagnation is judgment; and judgment is a purely human and subjective quality, not (at present at least) reducible to a mathematical standard. Therefore, it arises that no kind of calculation or business precaution on known lines can securely protect capital from loss or secure the best fruition of labor.

The ultimate resource in such matters is the selection of the right man for the job, and giving him the best facilities and a reasonably free hand. The faculty of accurate selection of men is what secures the success of the industrial director before any other quality.

But even after all precautions have been taken it must be recognized that all progress is inherently dependent on some risk, for the less risk the man at the helm of technical affairs is willing to take on his shoulders, the less is his possibility of production, and especially of keeping abreast of the times.

Great difference in these respects are to be observed between certain national temperaments. Thus, whilst we can find much to admire in the great fertility of ideas and dash in execution, which is characteristic of the American people, who have accomplished so much even amidst wreck and disaster; we may also admire the slow-going qualities of our German contemporaries, who, like the tortoise in the race with the hare, make an excellent showing by perseverance and intelligent, patient application in spite of the slower pace they travel at. We may admire, too much of the sturdy conservatism of the English manufacturers.

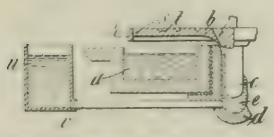


FIG. 5.—DETAILS OF REVOLVING CATHODE.

What we cannot admire is the obstinate and self-satisfied stupidity which is the true enemy to all progress, and though sometimes found in thoroughly prosperous communities is in reality only factoring on the labors of past generations, favored by accidental circumstances and contributing nothing at all to the sum of the world's work. The industrial progress of the future will be due to a blending and judicious selection of the best in all nations' characteristics, not to the exaggeration or exclusive triumph of any one. This cosmopolitan spirit is, I am glad to say, becoming the most general attitude of the technical workers and producers of to-day.

Metallurgical Calculations.

By J. W. Richards, Ph. D.
Professor of Metallurgy in Lehigh University.

THE RATIONALE OF HOT-BLAST AND DRY-BLAST

Problem 55 (for practice).

The blast furnace of Problem 54 (July number) had its blast dried before using, to the extent of leaving on an average 1.75 grains of moisture per cubic foot of air proper, at —5° C., in the air pumped. The composition of the ore, limestone and coke used was unchanged, also that of the pig iron made. The weights charged per 100 of pig iron were: Ore 177.6, flux 44.4, fuel 77.0, and the blast used calculates out oxygen 76.5, nitrogen 255.0, moisture 1.0. Analysis of gases: CO 19.9, CO^2 16, N^2 64.1 per cent. Product per day 447 tons. Temperature of gases 191° C., of blast 465° C. Piston displacement (air at — 5° C.) 34,000 cubic feet per minute. Assume heat in pig iron and slag same as before, in cooling water 20 per cent greater per day.

Required: (1) The volume of gases per 100 kg_o of pig iron made.

Answer: Measured dry, 355.9 m^3.

(2) A balance sheet of materials entering and leaving the furnace, per 100 units of pig iron.

	Charges.		Pig Iron.		Slag.		Gases.	
Ore 177.6	Fe^2O^3	135.7	Fe	95.0	.....		O	40.7
	H^2O	17.8	.....		.....		H^2O	17.8
	SiO^2	17.8	Si	1.0	SiO^2	15.7	O	1.1
	Al^2O^3	6.3	.....		Al^2O^3	6.3	.....	
Flux 44.4	SiO^2	2.2	.....		SiO^2	2.2	.....	
	CaO	21.1	.....		CaO	21.1	.....	
	MgO	2.1	.....		MgO	2.1	.....	
	CO^2	19.0	.....		.....		CO^2	19.0
Fuel 77.0	C	67.8	C	4.0	.....		C	63.8
	SiO^2	4.2	.....		SiO^2	4.2	.....	
	CaO	4.2	.....		CaO	4.2	.....	
	H^2O	0.8	.....		.....		H^2O	0.8
Blast 332.5	O^2	76.5	.....		.....		O	76.5
	N^2	255.0	.....		.....		N	255.0
	H^2O	1.0	.....		.....		H	0.1
							O	0.9
	Totals	631.5		100.0		55.8		475.7

(3) The volume of blast per 100 kg. of pig iron.

Answer: 2529 m^3 at —5 C.

(4) The efficiency of the blowing plant.

Answer: 82.3 per cent.

(5) The heat balance sheet of the furnace, per 100 of pig iron.

Developed.

Carbon to CO	92,950	Calories
Carbon to CO^2	206,955	"
Heat in hot blast	37,850	"
Solution of carbon in iron	2,820	"
Formation of slag	4,260	"
Total	344,835	"

Distributed.

Reduction of iron	165,870	"
Reduction of silicon	7,000	"
Expulsion of carbonic oxide (CO^2)	18,666	"
Evaporation of moisture of charges	11,342	"
Heat in waste gases	23,799	"

Decomposition of moisture of blast.....	3,225	Calories
Heat in slag.....	29,820	"
Heat in pig iron.....	32,500	"
Heat in cooling water.....	14,922	"
Lost by radiation and conduction (diff.).	37,791	"
	344,835	"

(6) Compare the heat items which are substantially different for the furnace run by moist and dried blast.

	Moist Blast.	*Dried Blast.*
Combustion of C to CO.....	131,220	92,950
Heat in waste gases.....	43,836	23,799
Decomposing moisture in blast.....	14,511	3,225
Loss by radiation and conduction....	51,180	37,791

It may be noticed, that using moist blast too much carbon was burnt to CO at the tuyeres; the chief item of economy with dried blast is the ability to get along with less. The smaller total amount of gases, particularly nitrogen, accounts for the lower temperature of the waste gases, with dried blast. The direct saving by reason of decomposition of the moisture is the smallest item of economy. The reduced losses by radiation and conduction are mainly because of the faster rate of running, these losses being nearly constant per day. The ratio of these losses is found to be 0.74, whereas, the inverse ratio of the pig iron productions per day was 0.79.

(7) Calculate the carbon burnt at the tuyeres, the proportion of the carbon available thus consumed. Compare these items with those of the furnace on moist blast.

	Moist Blast. %	*Dried Blast.* %
Carbon burnt at tuyeres.....	75.3	58.05
Total fixed carbon charged.....	84.3	67.8
Proportion burnt at tuyeres.....	89.3	85.6
Fixed carbon really available.....	79.4	62.9
Proportion burnt at tuyeres.....	94.8	92.3

(In some charcoal furnaces of low heat requirement, *i. e.*, with pure ores and fuel, as little as 37 parts of carbon is burned at the tuyeres per 100 of pig iron produced, which represents, moreover, only 70 to 75 per cent of the available fixed carbon in these furnaces.)

(8) The proportion of the whole heat requirement available in the region of the tuyeres.

Answer: 53 per cent.

(9) The proportion of the iron in the furnace which is reduced by solid carbon from FeO.

Answer: 23.8 per cent.

(10) The theoretical maximum of temperature at the tuyeres.

Answer: 1965° C.

Hot Blast.

For several centuries blast furnaces were run by charcoal as fuel and with cold blast. How great the variations in temperature of the cold blast may be, can be inferred from the experience of a furnace manager in the Urals, Russia, who noted temperatures of the air nearly 40° C. in the summer and —60° C. in the winter. Assuming an average temperature of 0° C. for unheated blast, burning charcoal to CO, the theoretical maximum of temperature before the tuyeres can be calculated as follows:

Heat generated by combustion		2430 Calories
Heat in hot carbon being burnt	0.5t — 120	"
Volume of CO and N^2 formed		5.3795 cubic meters

$$\text{Temperature} = \frac{2310 + 0.5t}{5.3944\ (0.303 + 0.000027t)}$$

Whence $t = 1678°$

This does not mean that the pig iron and slag will be carried to this temperature, any more than if a locomotive could run alone 90 miles an hour it could therefore pull a train of cars that fast. The hot gases, CO and N^2, are at the start at this temperature, and as they ascend and come in contact with the descending iron and slag, these are raised to temperatures approximating towards, but always lower than, the above. In fact, the temperature to which the iron and slag is raised depends on the relative quantities of iron and slag to fuel burnt, and the speed with which the furnace is worked.

If the blast is heated, its sensible heat is simply added to the numerator of the above expression. We can easily find out how much sensible heat the 4.4685 cubic meters of air brings in, at any temperature desired [Q = 4.4685 (0.303t + 0.000027t^2)], and then solve the quadratic anew. We thus find:

Temp. of Blast.	*Heat in Blast.*	*Theoretical Temp.*
0° C.	0 Calories	1678° C.
100° "	137 "	1762° "
200° "	276 "	1845° "
300° "	417 "	1929° "
400° "	561 "	2012° "
500° "	707 "	2096° "
600° "	856 "	2180° "
700° "	1007 "	2265° "
800° "	1160 "	2350° "
900° "	1316 "	2435° "
1000° "	1475 "	2520° "

The heating of the blast thus raises the maximum temperature available some 68° C. for each 100° C. to which the blast is heated. It not only increases the temperature available, but also the number of heat units, thus increasing both the quantity and the intensity of the heating in the tuyere region. Of these two items of increase, that of the intensity factor is the most important, since it regulates the rapidity of transfer of heat to the charge and the efficiency and speed of the smelting action of the furnace.

Dried Blast.

Each kilogram of water vapor decomposed absorbs 29040 ÷ 9 = 3227 Calories, which would not be needed if the 0.67 kilo. of carbon thus employed was oxidized by air instead of by moisture. Per kilogram of carbon oxidized by water vapor, there is absorbed 58,080 ÷ 12 = 4,840 Calories, while this kilogram of carbon can only furnish 2430 Calories in becoming CO, leaving a net absorption of 2410 Calories per kilogram of carbon thus burned, against which, however, can be credited the heat in the kilo. of carbon burned and the sensible heat in the water vapor itself; the former is 0.5t — 120, and the latter can be calculated from the volume of the water vapor, 1.8519 cubic meters. We thus have, per kilogram of carbon thus oxidized:

Temperature of Water Vapor.	*Heat in Moisture.*	*Heat in Products.*
100°	66 Calories	0.5t — 2464 Calories
200°	137 "	0.5t — 2393 "
300°	214 "	0.5t — 2316 "
400°	296 "	0.5t — 2234 "
500°	384 "	0.5t — 2146 "
600°	478 "	0.5t — 2052 "
700°	577 "	0.5t — 1953 "
800°	682 "	0.5t — 1848 "
900°	792 "	0.5t — 1738 "
1000°	907 "	0.5t — 1623 "

Since the heat in the carbon burnt (0.5t — 120) can never equal numerically 1623, it is seen that under no circumstances can the water vapor do anything but diminish the quantity of heat available at the tuyeres, while the products of its decomposition CO and H^2 increase the volume of the products and so diminish still further the intensity of temperature attainable.

The best way to determine the amount of moisture in the air is to draw it through a tube containing concentrated sulphuric acid, measure the quantity of dry air drawn through, and weigh the amount of moisture caught by the tube. This gives the weight of moisture *accompanying* unit volume of dry air

(not weight of moisture in unit volume of moist air). Determinations by the wet and dry bulb thermometers, the whirled psychrometer, humidity gauges, etc., are none of them so reliable as the above described method, which is direct, simple and accurate. The results may be obtained in grains per cubic foot of dry air or milligrams per liter. The best way to express them, for further use, is in ounces avoirdupois per cubic foot, or kilograms per cubic meter. The first is obtained by dividing the number of grains by 437.5, the second by dividing the milligrams per liter by 1000. The numbers thus obtained are practically identical in the two systems.

The theoretical temperatures attainable with moist blast of varying degrees of moisture and heated to various temperatures are obtained by applying the previously explained principles. We have already calculated the temperature obtained by burning carbon with dry air of various temperatures. We have also calculated a table of the deficit of heat available produced by the entrance of 1.5 kilos. of water vapor (which would oxidize 1 kilo. of carbon and produce 1.8519 cubic meters of CO and H^2). We are, therefore, prepared to calculate a table of the maximum temperature attainable using blast of any degree of humidity heated to any practical temperature. Before giving the table we will run through the details of one calculation, to make clear the method employed.

Illustration: What is the theoretical maximum temperature using air which carries normally 10 grams of moisture per cubic meter of dry air, reduced to standard conditions (*i. e.*, 10 grams per 1.293 kilograms of dry air), and heated to 500° C.?

It takes 3.5275 cubic meters of dry air at standard conditions to burn 1 kilogram of carbon. If dry, and at 500°, there is 2430 Calories generated by combustion, 707 Calories in the dry air used, 0.5t — 120 Calories in the hot carbon, and the total heat thus at hand raises the 5.3944 cubic meters of products to the temperature of 2096° C., as is determined by solving the equation

$$t = \frac{2430 + 707 + (0.5t - 120)}{5.3944\ (0.303 + 0.000027t)} = 2096°$$

As modified by the moisture, the 4.4685 cubic meters of dry air would be accompanied by 44.685 grams of moisture, or 0.044685 kg., which would oxidize two-thirds of its weight of carbon, or 0.02979 kg. of carbon, which would contribute to the heat available 0.02979 (0.5t — 2146) Calories, making a contribution of 0.015t — 64 Calories to the numerator of above equation. The products of combustion will be increased by H^2 and CO equal in volume to twice the volume of the moisture, or $2 \times 0.044685 \div 0.81 = 0.1102$ cubic meters, the mean specific heat of which goes into the denominator. We then have

$$t = \frac{3017 + 0.5t + 0.015t - 64}{(5.3944 + 0.1102)\ (0.303 + 0.000027t)} = 2030°$$

Another method of solution, not using the previously calculated tables but based entirely on first principles, is to base the whole calculation on the use of 1 cubic meter of dry air, with its accompanying moisture, as follows:

Oxygen present in 1 m^3 dry air $= 1.293 \times 3/13$	$= 0.2984$ kg.
Oxygen present in the moisture $= 0.010 \times 8/9$	$= 0.0089$ "
Total	$= 0.3073$ "
Carbon consumed $= 0.3073 \times 0.75$	$= 0.2305$ "
Volume of moisture $= 0.010 \div 0.81$	$= 0.0123$ m^3
Volume of oxygen in dry air	$= 0.2078$ "
Volume of products from dry air $= 1.0000 + 0.2078$	$= 1.2078$ "
Volume of products from moisture $= 2 \times 0.0123$	$= 0.0246$ "
Total volume of products	$= 1.2324$ "

Heat of combustion of carbon
$= 0.2305 \times 2430$ $= 560$ Cal.
Heat in carbon at t°
$= 0.2305 \times [0.5t - 120]$ $= 0.1152t - 28$ "
Heat in dry air at 500°
$= 1 \times [0.303 + 0.000027\ (500)]\ 500$ $= 158$ "
Heat in moisture at 500°
$= 0.0123\ [0.34 + 0.00015\ (500)]$ $= 3$ "
Heat absorbed in decomposing moisture
$= 0.0123 \times 2614$ $= -32$ "
Whence results the equation

$$t = \frac{0.1152t + 661}{1.2324\ (0.303 + 0.000027t)} = 2030°$$

By applying either of the two methods of calculation explained, the temperatures in the following table are obtained:

Theoretical Temperatures Before the Tuyeres.

Moisture	Grams per cubic meter of dry air reduced to 0° C.							
	Grams per cubic foot of dry air reduced to 0° C.							
Grams.	5.00	10.00	15.00	20.00	25.00	30.00	35.00	40.00
Grains.	2.19	4.38	6.57	8.75	10.94	13.13	15.32	17.50
Blast.								
40°	1678°	1647°	1615°	1573°	1536°	1507°	1471°	1443°
100°	1723	1692	1666	1627	1587	1548	1526	1496
200°	1807	1776	1751	1712	1673	1636	1612	1584
300°	1892	1861	1837	1800	1760	1725	1700	1673
400°	1976	1945	1921	1885	1846	1813	1786	1760
500°	2061	2030	2007	1970	1933	1902	1874	1848
600°	2146	2115	2093	2055	2020	1991	1962	1936
700°	2232	2201	2178	2144	2108	2080	2050	2025
800°	2318	2287	2264	2227	2195	2169	2138	2114
900°	2404	2373	2351	2313	2282	2257	2226	2203
1000°	2490	2459	2437	2399	2369	2345	2314	2292

The calculations show that the temperature before the tuyeres may vary as much as 235° C. = 423° F., from change in the moisture of the air, from dryness to warm air saturated with moisture.

Notes on Electrochemistry and Metallurgy in Great Britain.

(*From Our Special Correspondent.*)

The Faraday Society Meeting.

The last meeting before the monthly vacation was held on what was new ground for the Faraday Society, namely, in the lecture theater of the Society of Arts, instead of in the library of the Institution of Electrical Engineers. The reason was the large audience which assembled to hear Prof. Birkeland's paper on the "Fixation of Atmospheric Nitrogen" (extracts from which were given on page 295 of our last issue). The chair was taken by Prof. Silvanus P. Thompson.

After the paper had been read in a slightly compressed form, Dr. Borns opened the discussion by asking whether the actual yield was no' a mixture of nitrite and nitrate. As regards the temperature of working, the temperature of the arc was popularly supposed to be 3,500° C. In the electric fixation of nitrogen was there any maximum temperature for efficient working? The replies were given in each case to each speaker, so the author replied that he did not think that the temperature exceeded 3,500° C., owing to the quick diffusion, the incoming cold air limiting the temperature.

Mr. Hutton then praised the paper and congratulated Prof Birkeland on his success. The advance over earlier systems was very marked, the magnetic deflection of the arc being a special feature. It was stated that variations in frequency and air flow did not occasion much variation in the yield. Were

the results sufficiently definite to afford exact differentiation? He would also like to ask a less discreet question, namely, what was the actual yield? Did the figure of 500 to 600 kilogrammes per kw-year refer to the arc, or did it include the driving of the arc blowers, losses in transmission and in deflection. Prof. Birkeland answered that the magnets only took 0.7 per cent. In the new 30,000-hp. installation only 1,000 hp. was used for all other purposes than feeding the arc. As to variations in yield, structural design seemed more important than frequency and air flow. For instance, if the fire-chamber was too narrow the diameter of the flame increased, and after a certain point the output fell off.

Dr. Perkins asked whether if the air was drawn through at too great a rate, would it not reduce the yield by cooling the flame? Also, did not a too narrow a chamber ensure a greater cooling effect and lower the efficiency? Prof. Birkeland replied that too great an air velocity extinguished the arc.

Prof. Walker raised the question as to whether 79 grammes per kw-hour might be taken as the maximum yield. The reply was in the negative, to the effect that this was not the highest output. That figure had been cited from a particular experiment. The highest figures had to be kept secret.

Dr. Steinhart, speaking from a practical standpoint, wanted to know whether the estimate of £4 per ton for the electrically produced calcium nitrate covered only the cost of electricity, labor and raw material, or did it include depreciation, etc.? Also what was the maximum specific gravity of the nitric acid obtained? In his own experiments 1.35 had been the best. In reply, Prof. Birkeland explained that the price quoted covered all expenses. It was possible to produce 50 per cent nitric acid, but the sole purpose of the Notodden factory was to produce calcium nitrate. The cost per kw-year might be taken at 15 kronen (18 kronen = $5 roughly).

Mr. Crooks asked about the life and construction of the copper electrodes. How was it they did not melt, and how was cooling effected? The reply consisted in the passing round among the audience of a hollow water-cooled copper electrode, which had stood 500 kw. for 300 hours.

The discussion was wound up by Prof. Silvanus P. Thompson, who spoke of his own visits to Notodden, and praised its industrial and scientific features. Notodden was a commercial factory and nothing else. At Arendal all the experimental work was carried out. Prof. Witt's investigation was most exhaustive and the result of 500 grammes per kw-year must be regarded as a minimum and not as a maximum yield. Nitrite was produced as well as nitrate but the latter was increased by a further oxidation of the nitrite in two receivers before passing to the scrubbers and towers. The cost of power was all important, and he could vouch for the fact that the cost per horse-power-year at a pulp factory at Svaelgfos was only 10 shillings. Dr. Borne had spoken of arc temperature. In the carbon arc the limit was that of the volatilization of carbon. In other arcs other limiting factors arose. Really the origin of Prof. Birkeland's work lay in his early investigations of the phenomena of the Aurora Borealis, and also in the shadow effect in the Crookes' tube as limited by a current carrying solenoid, and also in the fact that cathode rays varied in their susceptibility to the action of magnetic fields.

The second item on the agenda, in the shape of an interim report by Dr. Haanel, met with a very restricted treatment, owing to the lateness of the hour. Mr. Harbord read a brief laudatory extract, pointing out the fact that titaniferous ores could not be used in blast furnaces owing to the refractory nature of the slag. (The paper of Dr. Haanel is essentially identical with his last preliminary report to the Canadian Government, which has been fully covered in our July issue, p. 265.)

Dr. G. Senter's paper on the "Electrolysis of Dilute Solutions of Acids and Alkalies at Low Potentials; Dissolving of Platinum at the Anode by a Direct Current," was then taken as read, discussion being deferred until the first meeting of the Autumn session.

PURIFICATION OF DRINKING WATER WITH HYPOCHLORITES.

A paper, read by Dr. S. Rideal and Mr. Ronald Orchard at the Bristol Congress of the Royal Sanitary Institute, "On the Prevention of the Growth of Algae in Water Supplies," contains an account of some exhaustive laboratory experiments with electrolytic hypochlorite solutions. The paper commences with a discussion of the value and objections (largely sentimental, in the opinion of the authors) of copper sulphate treatment. In regard to this they point out that the actual cost of the material is insignificant, the chief outlay being incurred in connection with labor, storage and the systematic testing of the product for freedom from copper. They advocate the use of electrolytic chlorine in the hypochlorite form, as even if such treated water was distributed with its full amount of "available chlorine" it would not be harmful in the high dilution employed. However, the removal of the chlorine is rapid and very certain, so that the only permanent addition to the water is a minute quantity of common salt.

For their experiments the authors used a solution obtained from a hypochlorite plant of Mr. W. Pollard Digby's invention. This solution, besides having the advantage of cheapness, contains but little undecomposed sodium chloride, and consequently the final additional of chloride to the treated water is very small.

In considering the treatment of water with electrolytic chlorine, first of all the immediate chlorine consumed figure must be determined, that is to say, the "available chlorine" at once taken up by the organic matter of the water, since obviously in order to successfully inhibit growth the addition of the chlorine should be equal to or in excess of this amount, although in one of our experiments an addition of only half the average Cl immediately consumed (0.17 parts per million), decidedly retarded growth in tap water.

Briefly, the conclusions arrived at from a lengthy series of experiments were:

1. Green growth can be entirely prevented by the use of electrolytic chlorine. 2. The different microscopic plants decidedly require varying amounts of chlorine to inhibit their development, confervæ appearing to be most susceptible to treatment. "Available chlorine" just in excess of that immediately consumed by the water, although disappearing itself in a few hours, inhibited conforvoid growth for some weeks. Desmids were found to be slightly more resistant, and in the experiments, required the addition of some 0.5 parts per million in excess of that immediately consumed by the water. Diatoms appeared to be still more resistant, and required one part per million of chlorine in excess. 3. Animalculæ, like daphnia and cyclops, are not killed by even larger doses of chlorine, and were found actively mottle after a month's treatment with eleven parts per million. 4. The addition of one part of copper sulphate per million was not successful with lake water, as it did not clarify, and the green growth was only slightly retarded. The effect was not equal to the use of 1.2 parts (0.2 in excess) of chlorine per million. 5. Under the microscope a distinct bleaching of the colored algæ was noticed with upwards of two points per million of "available chlorine," this cannot occur without strong physiological action. An exception to this occurred in Experiment No. 2 (with the lake water), where the desmid, *ankistrodesmus* was not bleached by 2.02 parts per million, and after some four weeks nibbetion appeared to be starting fresh growth.

In the case of drinking water treated with 0.44 parts per million, and infected with confervæ no growth occurred, and practically the only alteration in the chemical analysis was a reduction in the oxygen consumed figure, with an increase in the chloride from 2.1 to 2.4 parts per 100,000.

HYPOCHLORITES AND SEWAGE TREATMENT.

The report for 1905 of the directors of Oxychlorides, Ltd.,

to their shareholders, states that during the year the process and plant at Guildford have been under observation by the Royal Commission on Sewage Disposal, and it was in anticipation that the Commission's report would be available earlier that the shareholders' meeting had been delayed. Syndicates have been formed in Australia and in New Zealand for acquiring the company's rights. Those syndicates are purchasing from the company machines and dynamos, and also paying the incidental expenses.

The Iron and Steel Institute Meetings.

At the outset I must confess that the heading of this section does not adequately represent its contents. Really a longer title indicating the fact that the meetings were jointly held would perhaps have been better. This being cumbersome, and as the hosts, the Iron and Steel Institute, were at home, and as the guests were the American Institute of Mining Engineers, a shorter title will perhaps serve.

The meetings proper began on Tuesday, July 24, Mr. R. A. Hadfield presiding. As an opening, Mr. Hadfield tendered a hearty and cordial welcome to the guests, and dwelt on the importance of such meetings between the metallurgists of mother and daughter country. Such visits served to strengthen the already strong bonds between the two countries, and he recalled a soldierly pronouncement of Gen. Sherman's on the occasion of the visit of the Iron and Steel Institute in 1890 to New York. From international questions of political moment, Mr. Hadfield turned to statistical and financial questions, alluding to the great increases of production in the States, to statistics of trade, to increased pig iron productions, and to the risks of ore exhaustion.

Sir James Kitson then echoed Mr. Hadfield's words of welcome, remarking that the recent and regretted deaths of Sir Lowthian Bell, Sir Bernard Samuelson and Sir David Dale had left him with the melancholy distinction of being the senior past-president. He recalled some reminiscences of his visit with the Iron and Steel Institute to New York in 1890, when Abraham Hewitt had alluded to him as a manufacturer of fossil—*i. e.*, Yorkshire—iron. To this he demurred, for fossils of this nature were still "going strong." Although in the United Kingdom they could not rival the colossal figures of production in the States, he must point out that this fossil country now used more coal and produced more iron and steel than ever before in its history. In cordial words he bade the guests welcome, and hoped that their visit would be both happy and useful.

Capt. Robert Hunt then appreciatively acknowledged the welcome, and spoke of American indebtedness to British ideas, which they had tried to expand and to carry perhaps to a greater point of perfection, and certainly to greater dimensions in production.

After various items of formal business, three papers dealing with large gas engines using blast-furnace producer, and coke-oven gas, were read in abstract. Two by Mr. Bennett Brough (the paper on Belgian gas engines, by Prof. Hubert, and the paper on German gas engines, by Herr Reinhardt), while the third paper, on English installations, was read in person by Mr. Tom Westgarth. As a postscript to the paper Mr. Westgarth remarked that he regretted being unable to present any information regarding the large 500 and 1,000-hp. vertical marine type engines with which Messrs. Beardmore & Co. were experimenting. If the reversal of this could be secured, the type would be of considerable value for rolling mill work.

Mr. Julian Kennedy remarked that in the States they were only beginners in the gas engine business, and recognized the need of cleaning the gases. Dr. Raymond lightly alluded to the ominous signs of a deluge of gas engine papers which would shortly descend upon him. He would welcome it as bringing knowledge for, as a general rule, the only two people who really appreciated a paper were the author and the proof-reader. The gas engine promised economies, and the third place of decimals was worthy of attention in regard to savings. Prof. Kent said that hitherto Americans had hung back to see how things were going. They remembered that in regard to electric transportation the pioneers had lost money—those who waited stepped in at the end of five years and made profits, when the pioneers were scrapping their plants. They had now concluded that others had done enough scrapping, and were now securing licenses to manufacture the successful types. Another point was that blast furnace gases were irregular in quantity and quality; he, therefore, recommended auxiliary producers with bituminous coal, or coke-oven gases.

Mr. Duff then praised Herr Reinhardt's paper, and gave a retrospective account of his experiences seven years ago, when he had a surplus from some coke ovens thrown on his hands. There were no 500-hp. engines then available, so that he had to fall back on two 250-hp. engines. None of the three papers had mentioned an installation at Madrid, of Duff producers of Nurmberg engines, one of which had run for six months practically without a stop, no tar troubles arising.

Mr. Hamilton pointed out that in England the gas engine had got quite beyond the initiatory stage, and that now rapid developments might be anticipated. The competition of the future would settle the question of the relative values of two and four-cycle engines. Personally he attributed the latter to the poorness of furnace gas. He regarded the former as more efficient as an engine, but it must be remembered that where producer gas was employed radiation and other losses occurred at the producers.

Mr. Tannett-Walker said that he had been interested in gas engines all his life, and praised Herr Reinhardt's paper as a masterly treatise on the subject. Eight years ago he had been assured of gas engine perfection. Then it was certainly imperfect, and he had yet to be convinced as to the fulness of its present perfection.

After brief remarks by Mr. Mark Robinson and Prof. Turner, the one as a builder of gas engines for dynamo driving, the other as a user of several small gas engines, the largest of which was of 150 hp., Mr. B. H. Thwaite commenced to read the high sarcastic views of an aggrieved and neglected pioneer. He protested aginst the use of the word simultaneous as entirely inapplicable. As to gas cleaning, why the methods described by Herr Reinhardt as the latest evolution were substantially those disclosed in the Thwaite patent of 1894. A tactful whispered closure by Mr. Hadfield induced the speaker to omit several pages of protest, holding this over for the communications. A vote of thanks then brought the first day's meeting to a conclusion.

(Abstracts of various papers presented before the joint meeting of the Iron and Steel Institute and the American Institute of Mining Engineers will be found on pages 340 to 356 of our present issue, with some notes on the visit of the members of the American Institute of Mining Engineers to Germany.)

Market Report for July.

The most prominent feature in the metal markets during the past month has been the fall in the price of tin. Starting at £170 (3d) the price fell in the course of the week to £165. Since then the price has recovered, somewhat nervously, to £171, closing on Wednesday, the 27th, at £170.10 for cash, and £170 for three months. The fall is probably largely due to selling on the part of speculators who came in late during the boom.

The market is now a good deal steadier. Copper has experienced no very great differences, ranging from £81 to £78. As there is no surplus of material, the dealers can command prices. The closing on Wednesday were £82.2.6 and £81 for three months.

Pig iron has remained steady. Closed 51s. 4d. cash, 51s. 7½d one month.

Lead, steady, closing at £16.16.3 (English pig). Quicksilver, per flask of 75 pounds, £7 5.

Manganese ore (50 per cent and over) 1/1 per unit. Spelter, steady, closing £26.12.6.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

Industrial Electrochemistry.

The Measurement of Temperature in the Formation of Carborundum.—The results of an investigation for the purpose of determining the formation temperature of carborundum as well as the temperature at which it decomposes into graphite and silicon, are given by the experimenters, Messrs. Tucker and Lampen, in the *Journal of the Am. Chem. Soc.*, July, 1906. As fairly sharp lines of demarkation are observed between the layers of graphite, carborundum and siloxicon after a run of a carborundum furnace, the temperature might be obtained by temperature measurements at different points of the cross-section, after the furnace is brought to a stationary condition. A horizontal graphite tube was, therefore, placed through the center of the furnace, which tube contained a plug which could be pushed to any desired section, and its temperature measured by an optical pyrometer. The laboratory furnace, in which the experiments were conducted, was built on the general plan of a furnace used in the manufacture of carborundum, and filled with about 8 kilos. of a charge consisting of coke, 34.2 parts by weight; sand, 54.2 parts by weight, and sawdust, 9.9 parts by weight. A tube, bored out of Acheson graphite, 330 mm. long, 25 mm. outside and 18 mm. inside diameter, was put transversely through the center of the furnace. A graphite plug, 17.5 mm. in diameter and 9 mm. long, was put into the tube. The instrument used was a Wanner optical pyrometer. The method of carrying out the temperature measurements is briefly described, and the results are given. The authors found the decomposition point of carborundum into graphite and silicon to be at 2,218° C., and its formation temperature, that is, the point where the amorphous turns into crystalline, at about 1,920° C. From another experiment the decomposition point of carborundum was found to be 2,223° C., and the formation point 1,980° C. The line between siloxicon and carborundum is always less defined than that between carborundum and graphite. The average of the formation point is given as 1,950° C., and of the decomposition point 2,220° C.

Electric Steel Furnace.—During the recent trip of the American Institute of Mining Engineers through Germany, a visit was paid to the works of Richard Lindenberg in Remscheid, where the Héroult electric steel process is used, as developed by the engineers of the works and of the Elektrostahl Gesellschaft. The process is in all essentials the same as at the Holcomb Steel Works in Syracuse (our May issue, p. 164). The following description of the Lindenberg works is taken from *Iron Age*, Aug. 30: "Dr. Héroult himself was a member of the party, but, as was explained, had not visited the works before. The methods were explained by Dr. Fr. Eichhoff, consulting metallurgist, and by Mr. Lindenberg, the manager of the works. It was frankly stated, however, that some of the details of the practice developed at the Remscheid works were withheld. The plant consists of a 2-ton Wellman open hearth furnace, in which the raw material, principally scrap, is melted down, and the process is so conducted that the steel is over-oxidized. The steel is then transferred to the Héroult electric furnace, where the operation is really that which is known in crucible steel practice as 'killing' the steel. The quantity charged is about 2 tons, and the steel is purified in the electric furnace by the addition of scale or ore, and thus the elements in the metal, like silicon, carbon, manganese and phosphorus, are oxidized, sulphur, however, being an exception. The slag is cast off by tilting the furnace. Then a neutral slag is formed in the furnace by additions of lime and sand, under which deoxidation is carried on by means of carbon. Manganese, tungsten or other additions to the steel may be made to it by adding to the covering slag oxides of manganese or tungstic acid, and it is a special advantage of the method that practically all of the manganese thus added is recovered in the steel. The fact that the slag is white shows that no iron or other oxides are left in it. The yield therefore is high, being on an average 92½ per cent in the form of hammered blooms. The phosphorus contents of the steel have averaged 0.005 and of sulphur 0.012 per cent. The charge requires from 2 to 2½ hours independent of the size of the furnace. At Remscheid experience has shown that with a 2-ton furnace the cost of steel can be brought down to 120 marks ($30), and it is estimated that with a 10-ton furnace it can be brought down to 90 marks ($22.50) per ton. With a 2-ton furnace the requirement of electric power is 360 kw. per ton of steel, and it is estimated that with a 10-ton furnace this will be reduced to 150 kw. At Govtfors, Sweden, a 5-ton electric furnace has been in successful operation for a considerable period. During the visit of the engineers a cast of steel was made which poured quietly, while the stock of hammered blooms at the works and samples of bars which were shown proved it to be of a high quality."

Bleaching and Sterilizing by Means of Electric Discharges.—In our July issue, page 278, we referred to the method of Leetham for bleaching flour by means of a mixture of ozone and nitrogen oxides produced from air by means of electric discharges. Some notes on experiments in this line were given by S. Leetham and W. Cramp, at the York meeting of the British Association for the Advancement of Science, abstracted in London *Electrician*, Aug. 10. The effect of increasing the number of spark points in series to raise the necessary voltage, but not proportionally to the number of sparks, and a similar rule holds for an increase of length of spark gap. High-air speed also raises the pressure and sets up violent oscillations in the circuit. When the air resistance is high, ozone is the chief product. The ionization is then less marked. Ozone and oxides of nitrogen, contrary to many authorities, may commonly exist together without mutual destruction.

Metallurgy.

Gold.

Cyanide Practice at the Liberty Bell Mine, Telluride, Col.—In the *Engineering and Mining Journal*, July 28, Mr. W. F. Tracy outlines the method for treatment of a decomposed quartz ore containing a large amount of clay and shale, and the sticky nature of which prevents fine crushing before being fed to the batteries. There are eighty 850-pound stamps in the mill, to which solution containing 1 pound of cyanide per ton is fed in the proportion of four parts solution to one part of ore. The stamp duty is 4.3 tons per 24 hours, 14-mesh No. 22 iron wire screens being used. There are also provided three Abbé tire-type tube mills, equipped with spiral feed. Cast-iron tires and rollers have proved too soft, and have been replaced by rolled steel, which is proving satisfactory. The spiral feed is also stated to prove satisfactory. The linings, which were at first 2.5 inches thick, laid flat, have been replaced with such 4 inches thick, promising one year of life. Imported flints are used, and they are fed to the mills daily at a rate of about 1.3 pounds per ton of ore. Each mill is driven by a 5-hp induction motor. The sand, fed to the mills by two Dorr scraping sizers, contains about 11 per cent of slimes and 45 per cent moisture, 88 to 90 per cent of the discharged product will pass a 100-mesh screen. The tube mills take up somewhat 55 per cent of the mill product, or 95 tons each per 24 hours. The slimes and the tube mill discharge are passed over a second

set of amalgamated plates, six tables taking the product from one tube mill. The pulp from the tables passes through spitlutten to the settlers; the over-size is returned to the tube mills. Lime is added to the pulp in sufficient quantity to insure the solution being slightly alkaline when it reaches the second plates. The five settlers are each 33 feet in diameter by 10 feet 8 inches effective depth, slightly coned at the bottom. After settling about 14 hours, approximately one-half of the solution is decanted. The remaining pulp, consisting of two parts of solution to one part of ore, is discharged to the six agitators through central bottom discharge valves. The agitator vats are 17 feet in diameter by 11 feet to the top of the 45° cone. They hold 35 tons of dry ore per charge. A sufficient amount of cyanide in solution is added, to raise the strength of the solution to 1.5 pounds per ton. The pulp is agitated for 12 hours, and then discharged to an equalizer tank, from which it is drawn as required to the Moore filter plant. The latter contains two loading and four displacing tanks. There are four baskets, each containing sixteen plates, 6 x 8 feet, giving 6,336 square feet of filtering surface per basket. The filtering frames are covered with 20-ounce duck. The baskets are raised and lowered by two 30-ton hydraulic cranes, driven electrically. The pulp in the loading tanks is prevented from settling by air-lift agitation. Each basket remains in the loading tank for 1 hour, during which time about 16 tons of solution are drawn through the filters, and a 7/8-inch cake is formed with a vacuum of 17 to 19 inches. The basket is then transferred to the displacing tank, where the gold-bearing solution is displaced by water, an intermediate weak solution being used. The cakes leaving the loading tank contain about 33 per cent of moisture, about 75 per cent of which is drawn in from 30 to 40 minutes after the basket is transferred to the displacing tank, with only a trace of dilution. When the solution from the vacuum pump begins to show dilution, it is directed by a swinging nipple into a weak solution launder, from which it passes through the weak solution zinc boxes to waste. The values fall off rapidly after this point, the average time of displacing being about 1 hour and 20 minutes. Displacing is discontinued when the discharge from the vacuum pump shows 0.35 pound KCy per ton. This procedure leaves from 5 to 7 cents per ton of dry slime in the cake and about 3 cents worth of cyanide. Water at 10 pounds pressure is then turned into the baskets, and the cake blown under water in the displacing tank. The strong solution from the Moore filters is combined with the decanted solution, and passes through a clarifying filter and then to the zinc boxes. The latter are twelve in number, five compartment, each compartment containing 20 cubic feet of zinc. Two of the boxes are used for weak solution. The strong solution averages 1.2 to 1.3 pounds KCy per ton, and the weak 0.75 to 0.8 pound per ton.

Crushing and Grinding Practice at Kalgoorlie.—The much-debated question of tube mills versus grinding pans, which on account of the comparative tests of the two apparatus at the Ivanhoe mill, in which the pans appeared to better advantage, has been brought prominently before the metallurgical world, is taken up again by Mr. A. James in the *Mining and Scientific Press*, July 28. His remarks refer to West Australian practice. He believes that if better concentration can show a reasonably free cyaniding product, the necessity at Kalgoorlie for tube mills disappears. It is, however, by no means yet decided, and rather the reverse, that concentration as practiced in the district, is yielding a product capable of being bromo-cyanided without the very finest slimings, such as it appears only tube mills can as yet accomplish. An attempt to obtain Dr. Diehl's bromo-cyanide extraction by bromo-cyanide alone, without Dr. Diehl's bromo-cyanide method, namely, tube mill fine sliming, is apt to yield as poor results as are shown by one of the best mines in Kalgoorlie, where the residues, even with an ore supposed to be comparatively free from refractory minerals, are stated to be the highest in the neighborhood, 2½ dwt. per ton. The author then discusses more particularly the tests of tube mills versus grinding pans at the Ivanhoe plant, above referred to, and claims that, in comparing the relative efficiencies of pans and tube mills, an extraction test would have been more useful, more accurate and more practical than the Ivanhoe tests. He believes that the tube mill was choked with slime and could not do its most effective work, and that, therefore, the results show only a fraction of the work done elsewhere per horse-power-unit. He claims that the tests were unworthy of serious notice, mainly because the lack of opportunity for the free escape or removal of the finished tube mill product reduced them to an absurdity. On the other hand, the pans, especially the first pan, were allowed a much better opportunity of getting rid of their finished product, though even here the arrangements might have been improved. The author also refers to the difference of opinion locally regarding the working of the pans. At the Ivanhoe mill, the pans are provided with compensating weights, and they are highly thought of. At the next mine which the author visited, however, he was told that the compensating weight rings had been thrown out as having proved to be of no advantage whatever. At the next mine he was told that the old original 8-foot pans first introduced were the best pans on the field, and these 8-foot pans are being adopted for the most recent plants. He concludes with a plea for more authoritative work in this important matter.

Tube Mill Lining.—The practice of ore treatment at the Waihi mill, New Zealand, consisted of crushing under stamps to 40-mesh, the finer the pulp discharged from the battery the better the extraction in the cyanide plant. There was a limit to fine crushing, however, as beyond 40-mesh the stamp duty was decreased until fine pulverization became uneconomical. The Waihi was, therefore, one of the first plants to introduce the tube mill, which latter has proved most effective for regrinding. At the present time the stamps crush only to 15 or 20-mesh, the battery product being reground in tube mills to 150-mesh. The tube mills are provided with a special lining, designed and patented by Mr. H. P. Barry, the superintendent of the works. This lining is described and illustrated in the *Mining and Scientific Press*, July 28. Formerly it was very expensive to line the tube, silex from Iceland being used, which was laid with a special cement. It cost £80 to line a mill, and the time of service was only three months. Mr. Barry utilizes the flinty portions of the Waihi ore, and lays them in Portland cement; the lining costs £40 and lasts six months. The lining is formed of cast iron segments, which are curved to the shape of the tube, in sections having four or six divisions, each 4 x 6 inches. The following are the costs which have been found in actual operation:

Honeycombed casting, 28 cwt., at 15s.	£21
Broken quartz, 5 tons, at 20s.	5
Portland cement, 1 ton at mine, a 11s. per cask	4½
Sand (rough tailing), 1 ton	1
Labor at 9s. per day	8
Total	£39½

It is claimed that, as the casting can be made for 12s. per cwt., and the cost put down for quartz is rather excessive, the liner can be made for £30. The saving is stated to be about £260 per tube per annum. As the company is about to use ten or twelve tube mills this saving represents an important item.

Mining and Milling by Electric Power Machinery.—In the course of an illustrated article in the *Engineering Magazine*, August, 1906, on the application of electric power for driving various mining machines, Mr. C. V. Allen makes some remarks concerning the electric driving of stamp mills. According to him, in the electrical equipment of such mills the more usual practice is to drive the main shaft in sections, which in turn drive the cam shafts, so as to give the stamps from 100 to 104 drops per minute, there being two drops of the stamp to

each revolution of the cam shaft. He states that there are probably more mills equipped with 50-hp. motors driving units of twenty stamps each, though a few are operating with 100-hp. motors on forty stamps, and some with 350-hp. synchronous motors, driving an entire mill of 100 or 120 stamps. In almost all such installations as this last one referred to the mills were formerly driven by steam, and the steam connections are left as much intact as possible, if for any reason it should prove desirable or necessary to use steam. This method, of course entails considerable loss by friction in long-line shafts, and means also a complete shut-down of the mill in case of accident to the motor or transmission. The author thinks that there is little doubt that such drives in time will be changed to unit drives, as confidence is established in continuous electrical service. The ore crushing plant is a simple application of an induction motor of proper speed and horse-power, depending on the size and type of crusher used. One or more can be belted to the motor or to jack shafts driven by the motor. With this type of motor two pulleys can be placed on the motor shaft, extended on both sides, if desired, to drive two crushers. The fly-wheel of the crusher, together with the fly-wheel capacity of the motor rotor, is ample, with a motor of proper capacity, to withstand the heaviest shocks due to large or hard ore. In a cyanide plant, according to the author, the ideal condition is the application of individual motors to each machine, or small group of machines, as in the case of the mechanical agitators. By this method any one machine may be stopped when no work is to be done, or when repairs are necessary.

The Valuation of Shipping Ores.—In an article in the *Mining and Scientific Press,* July 14, Mr. R. Gilman Brown discusses the question of valuing ores for smelting purposes. He takes the case of a complex ore, which is free-milling to a certain extent, carries lead in appreciable amount and iron and zinc in sufficient quantity to modify the charge on the concentrated product. The tailings can be treated by cyanide, and yield a small but noteworthy product. For an average product the return from the books for a month show as follows:

Tonnage.	*Total.*	*Per Ton.*	*Per Cent.*
Saved by amalgamation...	$3,132.42	1.98	37.1
Saved by cyanide.....	1,123.59	0.71	13.3
Realized from concentrates	2,563.41	1.62	30.4
Lost in tailings.....	.....	1.02	19.2
Total	.....	5.33	100.00

Thus, it appears that the ore assayed $5.33 per ton, and that over 80 per cent is being recovered. However, the case appears different, when a single lot out of the six, which make up the shipment for one month, is taken. The basis of settlement is 95 per cent of the silver, 95 per cent of the gold at $20.00 per ounce, 90 per cent of the lead at 1 cent per pound, London quotation, with a reduction of 10 cents per ton for each unit above 10 per cent; 10 per cent zinc is allowed, 50 cents per unit being charged for each unit above this. The excess units of iron over silica are credited at 10 cents per unit per ton. Freight and treatment charges amount to $9.00 per ton. The smelter analysis was: 11.5 ounces silver, 0.95 ounce gold, 12.1 per cent lead, 10.3 per cent zinc, 28 per cent iron, 4 per cent silica, weight 19,386 tons. The following table gives the values of the shipment:

Metal.	Quantity.	Gross Value.	Deduction.	Net Value.	Gross Value Rec'd
					%
Silver	222.04 oz.	$136.83	$6.84	$129.99	95
Gold	18.417 oz.	380.68	12.34 18.42	349.92	92
Lead	4,691 lb.	139.38	46.91 9.25 4.07	79.15	57
Iron and Silica		46.53	.....	46.53	100
Total	.....	$703.42	$97.83	$605.50	86

DEDUCTIONS.

Excess zinc		$2.91		
Freight and treatment		174.47		
Total	$703.42	$275.21	$428.21	61

There were 115.3 tons of concentrates produced from the 1,580 tons of ore, making the proportion 7.3 per cent. The lot analyzed weighed 19,386 tons, thus the average represented 265.6 tons. Consequently, the gross value of the gold and silver in the concentrate is $1.94 per ton of ore, of the lead $0.52, of the iron and silica $0.17; a total of $2.63 being required to produce $1.62 from the sale of the product, the difference covering the various charges imposed by the smelter. It is thus seen that it is more or less difficult to figure out whether a block of ore is payable or not, but when the grade of ore and the degree of concentration are fairly uniform, an empirical factor can be deduced. This factor is 83 per cent in the present case, that is to say, the value realized by sale is 83 per cent of the precious metal value in the concentrate. Taking an average of 82 per cent for the shipments of the month under consideration, the corrected value per ton of ore is as follows:

Saved by amalgamation.....	$1.98	35.2%
Saved by cyanide.....	0.71	12.7
Saved in concentrates.....	1.98	35.2
Lost in tailings.....	0.95	16.9
	$5.62	100.0

Thus, from an ore assaying $5.62 in gold and silver values there will be realized $1.98 + 0.71 + 1.62 (1.98 × 82 per cent) = $4.31 or 77 per cent, which will be in the proper figure to use for calculating the value of any particular block of ore. The simplest case in valuing ore occurs when the whole product is shipping ore, where estimates should be based on analyses of group samples, representing average constituents of blocks or parts of blocks.

Copper.

A New Matte Separator.—In a paper read before the recent meeting of the Canadian Mining Institute, Mr. R. R. Hedley describes an arrangement which has been successfully used for some time at the lead smelting works of the Hall Mining & Smelting Co., at Nelson, B. C. It is the invention of Mr. H. Harris, and is based upon the idea of maintaining, outside of the furnace, the fairly complete separation of matte from slag which has taken place inside the furnace, instead of, as is usual, to allow the matte and slag to flow together to a receptacle, where they have to be separated again by gravity at a lower temperature. As shown in plan and vertical cross-section in the adjoining illustration, it consists essentially of an L-shaped cast-iron box, about 30 inches long, placed in free communication with the tapping hole, which latter should be as wide as possible. The apparatus is pressed close to the top jacket F of the smelting furnace, with its bottom on a level with the bottom of the tap hole F. When the furnace is in full operation, the matte and slag will be maintained at levels corresponding to the head afforded by the height of the slag

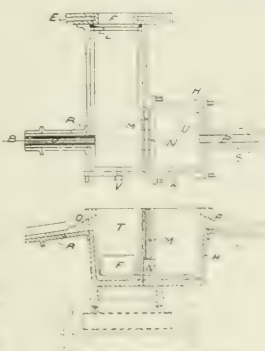


FIG. 1.—MATTE SEPARATOR.

and matte overflow O and P. The slag from the double stream flowing from the furnace will be arrested by the water-jacket barrier M, and will be maintained in the first compartment T to overflow at O. The matte will flow through the opening N into the second compartment U, and will rise until its head balances those of the matte and slag in the first compartment or the matte slag and blast pressure in the furnace, and will overflow at P. Thus the levels of matte and slag in the furnace will depend upon the weight of the blast and the height of the overflows O and P, which may be varied considerably at the will of the furnace man. The tap hole V serves for tapping out any small quantity of lead which may accumulate, and may also be used for tapping out the contents of the furnace when necessary. The author states that the device has now been perfected and is a regular adjunct of the furnace. The following advantages are claimed for it in comparison with the settling in forehearths: A better saving of lead and silver values in the slags; a more perfect equilibrium in the furnace, which is conducive to more perfect metallurgical work and less tendency to irregularities, which produce troublesome crusts and accretions; less time is lost in changing settlers; a saving of labor is obtained in handling the settlers and preparing the contents for resmelting; and, lastly, there is a saving in iron and steel tools as well as in matte pots. The apparatus is mounted on small wheels, which run on rails on a truck placed at a height corresponding to two short rails at the furnace. It will run for two or three weeks without change, which is quickly effected. It is not bolted to the furnace, the connection being made by tamping a small quantity of fireclay in the groove L, which, when filled with slag which chills between the surface of the packet and the flange, makes an efficient seal.

Electrolytic Treatment of Copper Ore at Miedzianka, Russian-Poland.—The ore at this locality consists essentially of copper glance, which is at times oxidized to azurite and malachite. It is treated by an electrolytic process, which is described by W. Stoeger in the *Oest. Zeitsch. für Berg. und Hittenwesen*, July 28. The ores are crushed and then roasted to a certain extent, the object being to obtain a mixture of copper sulphate and copper oxide. The roasted ore is lixiviated with dilute sulphuric acid, the copper being obtained as copper sulphate. The solution is then electrolyzed until only traces of copper remain in the electrolyte, a corresponding quantity of sulphuric acid being formed during electrolysis. This electrolyte is then again available for dissolving fresh quantities of ore. As the roasted ore contains the greater part of its copper in the form of sulphate, the amount of free sulphuric acid would increase after repeated extraction and electrolysis. However, there always remains some liquid in the ore residue, which has to be replaced by water. It is claimed, moreover, that as the electrolysis is not affected by varying amounts of free sulphuric acid, as well as by other salts, it is not necessary to keep the percentage of free sulphuric acid at a certain well-defined figure. The pulverized ores are mixed with 5 per cent of moist loam and formed into briquettes, which are first dried by the waste gases of the roasting furnace and then introduced into the latter. As the briquettes are porous, the roasting gases penetrate into their interior, and it is therefore possible to roast 10 tons per day in a small furnace. The roasted briquettes are again pulverized and brought into shallow vats, where they are treated with the dilute sulphuric acid obtained by the electrolysis. The solution thus obtained is clarified by filter presses. The copper is precipitated in vats similar to those used in the electric copper refining process, but with anodes of sheet lead, which are wrapped around with a textile fabric (Barchent). Circulation of the electrolyte is effected by stirrers, actuated by an eccentric. The vats are filled with the solution extracted from the ore. The solution contains about 5 per cent of copper and 1 per cent of free sulphuric acid. A current of 1,000 amperes, corresponding to a current density of 1 ampere per square decimeter of cathode surface, at 2.5 volts per vat, is passed through. The quantity of copper deposited amounts to 1.1 grams per ampere-hour, which is close to the theoretical value. The amount of electric power per 1 kilogram of copper is therefore 2.28 kw-hours, or 3.5-hp. hours. As one vat contains about 1 cubic meter of solution, the latter is in 35 hours sufficiently low in copper, that is, from 1½ to 1 per cent, to be run off and used again for extraction. The free sulphuric acid has by that time reached about 5 per cent. The ores are very rich, and the copper lost in the solution remaining in the residue is very small, even if the extracting solutions contain 1 to 1½ per cent of copper. This would, however, not be the case with poor ores; in that case electrolysis is continued until the solution contains only 1/10 per cent of copper, a lesser current density being used towards the end of the operation. Electrolysis is continued for about 1 month, until the copper cathodes are about 20 to 30 millimeters thick. It is stated that the cathode copper has an even red color and is nearly chemically pure, purer yet than the electrolytic copper from the refineries. The current is furnished by a dynamo of 12 volts and 1,000 amperes, power being obtained from a waterfall by means of a 50-hp. Francis turbine.

The Constitution of Copper Matte.—A good deal has been written regarding the constitution of copper matte, owing to the importance of the matter for the copper smelting industry. In the latest contribution to the subject, in *Metallurgie*, July 22, 1906, P. Röntgen bases his conclusions on laboratory experiments, carried out by melting together copper sulphide and

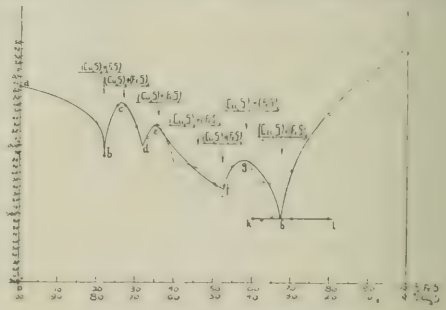


FIG. 2.—COOLING CURVE OF COPPER MATTE.

iron sulphide in different proportions in an electric furnace, for the purpose of constructing the cooling curve, the temperatures being measured with a Le Chatelier pyrometer. The investigation was supplemented by the microscopic examination of the products obtained. In making the melts it was observed that metallic copper crystallized out, which fact introduced a disturbing factor. The author gives the following explanation concerning this segregation of copper. In the liquid matter there is present Cu_2S besides FeS. During solidification a part of the FeS seizes upon the sulphur of a part of the Cu_2S and is itself converted into FeS_2. This change does not take place quantitatively, because the time of solidification is too short or the reaction approaches a state of equilibrium. The FeS_2 thus formed forms with a part of the copper sulphide a substance of different fusibility from that of a melt of Cu_2S and FeS, so that a part of the regulus separates out. He intends to investigate this problem further. The results of the investigation are given in the accompanying illustration, which represents the cooling curve. At *a* is the solidification point of the pure Cu_2S. The author deduces the following compounds from the appearance of the cooling curve: At *a*, Cu_2S; at *b*, a eutectic of the approximate composition $(Cu_2S)_1 + FeS$, $a + c$; at *c*, sulphosalt $(Cu_2S)_1$, $(FeS)_2$; at *d*, eutectic $(c + e)$; at *e*, sulphosalt Cu_2S, FeS; at *f*, eutectic $(e + g)$; at *g*, sulphosalt $(Cu_2S)_2$, $(FeS)_5$; at *h*,

eutectic (g + FeS); at i, FeS. Consequently, at least three chemical compounds would exist between Cu_2S and FeS. The branch $e - f$ of the curve is peculiar inasmuch as it is not convex upwards as the other parts of the curve. The author thinks that this is perhaps an indication that the curve runs different in reality; perhaps another maximum exists there, so that the curve would follow the dotted line. This part should be further investigated.

The Tacoma Copper Refinery.—This plant is located about 6 miles from Tacoma, Wash., on the shore of Puget Sound. It is driven throughout by electricity, the current of 40,000 volts tension being transmitted from the generating station at Electron, Wash., in the foot hills of Mt. Rainier. The current is reduced at the works to 100 volts, and about 2,000 hp. are utilized in the various departments. A brief illustrated description of the smelting works is given by Mr. D. A. Wiley in the *Engineering and Mining Journal*, July 28. A feature is a circular slag casting machine, which is intended to do away with much of the ordinary labor required to break up and remove the slag. The machine has a total capacity of 20 tons, 144 tilting molds being used. It is located at the side of the building where the furnaces are installed, and is served by an electric crane, which removes the ladles as fast as they are filled at the slag spout, and empties them into the molds. A reverberatory furnace is employed for the reduction of concentrates. The matte averages from 45 to 55 per cent copper. The molten matte is tapped into ladles, by which it is carried directly to the two converters, and blown by 3,500 cubic feet of air to each converter per minute. The furnace for casting the anodes is of the rotary type, and is heated by oil. It is also served by an electric traveling crane from the converter. The metal is cast into anodes weighing 125 pounds on an average. In the electrolytic department the cathodes are removed from the vats by an electric traveling crane and returned to the furnace department, where they are melted and cast into ingots, cakes, etc. The average daily output of the works is 30 to 43 tons. The ore smelting reverberatory has a capacity of 350 tons.

Lead.

Remarks on the Formation of Flue Dust and Furnace Accretion in Lead Smelting.—In *Metallurgie*, July 8, 1906, F. O. Doeltz and C. A. Graumann give a brief résumé of experiments made to determine the volatilization temperature of galena and lead sulphate. They summarize their conclusion as follows: 1. Galena is much more volatile than lead oxide and is volatile to a high degree already at a temperature of 869° C. At this temperature 18 per cent of loss were found in an hour. This fact furnishes an explanation of the large accretions of galena in the lead smelting furnaces; 2. The small veins of galena which are found in the walls of reverbatory or hearth furnaces when the latter are torn down may be formed by sublimation, that is, by the condensation of galena vapors; 3. Lead sulphate is not very volatile, but undergoes strong decomposition already at 1,000° C., contrary to statements made by other authors; 4. The lead sulphate found in the flue dust has in the main been formed subsequently from PbO and SO_3, or possibly PbS and O.

Zinc.

Remarks on the Distillation of Roasted Blende and the Calcination of Galmei.—Induced by the observation made at the Friedrichshütte smelting works in Upper Silesia, that roasted blende which had been exposed to the action of the atmosphere for a considerable time gave a better yield of zinc than other blende, F. O. Doeltz and C. A. Graumann conducted some experiments, the results of which they briefly describe in *Metallurgie*, July 8, 1906. They found that finely divided, cold, zinc oxide, kept evenly moist, can easily be converted into zinc carbonate by carbonic acid. The decomposition of zinc carbonate by heating is easily shown by baryta water, even at 137° C. A long continued, strong ignition is necessary for complete decomposition. The authors tentatively express the opinion that if roasted zinc blende, on account of remaining for a considerable period in an atmosphere rich in carbonic acid, takes up carbonic acid and is thereby partly changed into zinc carbonate, the amount of zinc carbonate exerts perhaps in the subsequent distillation process a similar favorable loosening influence, as the carbonic acid in the roasting and distillation of galmei. Graumann observed that when ZnO made from $ZnCO_3$ was pulverized, it did not become granular even after repeated ignition, but remained always loose and soft, and did not offer any resistance to pulverization. Quite contrary to this is the behavior of commercial zinc oxide as well as of the zinc oxide made from zinc sulphide.

Ore Milling in Wisconsin.—An article in the *Engineering and Mining Journal*, July 28, reviews briefly the practice and recent progress in the Wisconsin zinc-lead district. With regard to magnetic separation, the author states that the Wetherill separator has never been used in the field, owing to the royalty and the higher first cost. The two separators in regular use are the Cleveland-Knowles and the Dings, the latter of which is new and has only been used as yet at three places in the district. The separator has a low-intensity field and two magnets, a rougher and a cleaning magnet. The two types are stated to do about equally good work and to use about the same amount of power. Roasting is most important in preparing the ore for magnetic separation. Three roasters have been experimented with in the district, namely, the Dings, the Trego and the Galena. The latter alone is stated to have as yet established itself as a success. It operates on the principle of obtaining a slow, low roast, while the other two attempt a quick, high roast. It consists of a dust chamber and a fire-box, connected by a brick-lined iron cylinder, about 22 feet long and 52 inches diameter, rotating on tires by means of a gear wheel. The ore remains in the kiln from 2½ to 3 hours. It requires about 7.5 hp. to rotate the kiln. The roasted ore from the calciners is generally sprinkled with a small amount of water, so that the ore is perfectly dry by the time it reaches the separators. Before going to the separators the ore is sized by passing it through a trommel, one screen having ¼-inch and the other ⅛-inch openings. The sizes are kept separate, each going to a bin holding about 5 tons. One size and then the other is fed to the separator, this method of procedure permitting a better saving than the treatment of the fine and coarse together. The material obtained from the different separators throughout the district is stated to average about 59 to 60 per cent of zinc and carries about 3 per cent of iron, the saving being probably about 80 per cent of the zinc value. The middlings average probably between 7 and 10 per cent. at the different mills, and the tailings carry about 5 per cent. There is a decrease of lime in the finished ore, partly by dusting off in the furnaces and partly because some of the lime has enough iron in it to be separated by the magnets. The cost of magnetic separation is stated to vary from $1.00 to $2.00 per ton of the finished product, varying with the tonnage treated and the zinc contents of the raw ore. It is stated that at the Mills mine, near Hazel Green, a large concentrating plant is being erected, and a new method of magnetic separation is being tried. The concentrates will be roasted in a Trego calciner, and, when cooled, passed over an improved Wenstrom separator. A clean zinc product free from iron will be made, but in this procedure quite a bit of zinc will be drawn up with the iron. The impure iron product will then be passed by a shaking conveyor under a series of waving magnets, which will remove the iron and leave the zinc product in a fairly pure condition.

Iron.

Annealing and Hardening Furnace with Electrically-Heated Bath.—The process of annealing and hardening objects of steel requires a very careful supervision and regulation of the temperature, in order that the pieces may not

be detrimentally affected or perhaps spoiled altogether. In furnaces which are heated by ordinary methods, it is very difficult to obtain this precise regulation of the temperature, though it can be easily done in a furnace heated by electricity. Mr. L. M. Cohn, in the *Elektrotechnische Zeitschrift*, Aug. 2, describes an electric furnace constructed for this purpose. It consists essentially of a rectangular crucible, lined with refractory material, which receives the heating bath of molten salts. Two electrodes of soft wrought iron are introduced alongside of two opposite walls of the crucible. In order to prevent electrolysis, alternating current is used for heating the bath. A regulating transformer allows the precise regulation of the temperature. It has been a difficult matter to properly heat the high-priced qualities of steel, which have to be brought to a temperature of about 1,300° C. The electric furnace, however, can be brought to this temperature without any trouble. The electrolyte used for temperatures over 1,000° C. is pure barium chloride, the melting point of which lies at about 950° C. For lower temperatures, a mixture of two parts barium chloride and one part potassium chloride, the melting point of which is about 670° C., is employed. Pyrometric measurements, made at a furnace in practical operation, have shown that variations in temperature in the different parts of the bath cannot be detected, beyond a thin layer directly at the surface of the bath. The power consumed with a furnace, the melting space of which had a cross-section of 160 by 160 mm. and a depth of 175 mm., the primary voltage of the alternating current of 50 periods being an average of 190 volts, was 5.4 kw. for a temperature of 880° C., 8.5 kw. for 1,140° C., and 12.25 kw. for 1,300° C. The steel articles which are to be annealed or hardened are introduced into the bath and left in it until they have attained the color of the bath, which can easily be seen. It is claimed that the furnace is of advantage, not only in permitting the precise regulation of the temperature, but also by preventing the loss arising from spoiled material. Furthermore, the working of the furnace is very simple, and the heating proceeds much quicker than with the best gas furnaces, and, therefore, more work is done in the same time. In practical operation it has been possible to heat milling cutters of 120 mm. outside diameter, made of Novo steel, in 62 seconds to 1,300° C. The electrodes are gradually dissolved by electrolytic action, induced by the limited number of alternations of the current, which is usually 100 per second. The dissolved iron settles to the bottom of the furnace in the form of a slag, and can be easily removed with a spoon. The electrodes, when the bath is used at 1,300° C., have to be replaced after about 100 working hours, while they last about 1,000 working hours at 850° C. As they consist of ordinary soft wrought iron, the price is very low. The salts can be used for a long time, it being requisite to add about 1 kilogram per day, if the salts mentioned above are employed, and an average temperature of 1,300° C. is required. At larger intervals of time, say about every twelve months, according to the temperature at which the bath is used, it is necessary to replace the crucible. The costs of operation are therefore low, especially when compared with other furnaces for obtaining high temperatures.

Coke Ovens.

By-Product Coke Ovens in America.—In a paper read before the Engineers' Club of Philadelphia, and published in the July issue of the *Proceedings* of the Club, Mr. E. A. Moore gave a review of the growth of the by-product coking industry in the United States, and especially the work of the United Coke & Gas Co. and Dr. Schniewind. He briefly describes several plants, and notes the contrast between the older method of arranging a plant and the labor-saving appliances which have been introduced into the modern plants. The superstructure which supports the ovens has been materially improved by supporting them upon steel beam work arranged upon three longitudinal walls of concrete construction. The regenerators do not have to perform any more the duty of acting as main support to the ovens, and the contraction and expansion constantly going on in the oven brickwork can manifest itself without exciting such disturbing influence as formerly. Another material improvement has been the mechanical leveling device, which is mounted upon the pusher and operated by the same attendant. This device is of great service in maintaining the uniformity of the charge and insuring practically the same amount of coal being supplied to each oven. In combination with this is another labor-saving arrangement, on the more recent plants, which consists of a door-hoisting device so arranged as to be operated by the pusher man. The oven is discharged into a coke-quenching machine, a device made large enough to receive the entire charge of the oven practically unbroken. When the oven is pushed, the doors of the quenching machine are closed, and the quenching process is proceeded with by an arrangement of a water supply trench in combination with a motor-driven rotary pump which discharges the water into the cast-iron, water-jacketed box, in which the coke has been placed, in such a manner that the coke is instantaneously deluged with the quantity of water sufficient for quenching. The author states that the present capacity of the ovens contracted for under licenses of the United Coke & Gas Co. is 5,200,000 tons, and those contracted for by the Semet-Solvay Co. is 2,800,000 net tons of coal, producing on the basis of 75 per cent output in total 6,000,000 tons of coke; although some of the plants, notably the Cambria Steel Co. at Johnstown, Pa., get a considerably larger yield. He gives the following estimate of plant of 100 ovens, with a charging capacity of 7½ tons = 750 tons of coal daily, for beehive as compared to by-product ovens. *Beehive ovens*, daily results: Expenses, 750 net tons of coal, cost at mines $1 = $750; labor, repairs and all capital charges per ton of coal carbonized at 50 cents = $375, a total of $1,125. Coke output: Coal yielding 65 per cent coke = 487.5 net tons coke; cost per net ton coke, $2.50. *By-product ovens*, daily results: Expenses, 750 net tons coal, cost at mines $1 = $750; labor, repairs and all capital charges per ton of coal carbonized at 80 cents = $600, a total of $1,350. By-product returns (actual results): Ammonia, NH_3, 3,900 at 8.5 cents = $331.50; tar, 7,627.5 gallons at 1.8 cents = $137.30; benzol, less production cost = $187.50; gas, 3,472,000 at fuel value of 10 cents per 1,000 cubic feet, $347.25 = $1,003.55. This subtracted from $1,350 leaves $346.45. Coke output: Coal yielding 75 per cent coke = 502.5 net tons of coke; cost per net ton of coke, $0.01 6/10. Or, substituting 25 cents per 1,000 cubic feet for gas sold for illuminating purposes, it figures out as follows: Ammonia, $331.50; tar, $137.30; gas, 3,472,000 at 25 cents per 1,000 cubic feet = 868.00, a total of $1,336.8. This subtracted from $1,350 leaves $13.20. The cost per net ton of coke is, therefore, $0.02 35/100. The item of freight is ignored in this calculation, as the coke would offset the coal in either case.

Photometry.

Radiation Photometry.—In a paper before the British Association for Advancement of Science, reprinted in the London *Electrician* of Aug. 17, Prof. J. B. Henderson discusses recent advances in our knowledge of radiation phenomena and their bearing on radiation pyrometry. The main laws of radiation are four in number, and refer to an "ideal black body" (*i. e.*, one which emits and absorbs impartially all wave lengths, in contradistinction to bodies of selective radiation). These four laws are the Stefan-Boltzmann law of the total radiation given out by a black body, two forms of Wien's displacement law, and Planck's law of the partition of energy over the different wave lengths in the spectrum of a radiating body. All accurate radiation photometry is based on these four laws; the commercial radiation photometers employed especially either the Stefan-Boltzmann law by measuring the total radiation, or Planck's law by confining the measurement to a single well-defined luminous wave length (like the red line transmitted by

red copper oxide glass). These latter methods may be termed as optical pyrometry, because only the luminous portion of the spectrum is used. Pyrometers based on the measurement of total radiation have a black receiving surface for absorbing the radiation from the radiating body, and this surface may be one of the junctions of the thermocouple. In a perfect pyrometer of this kind the receiving surface should be ideally black, and in some very accurate laboratory investigations cavities have been used as receiving surfaces. But the error introduced in a practical instrument, due to this defect, is almost insignificant. Optical pyrometers confining the measurement to a single wave length are simply spectro-photometers of simple form. several commercial photometers are briefly described by the author, all of which have already been mentioned in our columns.

If a radiation pyrometer is to give correct absolute values of the temperature it must be remembered that its indications are based on the supposition of a black radiation body. Kirchhoff, to whom the foundation of the whole theory is due, showed long ago that the radiation inside an enclosure whose walls are at uniform temperature is the same as would be emitted by an ideal black body at the same temperature. If, then, we wish to obtain accurately the temperature of any material in a furnace by means of the radiation which it emits, we must take the radiation from a cavity whose walls are at the required temperature. If the material is in liquid or pulverous form, a tube closed at one end is inserted in the material and the radiation taken from the closed end. If the material is in ingot form, and there is no serviceable hole in it, we can use an auxiliary piece of similar material provided with a hole and placed against the ingot, so as to have practically the same temperature as the ingot. The hole, or cavity, may bottom against the ingot or in the auxiliary piece of material. To use a radiation pyrometer in the tempering of armor plate on a cooling curve, it would be necessary to lay another plate with holes in it on the one to be tempered, then heat the two together in the furnace, and after withdrawing them from the furnace to focus the thermometer on the bottom of one of the holes. When the required temperature is indicated the upper plate could be removed and the lower one quenched, the upper one being returned to the furnace with the next plate. In every case where the temperature is read while in the furnace, it is necessary to shield the radiation from luminous flames, after leaving the cavity. This may be done by means of tubes, and the temperature of the walls of these tubes will not affect the result, provided an image of the cavity is formed on the sensitive surface of the pyrometer.

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

Electric Furnaces.

Calcium Carbide.—H. L. Hartenstein, 12,519, Aug. 17. Application filed May 10, 1906.

Several patents of Mr. Hartenstein for the manufacture of calcium carbide have already been dealt with in our June issue, page 237. In the usual processes of making carbide lime and coke are the starting materials. In the present patent it is proposed to start with limestone rather than lime. According to the inventor, ordinary lime, as generally produced in kilns, is not of the uniform and high quality desirable for carbide manufacture. Further, in lime manufacture when the burning operation is completed the lime is permitted to cool down without in any way utilizing the heat stored in it. Finally, the crushing and pulverizing of the lime has certain disadvantages. It is, therefore, proposed to start with limestone, crush it to about 20 mesh, heat it to the point of incandescence in a revolving furnace, whereby the carbonic acid gas and other gases are given off, and then while hot, add coke, ground to about 50-mesh or finer. One part by weight of coke is used for three parts of limestone. The mass is then delivered to a reservoir, from which it is charged into one or more electric furnaces to be converted into carbide. In this manner the heat units stored in the lime during the burning of the limestone are utilized, and the amount of electric energy for the production of carbide is reduced. During the burning of the limestone a small portion of carbide dust or scrap, which is not suitable for other use, is added, in order to thoroughly expel and drive off any remaining carbon compounds and carbon dioxide. The carbide thus heats a superheating compound, and thereby serve to increase the temperature of the mass and to enable the mass to retain its heat when delivered into the distributing reservoir. The inventor adds to the following superheating compound to the heated mass during or immediately before the delivery of the mass into the distributing reservoir. It consists of 60 per cent calcium carbide, 20 per cent black oxide of manganese, 15 per cent bituminous coal, 2 per cent chlorate of potash, and 3 per cent of aluminium. The aluminium serves to drive off any phosphorus that may be present in the mass. From 15 to 20 pounds of superheating flux are used per ton of material.

Silica Glass.—J. F. Bottomley, R. S. Hutton, A. Page, 812,399, Feb. 13. Application filed March 21, 1903.

The object is the fusion of quartz and production of silica glass, the fusing point of which is above the temperature obtainable in gas or fuel furnaces as used in glass manufacture. The conditions of working are limited by the fact that by heating quartz or pure glass-makers' sand, unmixed with other substances, it is only possible to get it into a plastic state, and that further heating produces volatilization before a really fluid condition is obtained. In order, therefore, to produce articles of any desired shape, the inventors heat silica until it gets into a plastic condition by electric resistance heating, and then provide means whereby the plastic mass may be directly shaped or molded in situ into the desired form. One method of precedure is as follows: A hollow and perforated carbon rod, which serves as resistor, is embedded in a granulated mass of quartz. The Joulean heat of the current passing through the carbon rod brings the quartz next to it into plastic form, and then compressed air or gas is supplied through the hollow and perforated carbon tube, and the plastic quartz is thereby freely expanded and caused to assume any desired shape by blowing into an external mold. For details the reader must be referred to the patent specification.

Electrolytic Processes.

Lead Refining.—A. G. Betts, 827,702, Aug. 7. Application filed Oct. 31, 1902.

Details of construction and arrangement of electrolytic vats for lead refining. Claim 1 reads as follows: "In an electrolytic apparatus, the combination with an electrolytic vat; a bus-bar provided with relatively wide trough-shaped cavity and a plurality of wells of relatively small cross-sectional area opening in the bottom of said cavity, and fluid metal [mercury] in said wells; of a plurality of electrodes supported within said vat, and connections between said bus-bar and the respective electrodes having terminals immersed in the fluid metal in the respective wells."

Removing Oxide Scale from Iron.—C. J. Reed, 827,179 and 827,180, July 31. Application filed June 26, 1905, and Jan. 2, 1906.

To remove oxide scale from iron sheets, tools and wire, the latter are made cathode in a strong aqueous solution of an acid, preferably sulphuric acid of specific gravity 1.25 to 1.30, an acid content of 25 per cent, with a cathodic current density of 40 to 70 amps. per square foot, at a temperature of 150° C. Under these conditions the heavy scale on rolled iron or steel is completely removed in from 2 to 3 minutes. The chemical reaction which takes place is interesting, since the disappearance of the scale is not due to the reduction of the oxide to metallic iron. The oxide is reduced simply to a lower state of oxidation and electrolytically dissolved, ferrous sulphate

being produced. The solution of the oxide is effected without dissolving any of the iron, as in processes in which the article is made the anode. In patent 827,180 it is proposed to use an insoluble anode and to maintain sulphur dioxide in solution in the electrolyte, serving by its oxidation to both depolarize the anode and replenish the acid in solution. A diaphragm cell is employed, the sulphur dioxide gas being supplied to the anode compartment. By regulating the rate of supply of the gas the production of acid may be made to correspond with the rate of its consumption by the ferrous sulphate, the electrolyte thus being maintained at the proper concentration. As the amount of iron sulphate in the catholyte approaches saturation, the catholyte is withdrawn into a shallow pan and cooled to 0° C. The iron sulphate crystallizes out and the residual solution is returned to the anode compartment of the electrolytic cell.

Electroplating Cylindrical Articles.—R. C. Totten, 827,478, July 31. Application filed Jan. 31, 1905.

The apparatus is designed for uniformly plating circular or cylindrical bodies, such as metal-working rolls, treads of car-wheels, tires, pulleys, etc. The invention consists in details of construction of a revolving support for the articles to be plated, on which they rotate thus insuring the presentation of all points of the periphery of the articles toward the anode for equal lengths of time.

Sponge Plating.—F. R. Cunningham, 828,814, Aug. 14. Application filed Sept. 11, 1905.

In certain cases it is desirable to electroplate articles without suspending them in the vat containing the solution; for instance, when only a portion of the article is to be plated. Fig. 1 shows a glass tube A containing the anode 15 and a suitable quantity of the electrolyte. A sponge 8 is fixed in the terminal of the tube. By compressing the rubber ball B and immersing the brush or sponge 8 in the solution, a sufficient quantity of the latter is drawn up into tube A to partially or wholly cover the anode 15, and on applying the brush to the surface to be plated, connected with the negative pole of the supply circuit, the solution is discharged as required by the compression of the bulb. The quantity of solution applied is under complete control of the operator by means of the bulb.

FIG. 1.—SPONG PLATING.

Production of Chlorates.—A. E. Gibbs, 827,721, Aug. 7, 1906. Application filed March 25, 1904. (Assigned to National Electrolytic Co., of Niagara Falls.)

Clorates have hitherto been produced on a commercial scale by electrolyzing a chloride solution with platinum anodes in a cell without diaphragm and crystallizing out the chlorate thus produced, the best results being obtained when the operation is conducted at temperatures of about 70° to 80° C. The object of the present invention is to substitute carbon or graphite for platinum as anode material, and it is claimed that an ampere-hour efficiency as high as 96 or 98 per cent may be obtained. Graphite, it is said, was heretofore unsatisfactory for this purpose for two reasons. First, gases evolved at the anode break away particles of the graphite, and in the case of oxygen gas oxidize it; second, the required high temperature breaks up the graphite. The first trouble is overcome by using a diaphragm cell with a solution of chloride with addition of chromate in the anode compartment, this results in the simultaneous production of chlorate and bichromate by action of the chlorine upon the chromate, so that the evolution of the gas at the anode is almost suppressed. The second trouble is overcome by conducting the electrolysis at a comparatively low temperature, preferably below 37° C., removing the liquor from the neighborhood of the anode and heating. At low temperature the chlorine evolved at the anode reacts with the chromate according to the equation

$$6K_2CrO_4 + 3Cl_2 = 3K_2Cr_2O_7 + 3KClO + 3KCl$$

A portion of the hypochlorite splits up in the cell even at the low temperature; the remainder does so when heat is applied, according to the equation

$$3KClO = KClO_3 + 2KCl$$

Some or all of the alkali formed at the cathode is allowed to react with the bichromate at the anode, according to the equation

$$2KOH + K_2Cr_2O_7 = 2K_2CrO_4 + H_2O$$

So that the chromate is continually regenerated without increase or diminution of either chromate or bichromate, while the chlorate formation proceeds.

Electrolytic Treatment of Milk.—C. T. Willson, 829,303, Aug. 21. Application filed Jan. 30, 1906.

When a direct current is passed through milk from an anode of platinum or carbon to a cathode of nickel, the curd, in a form of a white substance resembling coagulated albumen, forms at the anode and is carried by the rising gas bubbles in small flakes to the surface, where it gathers. This curd consists of the albuminoids saturated with water and other constituents of the milk in solution. This action takes place until nearly all of the casein has been gathered. The whey then consists of water containing sugar of milk in solution and the remaining albumen. The curd is then removed. The temperature of the milk is artificially held constant.

Protective Coating on Iron or Steel Objects.—H. L. Hollis, 827,802, Aug. 7. Application filed July 7, 1905.

The iron or steel object is first made cathode in a solution of a caustic alkali for the sake of cleaning; it is then made anode and by oxidation a protective coating is formed. But during the formation of this protective coating foreign substances are apt to adhere to the subject; to remove them the objects are dried and immersed for a very short time in a bath of palm oil heated to 125° C. After removing the objects from the oil bath they are brushed or rubbed with bran. The oil treatment greatly lessens the danger of marring or scratching, and the objects will also resist rust for a much longer time.

Sulphate of Copper and Caustic Alkalies.—H. M. Granier, 829,778, Aug. 28. Application filed March 12, 1904.

Sodium chloride is electrolyzed in a diaphragm cell with iron cathodes and copper anodes. The cathode compartment contains pure sodium chloride solution and the cathodic product of electrolysis is caustic soda. The anode compartment contains a solution of sodium chloride saturated with cuprous chloride Cu_2Cl_2; the anodic product of electrolysis is cuprous chloride, which is precipitated in form of a white powder. (Since the anodic chlorine is thus immediately eliminated, it cannot lead to troublesome reactions with the caustic soda formed, like formation of hypochlorite, etc.) The cuprous chloride powder is collected and washed in water. The dry powder is treated with concentrated sulphuric acid up to the calcination point, resulting in the formation of hydrochloric acid and anyhdrous copper sulphate.

Gas Reactions.

Ozonizer.—E. L. Joseph, 829,790, Aug. 28. Application filed Dec. 12, 1905.

By means of the fan 6, driven by the motor 7 (Fig. —) the air is sucked through the filter or drying medium 2 into the ozonizer. There the current of air is obstructed by the wooden box 11, the back 12 of which is so shaped as to break up the air current and deflect it so that it must pass around the top, bottom and sides of box 11. In each of these four passages a panel (A and C at top and bottom respectively) are provided, each comprising a pair of electrodes 14 and 15;

each is formed of a sheet of non-oxidizable metal gauze, the two sheets of gauze being placed at opposite sides of a suitable

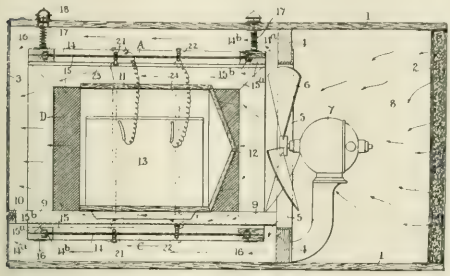


FIG. 2.—OZONIZER.

dielectric, like a sheet of mica. The two electrodes are connected with the high-tension terminals of a step-up transformer 13 in the box 11.

Combination of Gases.—D. R. Lovejoy, 829,872, Aug. 28. Application filed March 22, 1901. (Assigned to Atmospheric Products Co.)

The catalytic effect of contact substance, like platinum sponge, platinized asbestos, etc., is attributed by the inventor to their molecules being brought within acting distance of each other. To obtain the same effect it is suggested to make use of electrostatic attraction. If two gases are to combine, they are led into a chamber through separate metallic inlets which are electrostatically charged, one positive the other negative. During their passage through these inlets the molecules of one gas get positively charged, those of the other negatively charged. "On entering the chamber the positively-charged molecules of the one gas are attracted to the negatively-charged molecules of the other gas, and are thus brought within the range of attraction and a chemical combination thus effected." By this method gases may be caused to unite even when energy is absorbed by their union, as it is only necessary to charge the two sets of molecules to a sufficiently high potential, with respect to each other, to store in them sufficient energy to effect the combination after bringing them into chemical contact. Fig. 3 shows the arrangement. A is the chamber in which the gases combine. B are metallic inlets, surrounded by insulating sheaths D to prevent loss of electric charge by leakage. E are diaphragms of metallic or conducting gauze. C is the outlet for the combined gas. D and E are electrostatically charged from the dynamo F. As applications of this method the oxidation of atmospheric nitrogen and the formation of ozone are mentioned. In the latter case oxygen gas is introduced into A through both inlets B.

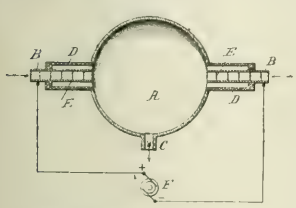


FIG. 3.—COMBINATION OF GASES.

BATTERIES.

Storage Battery.—G. A. Ford, 829,643, Aug. 28. Application filed Dec. 22, 1905.

The battery is built up of a series of plates in dish form one above the other. The plates are separated from each other by horizontal trays made of infrangible absorbent material, such as pulp board, and reinforced by a stiff frame. An electrode plate rests in each tray.

RECENT METALLURGICAL PATENTS.

SILVER.

Cyanide Process.—In connection with the article of Dr. J. W. Richards, published in this issue, on the metallurgical revolution in Guanajuato, Mex., two patents (828,287 and 828,288, of Aug. 7) are interesting, granted to Francis J. Hobson, of Guanajuato. The object is the extraction of silver from ores, in which the silver is present in combination with sulphur. The essential feature is the treatment of the ore with a solution, containing mercurous potassic cyanide, $KHg\,Cy_2$, which has a selective affinity for silver in combination with sulphur, the reaction being

$$AgS + KHgCy_2 = KAgCy_2 + HgS$$

The solvent employed by the author does not attack gold in any of the forms in which it appears to exist in the ore. (The above reaction is essentially different from the solution of gold and silver in potassium cyanide, since this requires free oxygen in the solution.) Two methods of producing the mercurous-potassic cyanide are described. One way is to add mercurous chloride Hg_2Cl_2 to a solution of potassium cyanide KCy. The other way is to add mercuric chloride $HgCl_2$ to the ordinary mill cyanide solutions containing ferro-cyanide of potassium, in which case the following two reactions occur in succession:

$$KFeCy_6 + 2HgCl_2 = Hg_2FeCy_6 + 4KCl$$

and

$$Hg_2FeCy_6 + 2KCy = KHgFeCy_6 + KHgCy_2$$

The most suitable strength of the solution is to some extent dependent upon the silver content of the ore and the form in which the metal is present. The inventor has obtained good results with solutions varying from 0.05 to 0.50 per cent.

VANADIUM.

Treatment of Vanadium Ores.—J. H. Haynes and W. D. Engle (828,850, Aug. 14) crush the vanadium ore to 12-mesh and then boil it with a solution of sodium carbonate for about an hour, until the vanadium contained in the ore is brought into solution; 100 pounds of sodium carbonate for 1 ton of ore for every per cent of vanadium gives good results. All the other elements present in the ore are undissolved, and are consequently left in the residue or tailings. The solution is drawn off and the vanadium is precipitated as calcium vanadate by the addition of water-slaked lime.

ZINC.

Distillation Without Retorts.—John Armstrong (839,283, Sept. 4) patents a zinc distillation process which has that common feature with the ordinary retort process, that the zinc must be oxidized and powdered and mixed with carbon for reduction. On the other hand, the process is intended to avoid the well-known disadvantages of retorts. The purpose is to operate without the use of retorts upon a large mass of material automatically and continuously, doing away with the intermittent system of charging and discharging small quantities of material in small retorts, and establishing a continuous flow of metallic vapor into the condensers, with uniform temperature in the reducing zones of the chambers, and also in the condensers. It is also claimed that the process permits the treatment of zinc ores containing lead with or without other metals. The furnace is constructed with upright chambers or cells, which are comparatively long, narrow and high, and are built with heating flues around them for the purpose of heating them by means of gas. The chambers are constructed in a row, and are surmounted by hoppers for charging. The charge after passing down through the long chamber, falls upon a revolving discharging apparatus, which discharges the exhausted residues, while a continuous supply of material is fed in at the hoppers at the top. The region of greatest heat is toward the lower portion of the chamber, the gas burners being provided at the bottom. The vapors are given off and pass through gills in the walls into a condenser filled with

coke or anthracite, where the metallic vapors descend, and are condensed into the metallic liquid state and percolate through the apertures in the bottom of the condenser. The carbon monoxide gas is carried away from the condenser under a baffle-wall up into an exit tube. The gills in the walls between the reduction chamber and the condenser have a cross-section of the shape of a reverse V, serving the double purpose of keeping the coke or anthracite on one side and the powdered charge on the other without mixing and of allowing free descent to both and a clear and uninterrupted path for the metallic vapors through them. The chambers being continually full of the powdered charge up to the hopper at the top, an effectual seal is thus provided against the atmosphere entering the chambers or condenser, there being constantly a plus pressure inside these parts. The chambers being constructed comparatively narrow, the intense heat penetrates the charge so that dense metallic vapors are constantly evolved, giving a steady and regular metallic condensation without the loss in pressure and oxide, which is inevitable with the retort method.

Gold.

Treatment of Slimes.—To separate a cyanide solution from slimes, W. T. Weekly, of Kalgoorlie, Western Australia (830,388, Sept. 4), proposes a plant which, viewed as a whole, may be classed as an automatic charging and discharging slime-treating machine, which with a little driving power and a minimum of supervision handles a comparatively large quantity of slimes in a given time. It comprises a series of trays to receive the material to be filtered, these trays being suitably supported and forming a series with lower trays. Each series is reversible. When the lower trays become, by transposition, the upper ones, they are in turn charged with material. Piping is so attached that when the upper trays are charged with slime, a series of vacua may be formed (or the air pressure may be so reduced) in chambers under those trays that liquid or solution will be caused to filter through into the lower chambers (called "vacuum chambers"), and thence to a receiver or one or other of a series of receivers. The vacuum chambers have apertured top plates of strong perforated metal, over which is spread a filtering medium. The trays, or tray-supports, are arranged on standards or carriers called "trucks," adapted to move in a circuit or along any suitable course on rails. Such vibration as the trays receive in their motion is beneficial. The trucks are connected by piping to a main or central chamber or chambers, any suitable vacuum pumping means or the like being used. By arranging the tray trucks or vacuum filters in and moving them repeatedly around a circuit, the process may be carried on so as to filter from a number of trays at once, successive charges of slime being fed to each tray and the resultant cakes of mineral discharged. When it is found that by increasing the speed of rotation higher extraction results are secured, the speed is very easily raised, with the further advantage that a relatively greater weight of slime will then be treated in a given time. The trucks have perforated, or drilled shafts or the like, through which passages connecting to the trays extend. One passage is a solution-conveying passage, and the other is an air-admission passage. When a pair of filters are moved so that the upper one becomes the lower, its contents cease to be subjected to the vacuum or suction and become subjected to air pressure. This causes the cake to fall out from what has become the lower tray. The filter cakes thus fall in a given place and may be readily removed thence by any practicable means.

Furnace Construction.

Blast Furnace. L. Heckscher (828,310, Aug. 14) describes means for cooling the walls of blast furnaces. He utilizes what is known as "outside cooling" for the protection of the brickwork of the walls, both above and below the mantle, in such a manner that the entire upright part of the furnace is cooled in addition to the bosh, thus particularly obviating the dangers due to coils in the "cooling plates," now generally embedded in the brickwork of blast furnaces.

Hot Blast.—Thomas Kiddie (830,152, Sept. 4) patents a hot blast system, in which the air from the blower is brought in contact with the heat from the waste gases of the furnace; the special feature is the endeavor to expose the air blast in a gradual manner to the waste heat of the furnace, exposing it first to the heat of the dust chamber or the flue farther from the furnace, and thereafter to the greater heat of the direct gases close to the furnace. On account of the long exposure and gradual elevation of temperature the air tube and jacketed linings are said to last longer, in comparison with the short travel at high temperatures in cast-iron pipes, as in the ordinary hot-blast stove. By introducing a damper in the furnace stack the velocity of the gases through the flues and dust chamber is decreased to a minimum, so as to insure a full utilization of the heat; by this means the amount of cold air drawn in at the feed doors is also correspondingly reduced and a more complete settling of the particles of flue-dust is obtained.

BOOK REVIEWS.

Catechism on Producer Gas. By Samuel S. Wyer, M. E. New York: McGraw Publishing Co. 42 pages. Illustrated. Price, $1.00 net.

This little book, which fully explains the fundamental principles of gas producers for power and fuel purposes, ought to be in the hands of every prospective user of power, as it explains in a simple way the chemistry and mechanical principles of gas producers. It enables the reader to understand the pros and cons of gas producer builders, and to form an opinion of his own. On that account this booklet will do a good deal towards educating the public in the gas power question, which certainly is now of general interest.

Illustrated Technical Dictionary. In six languages: English, German, French, Russian, Italian, Spanish. By K. Deinhardt and A. Schlomann. Vol. I.: Elements of Machinery and the tools most frequently used in metal and woodworking. By P. Stülpnagel. New York: McGraw Publishing Co., 1906. 403 pages. Price, $2.00.

This is the first of a series of technical dictionaries which, according to the programme, will be at least ten in number, each volume dealing with a separate industry and being complete in itself. The first volume covers elements of machinery and machine tools.

The arrangement is novel and has various decidedly recommendable features. The main part of the volume (256 pages) is not alphabetically arranged, but is classified rather like a hand-book. This par consists of three main divisions. The first division deals with machine elements and is sub-divided into twenty-three chapters, treating respectively with screws and bolts, keys, rivets, etc. The second division deals with tools, and is sub-divided into sixteen chapters, treating respectively with vises, tongs, anvils, etc. The third division is an appendix, dealing with engineering, drawing and miscellaneous matters.

Each chapter again contains the chief technical terms, as much as possible logically arranged. Each word is given in German, English, French, Russian, Italian and Spanish. Thus, to select one chapter at random, chapter 8 on couplings, enumerates the different types of couplings used in practice. The most remarkable and novel feature is, however, the general use of diagrams, formulas and symbols to explain the different terms. Thus in chapter 8 the different types of couplings are each illustrated by a diagram. Moreover, in drawings of more complicated apparatus the different parts are marked by letters, and in translating the terms for the different parts reference is made to these letters. This system has been carried out with great consequence and with so much

success that the authors could have claimed to give not only a dictionary in the six languages—English, German, French, Russian, Italian, Spanish—but one which translates all these languages into a seventh one, the universal language of engineers the world over—that of diagrams and formulas.

The second main part of the book (147 pages) contains two alphabetical indexes of all the words contained in the volume, one index comprising the German, English, French, Italian and Spanish languages, while for the Russian language a separate index is provided (on account of the different type). In this way it is easy to find any word in the first, hand-book-like part of the book, and the single volume, arranged according to this system replaces thirty bilingual dictionaries.

With respect to accuracy of translation the dictionary is excellent. We might almost say it is the first reliable technical dictionary that we have seen. The trouble with almost all technical dictionaries seems to be that when a new term appears and the author of a dictionary wants to take it in and does not know its meaning, he does not always take sufficient trouble to inquire of a responsible party, since it is so much easier to make a literal translation. Now, this is bad enough, but what is worse, "es erben sich Gesetz und Rechte wie eine ewige Krankheit fort," and it seems to be the same with terms in technical dictionaries; for when once such a foolish literal translation has appeared in one dictionary the authors of all others almost invariably copy it for their next "revised and enlarged" edition. This is the only explanation why a series of technical dictionaries give, for instance, as translation of "Drehstrom" the literal translation "rotatory current," which is even found in the latest special electrical dictionary, that by P. Blaschke, with a preface by Prof. F. Niethammer.

The dictionary under review is absolutely free from such ridiculous nonsense. We have found it reliable wherever we have used it. The only instance that might be mentioned in this connection is the English translation of "Flusseisen" (page 213), which is given as "ingot-iron." This is correct, since it is the officially adopted English term. But it is, nevertheless, a fact that it is scarcely used in this country, and that American iron and steel men almost invariably speak of "soft steel" or "mild steel" where the Germans speak of "Flusseisen."

Fuel Briquetting.

The problem of making briquettes has attracted numerous inventors, since its successful solution gives commercial value to the large quantities of fine ores or fine fuel, which are otherwise commercially valueless. We are glad to report some interesting recent developments in the introduction of the Mashek system of fuel briquetting.

The Traylor Engineering Co., of New York City, which has recently begun the manufacture of fuel briquetting machinery, and the installation of fuel briquetting plants under the Mashek system of briquetting, has already received several important orders. Three of these orders, for which the machinery is now being built at the company's works at Allentown, Pa., are, respectively, for Edward B. Arnold, No. 1 Broadway, New York, for his plant at the foot of West Forty-eighth Street, New York; the Semet-Solvay Co., for its Detroit plant; and the Montreal Light, Heat & Power Co., of Montreal, Canada.

These plants represent three interesting variations of the briquetting industry. Mr. Arnold is a large coal dealer. His plant is to make briquettes for domestic consumption, using the small sizes of anthracite coal. The Semet-Solvay Co. will make first-class fuel for domestic use from the coke breeze, which is a waste product from the Solvay by-product coke ovens, and the Montreal Light, Heat & Power Co. will make a smokeless briquette for family use from coke breeze resulting from gas making, using coal-tar pitch as a binder.

The adoption of the Mashek process and machinery by these three concerns is of particular importance [illegible] and has been experimenting for a long time with the best and most used European briquetting machinery and machinery of other design until they were all of them convinced with every system in use. Although the foreign process and machinery may be satisfactory abroad, they have proved somewhat like a commercial failure here. The briquettes produced are not of a size or shape to suit American [illegible] some of them requiring to be broken up before use. There was also too great because of the small product of the plants, the large amount of binder required, and the cost of the necessary attendance.

The Mashek plants and machinery are entirely automatic in their operation, no hand labor being required from the time the materials are received until the finished briquettes are ready to be delivered to the consumers. The briquettes are of a size and shape suitable for immediate use and can be easily handle by chute or shovel, and the production is nearly as large per hour as is the daily output of the foregoing plants. The briquettes are as hard and clean as anthracite coal and equal the best qualities of anthracite for use.

The cost of briquetting under this system has been demonstrated in plants now in operation to be from 75 cents to 92 cents per ton, including pitch for binder, fuel for the briquet dryers and heaters, lubricating oil, wear and tear and labor—in fact, including everything but the cost of the raw material to be briquetted. The system operates equally well on anthracite or bituminous coal, coke breeze or lignite. The design of the machinery and the installation of plants is under the personal supervision of Mr. G. J. Mashek, the inventor and patentee of this process and machinery.

Heavy Rolling-Mill Engine.

The Allis-Chalmers horizontal type rolling-mill engines have been for years used in many of the largest iron and steel mills of the country. Two units, larger than any heretofore built, however, were ordered some time ago for the Sharon, Pa., plant of the Carnegie Steel Co., and are now being built at the West Allis Works of the Allis-Chalmers Co., Milwaukee.

The engine frame and slide for the first engine was recently cast at the West Allis foundries, and has proved to be the largest and heaviest single piece ever poured in these shops, where monster engine castings of every size and kind are produced daily. The single piece for frame and slide in this case, however, weighed, roughly, 105 tons, and is designed for an engine whose cylinders will measure 50 inches and 78 inches in diameter and have a stroke of 60 inches.

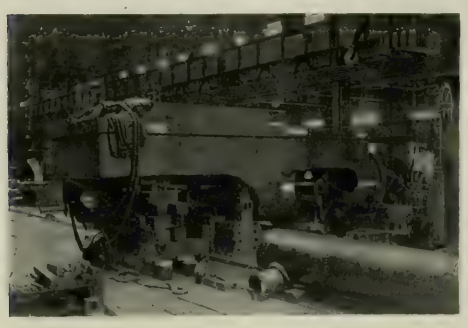
The pattern for this single casting measured 32 feet long, 11 feet wide and 10 feet high, representing the work of ten expert pattern makers for a period of over four months. The amount of lumber used in its construction reached the surprising aggregate of 22,000 feet and over. The pit in which the casting was poured measured 40 feet long, 15 feet wide and 11 feet deep into which were poured some 105 tons or more of molten metal, the pouring of which consumed seven or ten minutes' time. Nine ladles, four with a capacity of 25 tons each, one 13 tons and four 5 tons each, were used in the operation.

In building a rolling mill engine with a capacity like the one in question, there is a marked advantage in designing the frame and parts in a single piece for the purpose of securing the greatest possible rigidity to withstand the racking strains of rolling mill service. The remarkable size of this piece, however, required an extra expense of ingenuity in the handling of it during and after its production, which might well have proven a serious question. In the first place, a weight of this kind being very exceptional, extra precautions were taken against any mishaps in the process. Three cranes, with ample overload capacities and a rated aggregate of 145 tons, were used in conjunction to lift the casting from its pit. A

special tackle was devised so that the lifts should be straight upward, applied at the ends of the heaviest or trussed steel lifting bars.

The casting, after being deposited beside its pit, was allowed to cool for fifteen days before the operation of cleaning it was begun. The heat still given off after approximately twenty days' cooling could be felt several feet away from the huge mass. During the cleaning process, three men were able to work abreast inside the slide aperture while standing upright and without perceptible crowding.

In anticipation of the difficulty of handling a piece of this size, special cars, two in number, were ordered. One of these cars is to be used to carry the casting to its destination. In handling the huge mass after cleaning, nothing but a



PROGRESS THROUGH THE WORKS OF A 105-TON CASTING FROM FRAME AND SLIDE OF HORIZONTAL ROLLING-MILL ENGINE.

straight lift was attempted to load it upon the special car mentioned, which was brought to the side of its prospective burden over new and special track. The car was then used to convey the casting to machine shop No. 1, where the process of machining is now going on. The mass of metal is placed upon the floor of the machine shop and the heavy machine tools used in the finishing are brought to it.

The cars mentioned as being specially built for the Allis-Chalmers Company are unique in design and appearance, so they deserve brief mention. They are the concrete results obtained from the solving of a difficult transportation problem which confronted the engine builders. The question of adequate carrying facilities for heavy engine parts has become more and more persistent with the growth of the present tendency toward giant units. The cars used heretofore for the purpose, constructed of heavy material and with capacities of from 60 to 70 tons, having proven inadequate. The cars utilized up to the present time were, in fact, identical with those employed by the Government for the transportation of heavy ordnance. In seeking a suitable means for overcoming the difficulty of shipping such a piece, it was learned that no single car had ever been built with a capacity capable of carrying it, so that a special car, designed particularly for the purpose, would be the only possible solution. Accordingly, orders were placed for two 16-wheel flat cars of 100 tons capacity, which are the first of their kind in existence. A few details of construction showing the unusual sizes of these cars are as follows:

Length over end sills, 40 feet 2 inches
Width over side sills, 8 feet 9 inches
Height, rails to floor, 4 feet 4½ inches
Wheel base, 36 feet 2 inches
Wheels, diameter, 33-foot stand
Journals, 5½ inches x 10 inches
Weight of car estimated between 50,000 and 60,000 pounds
Rated capacity, 200,000 pounds

Magnetic Thermometer for Use in Hardening Steel.

At the recent York meeting of the British Association for the Advancement of Science, Mr. W. Taylor described an ingenious magnetic indicator of temperature, for use in hardening steel.

In his first experiments, the steel bar to be hardened was part of a magnetic circuit energized by an alternating-current coil on the yoke. Another bar, outside the furnace for heating the bar to be hardened, was in magnetic parallel with the steel bar. Each of these bars in parallel are surrounded by what may be called secondary coils, connected in series so that the currents induced in them are in opposition; the coils being such that, when the bars are cold, the induced currents are equal and opposite, and therefore no sound is perceptible in a telephone kept in the electric circuit.

Now since, for the purpose of hardening, steel has to be raised to about the temperature at which it loses most of its permeability, and since the loss of permeability by the heated steel destroys the balance of the induced currents described above, therefore the appearance of sound in the telephone gives notice of the moment for quenching the steel.

A rougher form of apparatus, which has been in daily use in the tool room for about four months, was described by the author. In this a permanent magnet produces a flux through a good magnetic circuit that includes the steel to be hardened. An alternative path is offered to the flux, but this path contains normally a small air-gap, which, however, can be closed by the rocking of an iron lever pivoted at its center at one end of the permanent magnet. Each half of the rocking lever is part of the two magnetic circuits mentioned. When the steel is heated till it loses its permeability the flux through the alternative path becomes big enough to rock the lever, and this rings a bell to notify the workman.

Notes.

Jubilee of the Coal-Tar Industry.—The Perkin memorial and international jubilee of the coal-tar color industry was recently celebrated in London. The American celebration is to take place in the Chemists' Club, New York City, Oct. 6. The next meeting of the American Electrochemical Society will be held at Columbia University, New York City, on Oct. 8 and 9, so as to give members an opportunity to attend both occasions. Full programs will be given in our next issue.

Belt Conveyors.—Bulletin No. 14 of the Robins Conveying Belt Co. gives a concise description of the design and operation of Robins belt conveyors in handling excavated earth and rock and concrete materials, with some very interesting illustrations of equipments in actual practice.

Filter Papers.—We have received samples and price lists of "ashless, chemically pure, toughened and starch-free" filter papers, made by Max Dreverhoff, in Dresden, Germany. This concern is represented in this country by George D. Feidt & Co., of Philadelphia.

Earthenware Exhausters.—We have received a reprint of a paper by Prof. George Lindner, on comparative trials of earthenware exhausters of the Deutsche Steinzengwaarenfabrik für Canalisation und Chemische Industrie of Friedrichsfeld, in Baden, Germany, which concern is represented in this country by Messrs. Fred. Bertuch & Co., of New York City. The paper of Prof. Lindner had been originally published in the *Chemical Trade Journal*, and gives a full account of the tests in form of tables and diagrams.

Courses in Industrial Chemistry.—The Pratt Institute, of Brooklyn, N. Y., has issued an illustrated pamphlet on their two year day course in industrial chemistry and their three-year evening course in technical chemistry. Classes begin work on Sept. 24 and 26 respectively. The pamphlet contains, besides a program of the courses, an illustrated description of the equipment of the chemical laboratories of Pratt Institute.

Oil Engines.—The borough of Kutztown, Pa., after operating a municipal lighting plant for some time, is increasing its capacity by the installation of a 125-hp. Hornsby-Akroyd oil engine. The present plant consists of a 65-hp. Hornsby-Akroyd oil engine belted to generator. Both engines were supplied by the De La Vergne Machine Co., New York. This company has recently issued a nicely illustrated pamphlet on the Hornsby-Akroyd oil engine, its design and operation. The ability of this engine to run successfully on crude and fuel oil places it in a class by itself, and has been no small factor in bringing about the installation of over 14,000 engines which are now in successful operation. These engines are built by the De La Vergne Machine Co., in single cylinder units up to 125 hp. in size, and in twin cylinder units up to 250 hp.

Steel-Hardening Metals.—A recent pamphlet of the United States Geological Survey deals with the production of steel-hardening metals in 1905, J. H. Pratt being the author. The following metals are dealt with, figures being given for home production and importation: Nickel and cobalt, chromium, tungsten, molybdenum, vanadium and uranium, titanium. Of these only tungsten is produced from ores in this country to any considerable extent, the United States production of tungsten having increased from 46 tons in 1900 to 292 in 1903, and 803 tons in 1905. Although there was only a very small production of nickel ore in the United States during 1905, there was a considerable quantity of matte and ore refined, which was imported from Canada and New Caledonia. The nickel obtained from this source is very much in excess of the quantity consumed in the United States, and there is therefore a large export of the metal.

Electrical Equipment of Iron and Steel Plants.—We have already briefly referred to the proposed electrical equipment of the new Gary, Ind., steel plant of the United States Steel Corporation. Since this plant is a splendid illustration of the remarkable extent to which the use of electricity for power purposes has been developed in the inumerable industrial plants of the country, the following details will be of interest: When completed it is estimated that the Gary plant will handle substantially 5,000,000 tons of ore a year, and produce annually approximately 2,500,000 tons of steel. There will be sixteen blast furnaces of 450 tons daily capacity each, and eighty-four basic open-hearth furnaces of 60 tons. The necessary electrical generating equipment capable of handling such an output is to have an initial capacity of 18,000 kw., and will be so designed that extensions may be added indefinitely at one or both ends. The initial equipment will have a capacity of 18,000 kw., of which 14,000 kw. are in 2,000-kw., 25-cycle, 2,300-volt units, and 4,000 kw. in 2,000-kw., 250-volt direct-current units. These generators are now on order from the Allis-Chalmers Co., Milwaukee, and they will be direct coupled to nine Allis-Chalmers horizontal twin-tandem gas engines. The power house building for the present is to be approximately 700 feet long, with a span in the main building of 88 feet. An 18-foot extension under the same roof through the entire length of the structure, has been planned in order to provide the necessary room for high-tension switches. The power house will be located immediately adjacent to the blast furnace blowing engine houses, and between the blast furnaces and the open-hearth furnaces, most advantageously placed for fuel supply and for securing a minimum length of transmission lines to the various departments using electric power. The Allis-Chalmers alternators and direct-current generators will be built at the company's electrical works in Cincinnati.

The Traylor Engineering Co. and the Traylor Manufacturing & Construction Co. have been consolidated into one corporation, under the name of the Traylor Engineering Co. The new corporation is organized under the laws of the State of New York. The public will continue to find the Traylor Engineering Co. at its offices in the Engineering Building, 118 Liberty Street, New York, and with its constantly increasing staff of experts always ready to give advice on mining, metallurgical, cement making, fuel briquetting and rock-crushing matters. The Traylor Engineering Co. has also recently entered into a contract whereby Shirley & Grant, of Reno, Nev., become exclusive representatives of that house for the State of Nevada, for the sale of their machinery, for mining, sampling, rock and ore crushing, milling, cyaniding, amalgamating, smelting, refining, fuel briquetting and cement making. A close relationship has been established under which the two houses will co-operate in designing and building entire plants, as well as in the sale of individual pieces of machinery. Shirley & Grant are well known in the Nevada territory, both as a firm and as individuals, and they have many friends among the mining fraternity. Messrs. Victor M. Braschi & Co., of the City of Mexico, who also represent the Ingersoll-Rand Co., the A. S. Cameron Steam Pump Works, and the Erie City Iron Works, have taken an extensive agency for the Traylor Engineering Company in Mexico.

Lungwitz Zinc Process.—As our readers will remember, a large-scale test was made last year at Warren, N. H., with the Lungwitz zinc process, the essential feature of which is the attempt to smelt zinc ores in a blast furnace under strong pressure so as to get the zinc in liquid form. Last year the tests were checked by various difficulties, and the results of the tests were believed not to be conclusive. Now these tests are to be resumed shortly.

American Mining Congress.—The annual meeting of the American Mining Congress will be held in Denver, Col., from Oct. 16 to 19.

Dry Blast.—The first European application of the Gayley dry-blast process will be made at the works of Guest, Keen & Nettlefords, in Dowlais.

Things Chemical.—This monthly trade publication of the Charles E. Sholes Co., of New York City, continues to bring interesting matter in form of short notes from chemical practice (for instance, on pickling and galvanizing iron, in the July issue, denaturized alcohol versus turpentine in the August issue). Other notes are written in a quite amusing style, and on the whole the publication is extremely well edited in the interest of the concerns which it represents. After the May issue had brought the portrait of Mr. Charles E. Sholes, the July issue contains that of Mr. Edward J. Duggan, vice-president of the Charles E. Sholes Co.

Analyzed Chemicals.—A small but interesting booklet, entitled *Baker's Analyzed Chemicals*, contains the detailed analyses of 161 chemically pure products of the J. T. Baker Chemical Co., of Easton, Pa. The analyses are given to show the presence or absence of suspected impurities and estimation of quantities when present. The figures given show the average results obtained from different lots.

Large Electric Steel Plant for Sweden.—According to the Stockholm correspondent of the *Frankfurter Zeitung*, there is a probability that the Kjellin electric furnace for the production of steel will be utilized on a much larger scale than heretofore in Sweden. A number of capitalists are said to have proposed to the Swedish government to organize a stock company to make steel under the Gröndal-Kjellin patents, provided the government will develop the immense water power at the Tralhätta waterfall, and also that in the Province in Norrland, in which there are extensive ore deposits. It is proposed to lease these water powers from the government, and to produce steel in such quantities as to make Sweden a steel exporting country. The power from the Tralhatta Fall will be brought to a point near Gothenburg, where a steel plant producing 500,000 tons yearly, according to the optimistic claims of the promoters, will be established. The works in Norrland will have about the same capacity. The two, it is claimed, will be able to supply all the steel required for home consumption, and it is hoped that the steel will be cheap enough to make export possible. The government is said to have accepted the proposi-

tion, and to have decided to commence work to develop power at the Tralhätta Fall. It is expected that 10,000 to 15,000 hp. will be available there as early as the year 1908. The new company has been given the title "Metallurgiska Patent Aktiebolaget." The Krupp works is reported to have acquired the Kjellin patents for Germany, and it is proposed to erect a plant for their utilization. In connection with these notes, which we find in *Iron Age*, it may be remarked that the process of Groendal, alluded to above, is for concentrating and briquetting iron ores, while the Kjellin process is for refining steel in an electric induction furnace; it has been described in detail repeatedly in this journal.

Coöperation in Physical Chemistry.—At the recent meeting of the American Chemical Society in Ithaca, Prof. W. D. Bancroft made some timely suggestions on coöperation in physical chemistry, as summarized in a recent issue of *Science:* We could be of more assistance to each other if we had a system of reports by which we knew what bits of research work the others were doing. Each has stored away in his memory a number of generally unfamiliar facts which he has stumbled upon in his reading or in his laboratory. These may not be important enough to him to justify his doing enough work to get anything worth publishing, or he may have more important matters on hand and so lack the time. If now any one of us learns that any one of the others is doing a bit of investigation into which this, that, or the other fact fits nicely, the observation can at once be turned over to the man who can use it, much to the benefit of both parties. It is probable that nobody goes to one of the meetings of the Chemical Society without getting a few suggestions of value to him. On the other hand, owing to the great distances and consequent expense, we do not get together as often as we should like. If we kept more in touch, we should be getting continually some of the advantages which we now get from the occasional meetings. The matter would not be difficult to arrange. In October and February each man could make out a list of the work planned or in operation. These reports could be manifolded and distributed. So far as I can see, the plan has practically no objectionable features and might be of great value. It seems, therefore, worth trying.

"By-Products."—According to the *Chemische Zeitschrift*, the following very interesting remarks were made by Commercienrat Grimberg, of Zeche Lothringen, in Bochum, Germany: Prof. Ostwald's contact process for winning nitric acid from coke-oven gases will be used on a large scale in a plant which will cost about $90,000, and which will be ready by Nov. 1. That tar, ammonia, benzene are somewhat more than "by-products." is shown by the annual report of the Gewerkschaft König Ludwig. The annual profit of $400,000 was made up of $167,500 from coal mining, $39,425 from coke making, and some $185,000 from the by-products, which, therefore, made up almost half the profit. Throughout the Ruhr mining district new plants are being erected, and the by-product coke oven installations are constantly increasing in number and importance.

Aluminium in Europe.—Like the Pittsburg Reduction Co. in this country, the British Aluminium Co. finds it necessary to increase the capacity of their works. Having fully utilized the power from the Falls of Foyers in Invernesss-shire, they have commenced a far more ambitious scheme. In the neighborhood of Kinlochleven is an area which has one of the largest rainfalls in the whole of the British Isles. The catchment basin in the position chosen has a large area, in the center of which is the site for the reservoir, at an elevation of 1,000 feet above the sea level, and only distant about 5 miles from the coast. According to the London *Electrician* the reservoir will be about 7¾ miles in length, with an average breadth of ½ mile. It will obliterate three small lakes, and its capacity will be about 20,000,000,000 gallons. The dam will be formed of concrete, and will be over ½ mile in length, and its greatest height in the center will be about 80 feet. The cost of the undertaking will exceed $2,000,000.

Personal.

Prof. Max Le Blanc has been appointed Prof. Ostwald's successor at the University of Leipzig, while Prof. Fritz Haber becomes Le Blanc's successor in Karlsruhe. Both Prof. Le Blanc and Prof. Haber are members of the American Electrochemical Society. Prof. Haber is well known personally to many American electrochemists and electrometallurgists from his extended trip through this country in 1902.

Mr. W. Mc A. Johnson is leaving for an extended tour of the West, to look into the subject of zinc concentrators with reference to their bearing on the metallurgy of zinc, in connection with the continuous zinc furnace which he has developed. Mr. Johnson will visit Kansas, Colorado, New Mexico, Arizona, Nevada, British Columbia and old Mexico, and expects to return about the first of December.

Mr. H. Hullegard has opened engineering offices in Ousby, Sweden, and will be specially interested in representing American manufacturers of machinery and apparatus for metallurgical and chemical work. Mr. Hullegard is not a stranger in this country. After graduating from the Royal Technical University, in Stockholm, as chemical engineer, he came to this country in 1900, and found employment with the Orford Copper Co., in Bayonne, N. J. He worked for a number of years in their various laboratories, and was for a couple of years in charge of their plant for winning the precious metals, gold, silver, platinum and paladium, from their Canadian nickel ores. Simultaneously he carried on the company's extensive experimental work on electrolytic nickel refining, and assisted Mr. V. Hybinette in working out the practical details of his new electrolytic nickel refining process, which was described in our January issue, page 34. For some time he also had charge of the electrolytic nickel refinery. Mr. Hullegard returned to Sweden last year, and opened an engineering office, representing American, German and English manufacturers.

Digest of U. S. Patents Prior to July, 1902.

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Since a large amount of material has already been published in this department, a concise index of the subjects dealt with in the past should be useful, no such index having been given before:

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Diaphragm cells for the production of caustic alkali, Vol. I., pp. 114, 152, 196, 232, 268, 300, 340.

Electrolytic production of bleaching and disinfecting liquids, Vol. I., pp. 368, 401, 433.

Electrolytic production of chlorates, Vol. I., p. 433.

Electrolytic production of white lead, Vol. I., pp. 474, 518.

Electrolytic production of dyes and pigments, Vol. I., pp. 518, 555.

Electrolytic production of various chemicals, Vol. I., p. 598; Vol. II., pp. 43, 80, 123, 210.

Production of aluminium from molten electrolytes, Vol. II., pp. 165, 210, 253.

Electrolysis of molten sodium salts, Vol. II., pp. 296, 333, 384, 434.

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Production of ozone and miscellaneous gas reactions, Vol. IV., pp. 162, 204, 252, 290, 338.

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The Wasteful Combustion of Coal.

Coal mining is the greatest mining industry of this country. The vast sedimentary rocks of the carboniferous age throughout North America carry from coast to coast large deposits of coal varying from graphite and anthracite to the brown lignites of Texas and Dakota. The aggregate tonnage is almost four hundred million tons per annum (three hundred and ninety millions in 1905), giving direct employment to an army of grimy miners, and indirectly to a still larger army in the industrial world. Coal is potential energy—the "stored-up sunlight," as Stephenson said, of past ages. The means for converting this treasure into actual and commercial energy should be the most efficient and economical that chemical knowledge and engineering skill can devise. Yet, on the contrary, we find the means usually employed most wasteful and inefficient. Let us consider the ordinary burning of soft coal under a boiler for raising steam. There are two operations—the coking or distillation of the volatile hydrocarbons and their combustion, and next the combustion of the coke (or fixed carbon in non-coking coal). The air from under the grate passes through the coke, and is, in part, turned into producer gas. It thus diminishes the activity of the oxidizing power of the atmosphere above the incandescent bed. These gases, not well mixed, pass on the water cooled metallic surface of the boiler, and combustion is stopped. For complete combustion, a high temperature for some interval is required, while the cold surface of the metal chills the gases to such a point that the "reaction velocity" of combustion is reduced to nil. This is made much worse by the intermittent character of the operation. We thus see that we are performing two distinct metallurgical operations in a cramped space under bad conditions, and are thus violating two canons of metallurgical science.

* * *

It is not hard to design a smoke consumer. A reverberatory copper furnace burns the "smokiest" of soft coal and produces little or no smoke. The reason is that the two operations of distillation and combustion are done separately under right conditions, and in a nearly continuous process. To make this, for purposes of steam raising, commercial as well as practical, is a harder matter. However well-placed thermal insulation and proper design this is possible, although great capital outlay militates against it. A better way would be larger central installations or by-product coke ovens selling gas at a cheap rate, and using the coke for producer-gas engine operations, as well as for metallurgical purposes and domestic uses. Or the "work" by producer gas could be utilized for a large gas engine, as the installation and electric power sold. Here we have two operations separate and both continuous, which is the proper set-out

circumstances for good metallurgical work. But, here again the bugbear of large capital outlay flares itself up in the face of those desiring an approximation to ideal and perfect utilization of the gifts of bountiful Dame Nature.

* * *

The work of the U. S. Geological Survey at the coal-testing plant at St. Louis is, however, welding the ideal with the real in the combustion of coal and making widespread the true theory of combustion of soft coal. This work is of so important a nature that Congress should continue it on a much amplified scale, and make it as fine an institution for research as the Department of Agriculture is for the "new farming" that is manifesting itself in increased yield per acre in the United States. The conservation of Nature's fixed resources, and the utilization of them for the benefit of posterity, is the highest ideal of statesmanship. And in this the world owes a debt of gratitude to the grand work of the German universities. The application of broad philosophical principles to every-day life is the keynote of the twentieth century.

The Eight-Hour Day in Smelting Plants.

Acute observers for ages have been predicting ever-increasing troubles in handling labor in this country, and, in fact, in all the world. The division of the unearned increment, or that portion of the wealth which comes to the complex organism society, is manifestly in many cases unfair, and the law of the survival of the fittest is rigorous and severe on the unfit. However, society blunders along, and a distinct amelioration of the laboring classes has resulted. The shortening of the hours of labor gives more time to relaxation and rest to the workman. In the metallurgical industry this same effect is marked. Wherever large machines are installed, the keenest nerves and "muscular co-ordination" are a prime requisite. Long hours will weary the workman so much that the necessary physical skill diminishes. And the end of the matter is that the efficiency of the workman is reduced and the efficiency of the machine is also lowered. Hence where great capital outlay for labor-saving machinery has been made, the pay and the hours of the workman are most satisfactory to him. All is summed up in the words "maximum efficiency." This is especially true of rolling steel blooms into finished form. And the expert steel roller receives pay commensurate with the high-priced professional ball player, whom, in fact, he resembles in quality of nerve and muscle.

* * *

In much work, however, the duties of the workman are for a great part of shift only those of attendance, and the workman can enjoy comparative ease with pipe in mouth, knowing by sound of machinery that "things are going all right," using little physical labor except where repairs are needed. This is seen to a more or less extent in the handling of mechanical reverberatory furnaces, concentration mills, and shaft furnaces in cases where charge descends easily and furnace works well due to excellent metallurgical conditions. Here a twelve-hour day is no great hardship to the furnaceman, and a short shift would be had not only because hourly [illegible] would be increased, but because the long shift gives the furnaceman a chance to gather a much better insight into the working of the furnace. Shaft furnaces, however, when hand charged, are run best on eight-hour shifts for hand charges and twelve-hour shifts for the furnaceman.

* * *

The question of operating plant at maximum efficiency demands men whose minds and bodies are run at maximum efficiency. The human body is something like a storage battery. Heavy discharge causes "sulfation," as excessive labor deteriorates the physical tone. Too rapid charging is inefficient, as is too much food for a man. A battery standing idle loses its ampere-hour capacity, as a man's muscles out of work get soft and flabby. In a general way the analogy is complete. Both are tender and complex electrochemical machines, and both must be cared for intelligently or they get "sick." In conclusion of this most important subject, we might state that, as a usual thing, twelve hours a day is too long a period for the human being to breathe and work in hot and bad air at trying and difficult operations. And the eight-hour day in the long run, under modern conditions of high-priced machinery and high-priced labor, will be found for the advantage of both employed and employer.

Education and Civilization.

At the present time when the universities begin their new terms, it is suggestive to think what the effect might be if all teaching and schooling should stop for fifty years. Every year about one-fiftieth of the human race dies, and about one-fortieth is born. Those dying take away with them all that they have learned in their lives, and those being born bring with them only the capacity to learn and a few animal instincts. The whole work then of educating the human race must be partly done over every fifty years. If, as suggested in the first sentence, we should miss one round, we would relapse many centuries, perhaps hundreds of them, toward the mere animalism from which our race originated. The preservation of our civilization itself requires, therefore, that education of the youth be maintained; it is the price we must pay for the continuance of our civilization; and it is evident that the preservation of our *progress* in civilization depends upon *progressive* improvement in the manner and substance of the instruction of the coming men.

* * *

Another consideration then presents itself, viz.: how much time and attention should be given to education or schooling, how far it should go, how high it should reach. This is the question which the wise and the foolish of many countries, races and centuries have been trying to solve, and no one has successfully solved it or ever will. It is one of those questions which cannot be solved arbitrarily by any one's logical conclusions; it is a question which solves itself. Over-education, if it is ever reached in any one case, will correct itself by resulting in inability, through lack of means, of over educating the succeeding generation. For if the immediate result of over-education is to make men vain, conceited and disinclined to hard work, mental or physical, it puts men out of touch and sympathy with their surroundings, out of harmony with the true principles of successful life, and its ultimate result will be inability to secure the means of over-educating the succeeding generation. Over-education would thus cause a re-

action, by which, for a time, the children would atone for the over-education of their ancestors by a period of salutary return to hard work and normal education. There can never be any fear of over-education producing any permanently baneful results.

* * *

There certainly is not at present in America an over-supply of university-trained men in business, professional or political life. On the contrary the demand at present is for more of them. The boy whose schooling stops at the grammar school, enters life with a seventeenth century education. The boy who graduates from the High School, gets a nineteenth century education. The university student gets a twentieth century education, and should be abreast of the best thought of his time. This constitutes the difference which education makes in the equipment of a man for his life work, and shows what advantage the university graduate possesses, in theory at least. To be placed abreast of the latest developments in science and industry, to start in the forefront of the world's advancing throngs, to stand on the shoulders of former workers, and to start with a possession of the best that they have to teach, is the privilege of the university student—which some foolishly ignore, many desire, and but a favored few enjoy.

* * *

Surely we do not want to imply that it is only the university graduate who is able to impress his individuality on contemporary life and to make history. The facts show too clearly that this is not so. In his admirable lecture before the Cornell students this Spring—at the time of the Ithaca meeting of the American Electrochemical Society—Mr. Edward G. Acheson, recognized by everybody as one of the foremost electrochemists of the world, tooks pains to emphasize that he is a self-made man and not a university graduate. Other captains of industry have proudly expressed themselves similarly. But a captain of industry is an exception. A genius will make his mark on the world under any conditions. Yet not everybody, fortunately, is a genius. A system of education has not to consider how to train the genius, but rather how to do the most good for progress on the whole, and considered in this light, there is no flaw in the above argument in favor of university education. Life, if it is worth while, is fighting. Goethe said, "Denn ich bin ein Mensch gewesen und das heisst eim Kämpfer sein." And in the fight which every man has to make in his life, his personal qualities are ultimately of decisive importance. But university education gives to a boy the best available weapons to make this fight.

* * *

There exists, unfortunately, in commercial work, say in a metallurgical plant, often a disagreeable feeling between the university graduates and those who are not, when they should stand together in harmony. Some university men are too liable to look down on those who were less fortunate. The practical self-made man, if he has honestly earned his position, has earned it through hard and conscientious labor. By tireless observation he has acquired a faculty of feeling or perceiving by intuition whether the process works correctly or not. Such knowledge as he has he has acquired by himself. It is stored in him, and it will die with him. He could not have learned it from books or from teachers. This fact makes him liable to underestimate or to wholly misunderstand the value of a scientific education. This is his mistake. But in many cases if we see what has been accomplished through consistent hard work with rule of thumb and try and miss methods we cannot but admire the man who did the work. Yet it remains a self-evident truth that we must get away from the rule of thumb, and must use instead exact scientific principles. The times have marvelously changed, and while we must admire the "practical man" of the past, the future belongs to the man whose work is based upon scientific principles. The practical furnace man in a metallurgical works of the past saw with his eyes whether the metal was at the right temperature. He did his work faithfully and successfully, but when he left the plant his knowledge went away with him. Now, in the period of pyrometers, a works is enabled to get a permanent record of the right succession of temperatures during a process. It is thus possible to repeat a successful process from the written record after success has once been established, and the plant is independent of the individual man. This illustrates clearly the industrial progress which has been made, and which is wholly the result of scientific progress, but the advance of pyrometry is only a single example of the tendency of the times—that is, advance of science for the sake of civilization and humanity.

The Effect of Colloids on Metal Deposition.

The beneficial effect which the addition of a very small quantity of a colloid to the electrolyte may have on the smoothness and density of the metal deposit is now well known. Mr. A. G. Betts was the first to employ this means on a large scale in his lead refining process. He found that the lead "trees" produced under ordinary conditions could be avoided, and a dense, coherent and solid lead deposit could be obtained by the addition of very small quantities of gelatine or glue to his electrolyte. Others have later applied the same means successfully for analogous purposes with other metals. But the rationale of the beneficial effect of the addition of a colloid has been a subject of much discussion and controversy. A new explanation has recently been offered, and has been made the basis of a patent by an engineer of the Siemens & Halske Co., Mr. J. A. Nussbaum, as mentioned in this issue under Analysts of Current Electrochemical Patents. His explanation is essentially that the metallic ions, when they give off their charges at the cathode, are not immediately deposited in metallic form, but exist for some time in a more or less unstable intermediate state. In this unstable form they are floating around in proximity to the cathode, and will pass over into the metallic state preferably at such places where metallic particles already exist (just as undercooled water will at once crystallize into ice around the smallest ice crystal dropped into the water). The effect of the addition of a colloid consists, according to Nussbaum, in the formation of a colloidal diaphragm over the cathode, which prevents the metal particles from floating freely around in their unstable intermediate form. This explanation seems, so far, purely hypothetical, but it has the advantage that it can be tested. If it is true, then only such colloids can be beneficial which travel with the current, since those colloids traveling in the other direction cannot form a protective diaphragm at the cathode.

Jubilee of the Coal-Tar Industry.

The jubilee of the coal-tar industries will be celebrated on Oct. 6 by a dinner, to be given at Delmonico's, New York, to Sir William Perkin, who invented the dye-stuff "mauve" fifty years ago.

A reception to Sir William will be held on Tuesday evening, Oct. 9, in the hall of the Chemists' Club, New York City.

The American committee which has arranged the Perkin celebration has extended a cordial invitation to the members of the American Electrochemical Society (which is to hold its meeting in New York City on Oct. 8 and 9) to attend both functions.

New York Meeting of the American Electrochemical Society.

The Autumn meeting of the American Electrochemical Society will be held in New York City on October 8 and 9. This is in consequence of a decision of the Board of Directors to try and hold one meeting each year, in Spring, in some place to be selected from year to year, and a second and secondary meeting in Autumn regularly in New York City. The meeting to be held will, therefore, be somewhat in the form of an experiment to see how the plan works out. The idea is that many members of the society have to make professional trips to New York City in the Autumn, and could arrange to combine them with attendance at the meeting.

In the present case, the dates have been so selected as to enable those who attend the Perkin Memorial Celebration of the Foundation of the Coal Tar Industry, which will take place October 6, to also attend the meeting of the society.

The members of the society are invited to join the committee of the Perkin Memorial Celebration in a banquet, to be given to Sir William Perkin at Delmonico's on the evening of Saturday, October 6, at 8 P. M. The cost is $5.00 per plate.

On Monday, October 8, at 9 A. M., a meeting of the Board of Directors will be held at Columbia University, while the first session for the reading and discussion of papers will begin at 10 o'clock. There will also be an experimental lecture at this session. On the same day, at 2.30 P. M., Dr. Charles Baskerville will deliver an experimental lecture at the Chemical Laboratory of the College of the City of New York, on "The Use of Ultra-violet Light in the Laboratory and in Practice."

Tickets will be provided by the local committee for visiting numerous points of interest on the same afternoon, including the largest power houses, the Electrical Testing Laboratories, the Pennsylvania Railroad tunnel under the Hudson River.

On Monday evening, at 8 o'clock, a beefsteak supper, the cost of which will be about $3.50 per plate, will be held at Reisenweber's, Fifty-eighth Street and Eighth Avenue, or in the Liederkranz Hall. Further announcements will be made in the session on Monday morning.

On Tuesday morning at 9.30 a second session will be held for the reading and discussion of papers, and an experimental lecture. In the afternoon the members of the society, and guests, will be taken to the works of the American Smelting & Refining Co., near Maurer Station, N. J. Another party will visit the laboratories of Mr. Thomas A. Edison, in Orange, and possibly the Weston Instrument Co's plant.

On Tuesday evening the members are invited to attend the reception given to Sir William Perkin at the Chemists' Club.

The Boards of Directors of the Chemists' Club and of the German Liederkranz Society have courteously placed their club houses at the disposal of the visiting members of the Society during the meeting.

Since it is intended to hold the Autumn meeting regularly in New York City, the local New York committee, which has been appointed, is expected to make all necessary arrangements, but not to raise money for entertainment, all cost of entertainment being borne by the members attending. Hotel headquarters will be at the Prince George Hotel, East Twenty-eighth Street, New York.

Dr. A. H. Elliot, 4 Irving Place, is chairman of the general local committee. Mr. H. B. Coho is chairman of the entertainment committee.

The following is a list of the papers which have so far been promised:

Charles Baskerville, The Use of Ultra-violet Light.

C. F. Burgess, title not yet received.

Henry S. Carhart, Formula for the e. m. f. of a Helmholtz Concentration Cell.

G. H. Cole and H. T. Barnes, An Aluminium and Magnesium Cell.

Carl Hering, Visible Migration of Particles Between Electrodes.

H. E. Patten, Some Factors Affecting the Distribution Law.

Henry Noel Potter, title not yet received.

Francis R. Pyne, Melting Points of Some Cryolite-Alumina Mixtures.

E. F. Roeber, Use of Pyrometers, with Exhibitions.

P. B. Sadtler and W. H. Walker, Double Decomposition of Zinc Sulphate and Sodium Chloride.

S. S. Sadtler, Some Small Laboratory Appliances for Electric Fusion and Other Work.

J. W. Turrentine, Copper Cathodes in Nitric Acid.

F. F. Schuetz, A Thermo-Electric Pyrometer for General Industrial Applications.

F. B. Crocker, The Decker Primary Battery.

The Iron and Steel Market.

The rush to buy pig iron for all deliveries which characterized August subsided before the middle of September. The scarcity of prompt iron has continued, and still higher prices have been paid for small lots. Barring very unfavorable weather conditions and interference with the movement of raw materials, the market for prompt iron will have to yield to a largely increased supply, and prices for prompt and future reach a more harmonious relation.

The market in finished steel products has been uniformly strong. Buying and specifying has been hardly as heavy in September as it was in August, but total bookings for the month have been beyond output, and there is still less business left to be done in the late fall and winter.

It is difficult to particularize, where demand is so uniformly good from all classes of consumers, but it may be remarked that the railroads have shown increased activity in the buying of steel cars and the placing of blanket contracts for bridges for 1907. Their interest in rails has correspondingly diminished, sales for next year being relatively light, while the Western roads are exerting great pressure on the rail mills to have deliveries on 1906 contracts rushed to permit of closing up of outside work before the worst weather sets in.

It is as thoroughly established as can be with anything so fickle as the American iron trade, that the present very prosperous condition will be maintained at least until some time in the second quarter of 1907. Beyond that time the course of the market will be determined by the relation worked out between physical results this winter and the personal equation.

Pig iron production, and consequently the production of crude steel and finished products, will be much larger than in the quarter just closed; how much larger cannot be forecasted, nor can it be told to what extent this increase has been anticipated by the buyers who have been so actively in the market. On the relation between this increase, whatever it may be, and the amount of increase which buyers anticipated, will probably

depend the course of the market after the first quarter of next year.

Pig Iron.

Early in September, Bessemer and basic pig iron advanced to $18.75 and $18.50 f. o. b. valley furnace, respectively, at which prices they have since been firmly held, with light sales. Occasional prompt lots of Bessemer have been sold by middlemen at $19.00, valley, equal to $19.85 delivered Pittsburg or Cleveland. The large steel interests have done practically nothing, having found they would have to adjust their steel output to their own supply of steel-making pig. The floating supply of both grades is fairly well taken up to the middle of next year.

Buying of foundry iron has also diminished, on account of the higher prices and the fact that the majority of large consumers are covered fairly well through the first quarter or first half of next year. Buying of prompt foundry iron has been active. In August this buying was by consumers who were not covered; in September many of the buyers had contracts, but were not getting the desired deliveries. Prices have advanced sharply, but the market has been too irregular to quote definitely. For anything like prompt delivery the valley furnaces have been obtaining $20 at furnace, or $20.85 delivered Pittsburg. Eastern furnaces have obtained $21.00 to $22.00 delivered to the Philadelphia district.

The Southern market is a shade easier, in the sense that the $15.00, Birmingham, price can be done in some instances on deliveries beginning late this year. For prompt shipment $15.50 to $16.00 is still obtained.

Production is increasing rapidly in all districts. The United States Steel Corporation since the middle of September has had about 95 per cent of its furnace capacity in blast, or above the normal, since the necessities of relining take an average of about 10 per cent. Many outside furnaces are coming in after relining, the humidity has decreased and unfavorable winter weather is not due yet. In Alabama there is a car shortage, which has interfered with even the short hauls for coke, limestone and ore, and is claimed to have somewhat interfered with actual pig iron production, although shipments are made fairly well.

Billets and Sheet Bars.

The supply of soft steel has materially improved, but as producers were so far behind in deliveries on their regular contracts but little additional material has been thrown on the open market. Soft steel billets can still be quoted at $28.00 to $29.00, Pittsburg, for Bessemer, and $29.00 to $30.00 for open-hearth. For delivery next year considerably lower prices have been quoted in some instances, and it is probable that if any large tonnage is sold it will be at a couple of dollars or so below the prices just named.

Sheet bars are nominally $30.00 to $31.00, Pittsburg, for odd lots, but the bulk of the tonnage moving is at $29.00 or less. Two sales of round lots for first quarter delivery have been made at less than a basis of $29.00, Pittsburg. On the quarterly contracts of the leading interest a price of $28.00 was made for the second quarter, and of $29.00 for the third quarter. A considerable portion of the steel specified under these contracts for earlier delivery will go out in the current quarter.

Attention should be called to an anomaly which has existed for some time and has become more pronounced in the past month, and that is the relative scarcity of steel which answers to any description other than that of ordinary soft steel billets. Billets under the standard 4 x 4 size have long been very scarce, and in many cases have been bringing the merchant bar price of 1.50c., or $33.00 per gross ton, there being no extra on such sizes. It is a fact that a large tonnage of such billets has actually been rolled on merchant mills. There have been times when it was difficult for a manufacturer to obtain any extra at all for small billets over 4 x 4. The large mills are being pushed to the limit, and do not care to reduce out put, when the ingots are available, to make special sizes except for their own requirements.

Similar large premiums rule in needs of any special analysis whatever. Large orders for axle billets and carbon forgings have almost gone begging, when the latter was selling at $2.00 to $3.00 a ton more above the regular run of basic soft steel. In times of comparative dullness the steel mills have been only too glad to make material of special quality at a slight advance. It is not that demand for special steels has increased, but that there is such pressure that any departure means a loss of output. Furthermore, the increases in steel making capacity in the past few years have been chiefly in the direction of making ordinary steel. The very large producers have enlarged their Bessemer and open-hearth departments in order to find enough steel for their own finishing mills, not for the purpose of putting crude steel on the market. The smaller interests which have built steel mills have done so to supply their own requirements, and while they usually have limited tonnage of crude steel for the market, they can readily sell the surplus as soft steel, and do not need to bother with special analyses.

Finished Material.

The great bulk of the finished steel production of the country to the end of the first quarter is disposed of, either by outright sales or by contracts upon which specifications are being placed freely.

Rails have been quiet the past month, but it is between seasons. The early buying filled up the Southern and Western mills for nearly all of next year, while the Eastern mills are sold up for from a fourth to a half of the prospective production. Light rails have been very active, the mills making light rails from new steel being sold up for from three to four months ahead, which is very far ahead for this trade. The rerolling mills have all the prompt business they can handle.

Large steel car orders are being placed, and the car plants will soon be filled until nearly the middle of next year. The lake shipyards are almost filled for next year, and specifications for plates and shapes are correspondingly good. Fabricated structures are being placed freely.

Three large steel bar producers—Cambria, Republic and Crucible—have now advanced their price on steel bars to a minimum of 1.60c., but the Steel Corporation and some other producers are opposed to any advance, and continue to sell at 1.50c., which will remain the market, although deliveries are getting further behind. Tin plate has been very active, contracts being placed freely for first quarter, and in some cases also for second quarter. Sheets are firm but hardly as active as in August, and the prospect of an advance has almost disappeared.

The following prices are f. o. b. Pittsburg, plus regular rail freight to destination, and are regular mill prices on carload and larger lots for forward delivery; in the case of sheets and tin plates a concession of 5 cents a hundred pounds is usually given to good buyers; in wire products a similar concession is given to the largest jobbers.

Beams and channels, 15 inches and under, $1.56 per 100 pounds; over, $1.75; beams and channels over 15 inches, $1.80.

Plates, tank quality, ¼ inch and heavier, 100 inches wide or less, $1.60.

Sheets, 28 gauge, box annealed, one pass, cold rolled, $2.50; galvanized, 28 gauge, $3.55; blue annealed, No. 10 gauge, $1.75.

Plain wire, $1.70 per 100 pounds; wire nails, $1.85 per keg, both base.

Tin plates, 100 pound cokes, $3.75 per box.

Obituary. We regret to report the death at Döbling, on Sept. 5, by his own hand, of Prof. Ludwig Boltzmann, of the University of Vienna. He was one of the most brilliant contemporaneous theoretical physicists, and especially distinguished himself in the development of thermodynamics, the mechanical theory of gases and thermal energies. The

World's Pig Iron Production.

Provisional or final statistics of pig iron production in the first half of 1906 have been returned from the United States, Great Britain, Germany, Belgium and Canada, from which fairly trustworthy deductions may be made for the entire calendar year, while other countries may be assumed to have changed their production but little from 1905, and the total prospective pig iron production of the world in 1906 may be approximated with a probable error of considerably less than a million tons, as in the table below, which shows the ascertained outputs in 1900 and 1905 as far as possible. The United States, Great Britain and Canada return their statistics in gross tons of 2,240 pounds; other countries use the metric tons. Not to do violence to the statistics of two of the three greatest iron producing nations, the metric tons are reduced to gross tons to permit of the figures being added.

World's Pig Iron Production.

Tons, 2,240 Pounds.

	1900.	1905.	1906. (Prospective)
United States.....	13,789,242	22,992,380	25,300,000
Germany	8,381,373	10,813,979	12,055,000
Great Britain.....	8,959,691	9,592,737	9,890,000
France	2,669,966	3,028,376	3,028,376
Russia	2,889,789	2,931,257[1]	2,931,257
Austria-Hungary .	1,472,695	1,300,000[2]	1,400,000
Belgium	1,001,872	1,329,109	1,385,000
Sweden	518,263	518,967	518,967
Spain	280,315	377,046	377,046
Canada	86,090	468,003	570,000
Italy	15,019	110,819	110,819
Japan	.....	33,335[4]	33,335
Other countries....	100,000	200,000	200,200
Totals	40,173,915	53,695,189	57,800,000

[1] 1904. [2] Estimated. [3] 1901. [4] 1903.

British Iron and Steel Production.

The British Iron Trade Association has returned the production of pig iron and of open-hearth steel ingots and castings in the first half of this year, and we make the following comparisons, the figures all referring to gross tons of 2,240 pounds:

Pig Iron Production.

First half 1905.....	4,621,600
Second half 1905	4,971,137
First half 1906.....	4,905,424

As the last six months of the year contain three days more than the first six months, by actual half years the production in the first half of this year would exceed that of the second half of last year, and thereby become the greatest production in any half year in the country's history. The fact is worth noting, since the British iron trade does not break records with American facility. As far back as 1868 as much iron was made in one year as is now made in a half year.

Open-Hearth Steel Production.

First half 1905	1,980,095
Second half 1905	1,890,053
First half 1906	2,106,853

Of the production in the first half of this year, 1,038,667 tons were made by the acid process and 558,186 tons by the basic. The acid process gained only 10,990 tons over the first half of 1905, while the basic gained 205,789 tons.

Hydrochloric Acid and Caustic Potash.

The Roberts Chemical Co., of Niagara Falls, electrolyze potassium chloride. At the cathode caustic potash is produced and hydrogen is set free, while at the anode chlorine is set free. The quantities of hydrogen and chlorine are in equivalent proportions, so that they may be combined to form hydrochloric acid. This is done by the Roberts Chemical Co., the two products of which are, therefore, hydrochloric acid and caustic potash. In the September issue of *Things Chemical*, of the Charles E. Sholes Co., who are the sales agents of the Roberts Chemical Co., it is pointed out that on account of its method of production, this hydrochloric acid cannot contain the ordinary impurities of muriatic acid unless they were expressly and intentionally added. Its only impurity is a small amount of free chlorine, which gives it a pale yellow color. This acid is of special interest to manufacturers of fine chemical products, gas mantles, cereals and foodstuffs (where there must be no arsenic), high grade and fancy leathers, and in general, for all purposes where quality is of more importance than a slightly increased first cost. The electrolytic caustic potash is made in three forms: fused (solid) in drums of 100, 200 and 800 pounds; broken in drums of 100, 250, 500 and 1,000 pounds, and caustic potash solution in drums of 325, 650 and 1,350 pounds. The solid and broken grades analyze not less than 85 to 87 per cent KOH and 10 to 15 per cent carbonate and chloride of potassium. The solution contains 45 to 47 per cent KOH. Caustic potash is now very largely used in electroplating shops for removing grease from metallic surfaces before plating. In the laundry, a dilute solution of caustic potash is an excellent "builder," for the soap will remove all grease and produces excellent suds, without injury to the fabric. Soap makers use caustic potash especially for all the finest soaps, whereas caustic soda is used for the common grades.

British Steel Combine.

The British steel tube manufacturers have finally effected a combination, after years of effort, the importance of which is evident by the fact that the annual gross output of the firms concerned amounts to 300,000 tons, valued at over $29,000,000. There are some sixty firms in the tube trade in the United Kingdom, and with the exception of one Glasgow house they have, according to the London *Economist*, entered into a compact to cease from cutting prices of a prepared schedule. Discounts on all kinds of tubes, except those for boilers have been reduced by 2.5 per cent, and prices of all sections of tubes for export have been advanced by 5 per cent on the net, these prices to hold firm until officially altered, which may be before long and on the upward grade. Boiler tubes have been left off the schedule, because the independent Glasgow firm makes a specialty of these. Germany is not now a competitor in steel tubes, and it is anticipated by expert authorities that with this agreement an international arrangement for the regulation of the world's tube trade will sooner or later be evolved. Measures toward that object are to be at once undertaken. It is calculated that out of the total British product 70 per cent goes abroad. The British Iron Trade Association has ascertained that the total output of open-hearth steel in the United Kingdom in the first half of 1906 was 2,183,850 tons, which compares favorably with an output of 1,980,095 tons in the first half of 1905, and 1,970,129 tons in the first half of 1904. The increase in 1906 was shared by every manufacturing district.

Obituary.—Mr. Louis Schutte, president of the Schutte & Koerting Co., Philadelphia, Pa., died on Sept. 29, at the age of 65 years. Mr. Schutte was born in Germany and graduated from the School of Technology in Hanover in 1862. In his professional work he later became associated with Mr. Ernst Koerting. He came in 1870 to this country, and founded, together with Mr. Koerting, the Philadelphia firm which bears their names.

Metallurgical Calculations.

By J. W. Richards, Ph. D.
Professor of Metallurgy in Lehigh University.

PRODUCTION, HEATING AND DRYING OF AIR BLAST.

This is a subject intimately related to the running of iron blast furnaces, and incidentally related more or less to all classes of metallurgical operations. The principles involved are those of physics—mechanical and thermal—and when once thoroughly understood can be used for the most various classes of metallurgical problems.

Production of Blast.

There are two different principles upon which air is compressed, represented by the fan and the piston machines. The first is constant in its operation, the second intermittent, the first draws in and expels a continuous current of air, the second draws in a given quantity of air, compresses it and expels it. Measuring the work done by the difference in static conditions of the compressed and of the uncompressed air, the work actually done is a fixed and calculable quantity, independent of what type of machine performs it. During the compression, heat is generated, and the mechanical work done includes the mechanical equivalent of the heat thus generated. This is allowed for in the well-known formula for adiabatic compression:

$$\text{Work} = \frac{y}{y-1} V_o p_o \left[\left(\frac{p_1}{p_o} \right)^{\frac{y-1}{y}} - 1 \right]$$

in which

y = the ratio of the specific heat of air at constant pressure to that at constant volume = 1.408.
V_o = the volume of the uncompressed air.
p_o = the tension of the uncompressed air.
p_1 = the tension of the compressed air.

If we use the value of $y = 1.408$, the coefficient $\frac{y}{y-1}$ becomes 3.45, and $\frac{y-1}{y} = 0.29$. If we then use the other dimensions in feet and pounds, the resulting work done will be in foot-pounds; if we use meters and kilograms, in kilogram-meters. The element of time does not enter into this equation, the work actually done is the same for compressing a given amount of air, and is independent of the time. If I lift a kilogram 1 meter high, the same amount of work is done whether I lift it in 1 minute or in 1 second; the *rate of doing work* would be, however, sixty times as great in the last instance as in the first, but the actual amount of work accomplished is the same in each case. If, therefore, we use in the formula the volume of air entering the compressor per minute, the result will give the work done per minute; if we use the volume per second we get the work done per second. If we wish to transpose the work done into horse-power, we bear in mind that a horse-power is understood in English units as 33,000 foot-pounds of work done per minute, and in the metric system as 75 kilogrammeters of work done per second.

In applying the formula we may further notice that $\frac{p_1}{p_o}$ expresses the ratio of compression; that is, by how many times the tension of the original air is increased. If air at one atmosphere tension (such as the air we usually have to breathe) is compressed to two atmospheres tension, the ratio of compression is 2; if air entered a machine at two atmospheres tension and was therein compressed to four atmospheres, the ratio of compression would be likewise 2, and the work done (for a given quantity of air) the same as before.

The effective pressure, as shown on a gauge, is the difference between these two tensions; it is not p. It is highly important that this point be held clearly in mind. The pressure of the uncompressed air is its barometric pressure as measured against a vacuum; the tension of the compressed air is likewise its pressure as measured against a vacuum; the effective pressure of the compressed air is the difference between these two tensions; the tension of the compressed air is, therefore (by transposition), equal to the barometric pressure of the uncompressed air plus the effective pressure of the compressed air, *i. e.*, plus the gauge pressure. Never use the effective pressure of the compressed air as p_1, but always add to it the barometric pressure of the uncompressed air p_o for the correct value of p_1.

Regarding the volume of uncompressed air, V_o is its volume measured at the pressure P_o, *i. e.*, the actual volume of uncompressed air at its actual tension. If more convenient, however, we may use the volume of this air measured at standard barometric pressure, if we multiply it by p_o, the standard pressure. Thus, if we know the volume of uncompressed air measured at one atmosphere standard tension, we may use for the p_o immediately following it the standard pressure 10.334 (kilos. per square meter), or 2,120 (pounds per square foot). By the law of the reciprocal nature of volume and pressure, the product thus obtained is the same as would be found by multiplying the volume of the same air at any other pressure by that pressure ($V_o p_o = V_1 p_1 = V_2 p_2$, etc.). Whatever pressure we use in getting the product $V_o p_o$, however, we must use only the actual tension of the uncompressed air as the denominator in getting the ratio of compression ($p_1 \div p_o$).

Problem 56.

A blast furnace requires 2,615 cubic meters of air, measured at — 5° C., for every metric ton of pig iron made. Assuming outside barometric pressure 735 millimeters of mercury column, the efficient pressure of the compressed blast to be 1.0 kilogram per square centimeter (as measured in a blast regulating reservoir), a mechanical efficiency of the blowing engine of 90 per cent. of delivery of blast 82.3 per cent, and output 447 tons per day:

Required:

(1) The ratio of compression of the blast.

(2) The piston displacement per ton of pig iron made.

(3) The work of the blowing cylinders, per ton of iron made.

(4) The gross or indicated horse-power of the steam cylinders.

Solution:

(1) The effective pressure being 1.0 kilogram per square centimeter, represents $\frac{1.0000}{1.0334} \times 760 = 735$ millimeters of mercury. But the uncompressed air is at 735 mm barometric tension, therefore, the total tension of the compressed air, p_1, is 1470 mm. of mercury, and the

$$\text{Ratio of compression} \quad \frac{1470}{735} = 2.0 \qquad (1)$$

(2) Piston displacement per metric ton of pig iron:

$$\frac{2615}{0.823} = 3,165 \text{ cubic meters} \qquad (2)$$

$$= 111,560 \text{ cubic feet}$$

(3) The volume displaced must be the basis of calculating the power required, since subsequent losses (as are those 82.3 — 17.7 per cent) do not affect the work done in the blowing cylinder. This volume is 3,165 cubic meters, representing that volume of uncompressed air at — 5° C. and 735 mm. pressure. The initial temperature of the air does not affect the work to be done, and we can either use its volume at 735 mm. tension or correct this to 760 mm. tension, just as

we chose. If we consider the volume at 735 mm. tension, then the product of volume and pressure is

$$Vp = 3.105 \times \left(10.334 \times \frac{735}{760} \right) = 31,755.000$$

If we first change the volume to standard pressure, we have

$$V_o p_o = \left(3.105 \times \frac{735}{760} \right) \times 10.334 = 31,755.000$$

The work done in compression, in the blowing cylinder, is, therefore,

$$\begin{aligned} W &= 3.45 \times 3.073 \times 10.334 \times \left[2^{1.29} - 1 \right] \\ &= 109,554.750 \times \left[1.2225 - 1 \right] \\ &= \mathbf{24,326,800} \text{ kilogrammeters.} \end{aligned} \qquad (3)$$

(4) The steam in the steam cylinders does more mechanical work than would be represented by the compression of air in the blowing cylinders. In this case, we assume 90 per cent mechanical efficiency, or 10 per cent mechanical loss. The gross work of the steam is, therefore,

$$24,326,800 \div 0.90 = 27,029,800 \text{ kilogrammeters,}$$

or, in horse-power,

$$\frac{27,029,900 \times 447}{1440 \times 4500} = \mathbf{1,865} \text{ horse-power.}$$

This amounts to 4.17 hp. per metric ton of pig iron made per day.

Measurement of Pressure.

In making calculations of the work done in furnishing blast, as in the preceding problem, it is important to note how the pressure of the compressed air is measured. The real pressure is measured correctly by a pressure gauge only where the air is comparatively at rest, as on a pressure-equalizing reservoir. If the pipe communicating with a gauge is connected with a blast main, in which the velocity of the air is considerable, the pressure recorded will vary greatly with the direction of the end of this tube relative to the direction of the air current. The total pressure of the air in motion is the pressure which it exerts against the sides of the main, plus the pressure which has been used in giving it velocity. If the pressure gauge tube is cut off at right angles to the flow of air in the main, the pressure recorded is even less than the actual static pressure of the air against the sides of the main (because of suction effect), and does not include at all the pressure represented by the velocity. The only way to measure correctly in a main the total pressure which has been impressed upon the blast is to bend the end of the pressure tube until it is parallel with the axis of the main and faces the current of air. The gauge then records the static pressure, plus the velocity head, and gives the proper pressure to use in calculating the work done by the blowing engine. However, it is still preferable to put the gauge on a blast reservoir, if there is one, where the air is nearly at rest and velocity head approximates zero—for the reason that the velocity of air passing through a tube is greatest in the center and least against the walls, and it is difficult to place the pressure tube so as to measure the mean velocity. An approximation to the mean value is found by placing the end of the pressure tube not in the center but at one-third of the radius of the main from the center.

Indicator Cards.

If the blowing engine cylinder can be tested by pressure indicator cards, then a different method of obtaining the work of producing the blast is furnished. The integration of the diagram gives the mean pressure on the piston during the stroke. Expressing this in kilograms per square meter or pounds per square foot, and multiplying by the area of the piston and the length of the cylinder, we get the work done per stroke; again multiplying by the number of strokes per minute we have the work done per minute, from which is readily obtained the horse-power. These operations are usually combined in the formula:

$$\text{Work} = P \times L \times A \times N$$

If we observe that $L \times A \times N$ is the piston displacement per minute, the formula becomes:

$$\text{Work} = P \times \text{Piston displacement per minute,}$$

in which, it must not be forgotten, P represents mean pressure on the piston during the stroke, and is not to be confounded with gauge pressure of the compressed air. Such a formula is, therefore, totally inapplicable to finding the work done by a fan or rotary blower, in which only the final pressure of the compressed air is known. The mistake of so applying it is often made. When only the final pressure of the compressed air is known, the formula for adiabatic compression is the only correct one to use.

It may not be useless to call attention to the fact that when using the formula for adiabatic compression, the raising of the ratio of compression to the 0.29 power has to be done by logarithms:

$$\log \left(\frac{p_1}{p_o} \right)^{0.29} = \log. \left(\frac{p_1}{p_o} \right) \times 0.29$$

If a table of logarithms is not at hand, a satisfactory approximation may be made by taking the cube root of the ratio, since

$$\sqrt[3]{\left(\frac{p_1}{p_o} \right)} = \left(\frac{p_1}{p_o} \right)^{0.33}$$

Heating of the Blast.

Air blast is commonly heated either continuously, by direct transmission of heat through metal or earthenware pipes, or discontinuously by the heating up of fire-brick surfaces, which are subsequently cooled by the blast. It is not within the province of these calculations to enter into a description of the various types of such hot-blast stoves, but to indicate the principles upon which the metallurgist can base calculations as to the efficiency of such stoves, and thus be prepared to find out which do the best work, and wherein lie the principal advantages or disadvantages of each type.

The efficiency of a hot-blast stove is measured by the ratio of the heat imparted to the blast to that contained in the air and fuel used and generated by combustion. It is a furnace whose useful effect is the heat imparted to the air, and all other items of heat distribution are more or less necessary losses. The problem is simplest when the stove is a recuperator (continuous type), using solid fuel. Then the item of heat generation is perfectly definite, since the amount of fuel used in a given time can be definitely determined. When the hot-blast stove receives gaseous fuel, however, from a blast furnace, the amount of gas received by the stoves is usually a very uncertain quantity, since only part of all the gas produced by the furnace is used by the stoves, and the question of what fraction is thus used is difficult to determine. In such cases, the usual method of comparing the sizes of the pipes leading gas to the stoves and those carrying gas to boilers, etc., affords but a very uncertain determination, since the draft and consequent velocity of the gas may be very different in the two sets of pipes. In such cases, knowing the total volume of gas produced by the furnace, not only the relative sizes of the hot-blast stove pipes should be noted, but also the relative velocities of the gas currents in the two sets of pipes. Another method is to determine the quantity of chimney gas passing away from the stoves into the chimney flues, by measuring the size of the chimney flue, temperature of the gases and average velocity; given in addition the chemical analyses of the chimney gas and of the furnace gas, the quantity of the latter being used can be calculated, using the carbon in the two gases as the basis of calculation.

Problem 57.

Statement.—Air for drying peat in a kiln is heated from 0° C. to 150° C. by an iron pipe stove, the latter consuming dried peat, whose ultimate analysis is:

Carbon	49.70	per cent
Hydrogen	5.33	"
Oxygen	30.76	"
Nitrogen	1.01	"
Ash	13.23	"

The calorific power of this peat is 4249, and by the combustion of 92.5 kilograms, 5122 cubic meters of air is heated to the required temperature. The products of combustion pass away from the stove at 200° C., and contain (analyzed dry) 14.8 per cent of CO^2, and no CO or CH^4 or other gas containing carbon.

Required:

(1) The heat generated in the stove.

(2) The heat in the hot air, and the net efficiency of the stove.

(3) The volume and composition of the products of combustion in the stove.

(4) The heat carried out in the chimney of the stove.

(5) The heat lost by radiation and combustion.

(6) The excess of air used in burning the peat.

(1) Heat generated in the stove:

$4249 \times 92.5 = 393{,}033$ Calories (1)

(2) Heat imparted to the air:

$[0.303 + 0.000027\ (150)] \times 150 \times 5122 = 235{,}907$ "

Net efficiency = **60.0** per cent (2)

(3) Volume of products of combustion:

Weight of carbon in fuel burnt —

$92.5 \times 0.4970 = 45.97$ kilos.

Volume of CO^2 thus produced —

$45.97 \div 0.54 = 85.13$ cub. meters

Volume of (dry) gas produced —

$85.13 \div 0.148 = 575.20$ " "

Volume of N^2 and O^2 in products —

$575.2 - 85.13 = 490.07$ " "

Volume of water vapor in products —

$92.5 \times 0.0533 \times 9 \div 0.81 = 54.78$ " "

Percentage composition of products of combustion:

	Moist.	*Dried.*
CO^2	13.5	14.8
N^2 and O^2	77.8	85.2
H^2O	8.7	...

To separate the nitrogen and oxygen would necessitate considerable calculation, and is not required just here, because the two gases have the same heat capacity per cubic meter, and therefore can thermally be reckoned together:

(4) Heat in the chimney gases at 200°:

CO^2	$85.13\ m^3 \times 0.4140$	$= 35.24$
N^2 and O^2	$490.07\ m^3 \times 0.3084$	$= 151.14$
H^2O	$54.78\ m^3 \times 0.3700$	$= 20.27$
		$206.65 \times 200 = 41{,}330$ Cal.

Proportion thus lost = **10.5** P. C. (4)

(5) Loss by radiation and conduction:

$100 - (60.0 + 10.5) =$ **29.5** P. C. (5)

(6) The separation of N^2 and O^2 in the products is not easy. It is best based on the consideration that part of these consists of the nitrogen of the air which was necessary for combustion (plus the nitrogen of the fuel) and of the excess air. The first can be calculated, and thus the latter obtained by difference, and then the percentage of air in excess determined.

Oxygen necessary for combustion:

C to $CO^2 = 45.97 \times \frac{32}{12} = 122.59$ kilos.

H to $H^2O = 4.93 \times$ 39.44 "

162.03 "

Oxygen present in peat 92.5×0.3076 28.45 "

Oxygen to be supplied = 133.58

Nitrogen accompanying = 445.27

Air necessary = 578.85 "

= 447.7 m.

Nitrogen from coal, 92.5×0.0101 = 0.93 kilos.

Total nitrogen from coal and necessary air = 446.2 "

Volume = $446.2 \div 1.26$ = 354.1 m^3

Total N^2 and O^2 in products = 490.1 "

Therefore, excess air = 136.0 "

Percentage of excess air $\frac{136.0}{447.7} = 0.303$ = 30.3 P. C. (6)

Problem 58.

A blast furnace has three hot-blast stoves, two of which are always on gas and one on blast. Per long ton of pig iron produced there issues from the furnace (reduced to standard temperature and pressure):

Nitrogen	81,763	cubic feet
Carbon monoxide	25,383	" "
Carbon di-oxide	20,409	" "
Water vapor	8,230	" "

For the same quantity of pig iron made, 92,330 cubic feet (standard conditions) of air is heated in the stoves from —5° C. to 465° C. The hot gases reach the stoves at 175° C., are there burned by 10 per cent excess of air at 0°, and the chimney gases resulting (assume perfect combustion) pass out of the stoves at 120° C.

Required:

(1) The net efficiency of the stoves, assuming they receive 25, 30, 35, 40, 45 or 50 per cent of the gas produced by the furnace.

(2) Assuming that each stove radiates and loses to the ground one-third as much heat as the blast furnace itself (the heat balance sheet of the furnace showed 846,518 pound Calories thus lost per long ton of pig iron produced), what proportion of the whole gas goes to the hot-blast stoves, and what is the net efficiency of the stoves?

Solution:

(1) We will first calculate the efficiency of the stoves, assuming that they received *all* the gas produced, from which datum can then be obtained the efficiency, supposing any assumed fraction of the gas goes to the stoves.

Heat in the blast (—5° to 465°):

$92{,}330 \times [0.303 + 0.000027\ (405 - (-5))] \times [405 - (-5)] =$

$92{,}330 \times 0.31542 \times 470 = 13{,}687{,}683$ ounce Cal.

= 855,480 pound Cal.

Sensible heat in the hot gases (0° to 175°):

N^2 and CO	107.146×0.3077	$= 32.960$
CO^2	$20{,}409 \times 0.4085$	$= 8.337$
H^2O	$8{,}230 \times 0.3763$	$= 3.097$

Heat capacity per 1° = 44.403 ounce Cal.

= 2.775 pound Cal.

2.775×175 = 485.625 " "

Heat generated by combustion:

CO to CO^2 $25{,}383 \times 3062$ = 77,722,746 ounce Cal.

4,857.672 pound Cal.

Total heat available = 5,343,297 " "

If the gas were all used in the stoves the efficiency of the latter would be only

$\frac{855{,}480}{5{,}343{,}297} = 0.160$ = 16.0 per cent.

With smaller proportions of the whole gas used the efficiency of the stoves calculates out as follows:

Using 50 per cent of the gas.....	Efficiency	32.0	per cent		
" 45 " "	"	36.0	"		
" 40 " "	"	40.0	"		
" 35 " "	"	45.7	"		
" 30 " "	"	53.3	"		
" 25 " "	"	64.0	"		
" 16 " "	"	100.0	"		(1)

All that the above analysis tells us is, that certainly over 16 per cent of the gas produced by the furnace must be used by the stoves; but if we can deduce any probable value for the percentage of the gas actually used, such as by measuring the several gas mains and the velocities of the gas in each, we can then reckon out the value of the efficiency of the stoves, with about the same degree of probability. Blast furnaces use from 33 to 60 per cent of the gas they produce in the stoves. If we assume 50 per cent, in this case, the efficiency of the stoves would be 32 per cent.

(2) There is another way of solving the problem, which is to either measure, calculate or assume the heat lost by radiation and conduction from the stoves; adding to this the heat going out in the products of combustion, and the net heat in hot blast, the sum is the total heat which has been brought into and generated within the stoves. But, all the gas would bring in and generate in the stoves 5,343,297 Calories; we can, therefore, find easily what proportion of all the gas is being used in the stoves. In requirement (2) we are told to assume that the three stoves lose by radiation and conduction 846,518 pound Calories per long ton of pig iron made, an amount equal to that similarly lost by the furnace itself. The heat in the chimney gases, at 120°, can be thus calculated:

Oxygen required $\left(\frac{1}{2}\text{ CO}\right)$	=	12,092	cubic feet.
Air required	=	61,017	" "
Excess air used	=	6,102	" "
Nitrogen of required air	=	48,325	" "
Nitrogen already in gas	=	81,763	" "
Nitrogen in these two items	=	130,088	" "

Chimney products:

CO^2	=	45,792	" "
H^2O	=	8,230	" "
N^2	=	130,088	" "
Excess air	=	6,102	" "

Heat in chimney gases:

CO^2	45,792 × 0.3964	=	18,152	ounce Cal.
H^2O	8,230 × 0.3580	=	2,946	" "
N^2 and air	136,190 × 0.3064	=	41,729	" "
Heat capacity per 1°		=	62,827	" "
		=	3,927	pound Cal.
Heat capacity per 120°		=	471,122	" "

If all the furnace gas were burnt in the stoves, 471,122 pound Calories would go up the stove chimneys. But, since only a part of the gas is so used, only a fraction of this amount of heat is lost to the chimneys. If we call X the proportion of the gases used, then 471,122 X will represent the chimney loss, and the total heat requirements of the stoves will be:

Heat in air blast	=	855,480	lb. Cal.
Heat in chimney gases	=	471,122 X	"
Radiation and conduction	=	846,518	"
Total	=	1,701,998 + 471,122 X	lb. Cal.

The proportion of the total gases needed to supply this, X, is

$$\frac{1{,}701{,}998 + 471{,}122\ X}{5{,}343{,}297} = X$$

whence

$$X = 0.349 = \mathbf{34.9}\text{ per cent} \qquad (2)$$

and the net efficiency of the stoves is

$$\frac{855{,}480}{5{,}343{,}297 \times 0.349} = \frac{855{,}480}{1{,}864{,}812} = \mathbf{45.9}\text{ per cent} \qquad (2)$$

Arithmetically, the finding of X can be simplified, perhaps, by considering that the chimney loss represents in any case 471,122 ÷ 5,343,297 = 0.088 = 8.8 per cent of the total heat received by the stoves, leaving 91.2 per cent to be applied to heating the blast and for radiation and conduction loss. The last two items, however, must amount to 1,701,998 Calories, and, therefore, the total heat requirement of the stoves is

$$\frac{1{,}701{,}998}{0.912} = 1{,}866{,}200\text{ pound Calories,}$$

requiring

$$\frac{1{,}866{,}200}{5{,}343{,}297} = 0.349 = \mathbf{34.9}\text{ per cent} \qquad (2)$$

of all the gas produced by the furnace.

In the above solution the only uncertain factor in the calculation is the radiation and conduction loss from the stoves, and this uncertainty does not largely affect the reliability of the result obtained, allowing that we have assumed an approximately correct value for this loss. All uncertainty could be dispelled, however, were the radiation and conduction losses measured directly. This could be accomplished by finding experimentally the temperature of the outside shells of the stoves, and calculating the external losses of heat by the laws of radiation and conduction; but in order to do this satisfactorily, it would be necessary to divide the shells of the stoves into zones, and determine the temperature and calculate the losses from each zone separately—a rather long operation, but one worth doing.

Drying Air Blast.

The advantage of dried blast has been already discussed at length in these calculations. The advantage is due primarily to the higher temperature obtained when the moisture has been removed.

The means adopted commercially for drying the air have been those of refrigerating the uncompressed air, before its entrance into the blowing cylinders. This has the great advantage of furnishing the cylinders with cold air, and, therefore, of greatly increasing their blowing capacity, since the weight or quantity of air blown is proportional to the absolute temperature of the air entering the cylinders.

Illustration: Outside air being at 30° C., what increase in the amount of air furnished by blowing engines will result if the temperature of the air is artificially reduced to —5° C.? How much slower can the engines be driven, in the second case, in order to furnish the same weight of air as before?

The two temperatures are respectively 273 + 30 and 273 — 5 absolute, that is, 303 and 268. The engines, if run at uniform speed, would furnish 303 ÷ 268 = 1.13 times as much air in the second case, or 13 per cent more. If the engines were slowed down to 268 ÷ 303 = 0.884 of their former speed they would furnish the same amount of air; that is, they could be run 11.6 per cent slower. In fact, they could be run more than 11.6 per cent slower in the second case, and yet supply the same quantity of air, because at the slower running the delivery efficiency is somewhat higher.

The disadvantage of cooling the uncompressed blast is that it must be cooled much more, to eliminate from it a given percentage of its moisture, than if it were first compressed, and to obtain nearly dry air the moisture must be practically frozen out.

Illustration: Air at 30° C. and 85 per cent humidity is to be cooled until 95 per cent of its moisture is eliminated, without compression; to what temperature must it be cooled?

From the tables of aqueous tension, we find that the maximum tension which aqueous vapor can exert at 30° C. is 31.5 millimeters, which means practically that $\frac{31.5}{760}$ of a cubic meter of moisture accompanies $\frac{728.5}{760}$ of a cubic meter of air proper. If the humidity is 85 per cent, then this same quantity of air is accompanied by

$$\frac{31.5}{760} \times 0.85 = 0.0352 \text{ cubic meters}$$

of moisture, or, per cubic meter of air measured dry —

$$0.0352 \div \frac{728.5}{760} = 0.0368 \text{ cubic meters.}$$

If 95 per cent of this is removed by cooling, then the moisture left, per cubic meter of air measured dry, is

$$0.0368 \times 0.05 = 0.00184 \text{ cubic meters.}$$

And the relative tensions of air and moisture, to attain this dryness, must have been reduced to

$$1.0000 : 0.00184.$$

Since the sum of these tensions is always 760 mm. in the uncompressed air, the actual tensions of air and moisture will be

$$758.6 : 1.4,$$

that is, the temperature must be reduced until the moisture present can exert only 1.4 mm. pressure. This, on examining the tables of tension of aqueous vapor is found to be less than 0° C., in fact —15° C.

Problem 59.

Air at 30° C., carrying 85 per cent of its possible amount of moisture, is cooled to 0° C., and the moisture condensed to liquid water at that temperature. Barometer 760 mm.

Required:

(1) The percentage of the moisture condensed.

(2) The amount of heat to be abstracted from each cubic meter of original moist air (refrigerating effect).

(3) The percentage of moisture which would be condensed if the temperature were reduced to —5° C.

(4) The total heat to be abstracted, per cubic meter of original air, in the latter case.

Solution:

(1) From the preceding illustration we can take the figures that in the moist air taken, each cubic meter of air proper is accompanied by 0.0368 cubic meter of moisture. In the cooled air at 0°, the relative volumes of air proper and residual moisture will be as their relative tensions, viz.:

$$755.4 : 4.6$$

or

$$1 : 0.0061.$$

The moisture accompanying the given quantity of air is, therefore, reduced from 0.0368 to 0.0061, showing a condensation of 0.0307, equal to

$$\frac{0.0307}{0.0368} = 0.834 = \mathbf{83.4} \text{ per cent.} \qquad (1)$$

(2) The air at 30° contains, as before figured out, air proper and moisture in the proportions

$$1 : 0.0368.$$

Or, in 1 cubic meter of moist air, in the proportions

$$0.9645 : 0.0355.$$

We have, therefore, to calculate the heat to be extracted from 0.9645 cubic meter of air proper, falling 30° to 0°, and from 0.0355 cubic meter of water vapor, falling 30° to 0° and 83.4 per cent of the latter condensing to liquid at 0°. The figures are, remembering that the volumes heretofore handled are at 30°:

Air proper:

$$0.9645 \times \frac{273}{303} \times 0.3038 \times 30 = 7.921 \text{ Calories}$$

Moisture, if all uncondensed:

$$0.0355 \times \frac{273}{303} \times 0.3445 \times 30 = 0.332 \text{ "}$$

Heat of condensation:

$$0.0355 \times 0.834 \times \frac{273}{303} \times 0.81 \times 606.5 = 13.105$$

Total 21.357 " (2)

(3) If the temperature were reduced to —5° C. the tension of the residual moisture would be 3.4 mm. of mercury, and that of the air proper 756.6 mm., their relative volumes would be in the same ratio, viz.:

$$756.6 : 3.4$$

or

$$1 : 0.0045,$$

showing that out of 0.0368 cubic meter of moisture originally accompanying 1 cubic meter of air proper, 0.0323 had been condensed, or

$$\frac{0.0323}{0.0368} = 0.878 = \mathbf{87.8} \text{ per cent.} \qquad (3)$$

(4) The condensation is seen to be 87.8 — 83.4, or 4.4 per cent more, if the air is cooled to —5° C. A considerably larger amount of refrigeration, however, is required in this case, because the moisture would all be frozen. The 1 cubic meter of moist air at 30°, containing, as before calculated, 0.9645 cubic meter of air proper and 0.0355 cubic meter of moisture, would have 87.8 per cent of the latter condensed to ice, or 0.03117 cubic meter, and, therefore, 0.00433 cubic meter remaining uncondensed.

Air proper, 30° to —5° C.:

$$0.9645 \times \frac{273}{303} \times 0.3037 \times 35 = 9.237 \text{ Calories}$$

Moisture, if all uncondensed, 30° to —5° C.:

$$0.0355 \times \frac{273}{303} \times 0.3438 \times 35 = 0.385 \text{ "}$$

Condensation of 0.03177 m^3 to liq. at —5° C.:

$$0.03177 \times \frac{273}{303} \times 0.81 \times 605 = 14.025 \text{ "}$$

Freezing of same, at —5° C.:

$$0.03177 \times \frac{273}{303} \times 0.81 \times 80 = 1.855 \text{ "}$$

$$= 25.502 \text{ "} \qquad (4)$$

The conclusion is, that an increased condensation of 4.4 per cent has been obtained by an increase in the refrigerating requirement of 25.502 — 21.357 = 4.145 Calories, or nearly 20 per cent. Another way of stating the comparison is that when not freezing the moisture condensed, about 4 per cent of the moisture was condensed from each cubic meter of air for one Calorie of refrigeration, whereas, the removal of additional moisture by cooling below zero requires nearly one Calorie refrigeration for each additional per cent of moisture eliminated.

A practical conclusion is, that the expense of refrigeration might easily be justified down to 0° C., and yet be unjustified by the results when cooling below 0°.

Mr. James Gayley has, I believe, patented the scheme of refrigerating the air in two stages, first, nearly to zero, removing the moisture thus condensed as liquid, and then cooling the nearly dry air further, eliminating more moisture as

ice. In this manner, the amount of refrigeration required is reduced by the latent heat of solidification of the larger part of the moisture, and cooling below zero becomes profitable. The saving in the above example would be 80 Calories per kilogram on all the moisture condensed at 0°, viz.:

$$0.0216 \times 80 = 1.728 \text{ Calories.}$$

Cutting down the refrigeration required from 25.502 to 23.774 Calories, that is, enabling 4.4 per cent additional drying to be produced for 2.4 Calories additional refrigeration. This scheme of Mr. Gayley is founded on sound scientific as well as practical considerations.

The idea of cooling the compressed blast by moderately cool water, recently proposed, is also a very practical idea and founded on sound scientific principles. At a given temperature moisture cannot exert more than a maximum vapor tension. It follows that if we start with air saturated with moisture, at a given temperature, and compress it to double its initial tension, keeping its temperature constant, about half of its moisture must condense out. If, at the same time, it is artificially cooled, then more than half of its moisture will condense as liquid.

If we start with air not saturated with moisture, the compression, temperature being constant, will increase the tension of the moisture present until the air becomes saturated, after which increased pressure will cause condensation.

Illustration: Air at 30° C., 85 per cent saturated with moisture, is compressed. At what effective pressure does it become saturated with moisture, temperature remaining constant?

The moisture present is exerting 85 per cent of its maximum vapor tension at this temperature. Therefore, the tension must be increased in the ratio of 85 to 100, to make the air saturated, viz.: in the ratio 1 to 1.177. The effective pressure necessary to be applied is, therefore, 1.177 — 1.000 = 0.177 atmospheres (2.6 pounds per square inch).

Illustration: If air at 30° C., 85 per cent saturated with moisture, is compressed to one atmosphere effective pressure and its temperature kept constant, what proportion of its moisture will condense?

Before compression, the tension of the moisture being $31.5 \times 0.85 = 26.8$ mm., the relative volumes of air proper and moisture are as

$$733.2 : 26.8$$

or, as

$$1 : 0.0367$$

After compression, the tension on the mixture being two atmospheres (1520 mm.), and the tension of the uncondensed moisture being its maximum or 31.5 mm., the relative volumes of air and uncondensed moisture will be as

$$1488.5 : 31.5,$$

or, as

$$1 : 0.0212.$$

The proportion of the original moisture remaining uncondensed, is therefore.

$$\frac{0.0212}{0.0367} = 0.578 = 57.8 \text{ per cent.}$$

and the amount condensed out = 42.2 per cent

Problem 60.

Air at 30° C., and 85 per cent saturated with moisture is compressed to one atmosphere effective pressure (760 mm. of mercury), and simultaneously cooled by river water to 10° C. Barometer 730 mm.

Required:

(1) The percentage of the original moisture condensed.

(2) The weight of moisture remaining in the air, expressed in grams per cubic meter of dry air at standard conditions (1 c. per 1.293 kilograms of air proper).

Solution. (1)

Tension of uncompressed moist air = 730.0 mm.
Tension of moisture present 31.5×0.85 = 26.8 "
Tension of air proper, uncompressed = 703.2 "
Relative volumes of air proper and moisture = 1 : 0.0381
Tension of compressed moist air 730 — 760 = 1490.0 mm.
Tension of uncondensed moisture (maximum tension at 10°) = 9.1 "
Tension of air proper, when compressed = 1480.9 "
Relative volumes of air proper and uncondensed moisture = 1 : 0.0061

Proportion of moisture condensed:

$$\frac{0.0381 - 0.0061}{0.0381} = 0.84 = \mathbf{84.0} \text{ per cent.} \qquad (1)$$

(2) The relative volumes of air proper and uncondensed moisture are, as found above.

$$1 : 0.0061.$$

And the relative specific gravities of air proper and moisture are as the standard weights of 1 cubic meter of each, viz.: as

$$1.293 : 0.81.$$

It follows, therefore, that 1.293 kilos. of air proper (1 cubic meter at standard conditions) must be accompanied by

$$0.0061 \times 0.81 = 0.0049 \text{ kg. of moisture,}$$

or $= \mathbf{4.9}$ grams. (2)

The original moist air contained, similarly considered,

$$0.0381 \times 0.81 = 0.0309 \text{ kg.}$$
$$= \mathbf{30.9} \text{ grams.}$$

The Electrical Discharge in Air and its Commercial Application.[1]

By William Cramp and Sidney Leetham.

It was in 1903 that Mr. Sidney Leetham discovered, after experimenting with the bleaching of flour by means of ozone, that a much more powerful bleaching agent than ozone was produced if air that had been ozonized were passed through boxes in which electrical discharges between spark points were taking place. It was evident from the smell of the gas that the new product was different to the old, and some time was spent in noticing and attempting to ascertain clearly what this new product was. Dr. H. E. Armstrong made an analysis and about that time wrote: "I am much struck with the nature of the results obtained, viz.: the rapidity and certainty with which the effect is produced and the improved bleach as compared with that produced by oxides of nitrogen. I believe * * * that it is a joint effect of ozone and oxides of nitrogen, which I find are both present, though the total amount is not more than two parts in 10,000. I am satisfied that by this mixture an effect is produced which is so far better than anything that can be attained by either of the constituents as to be accounted a *new technical effect.*"

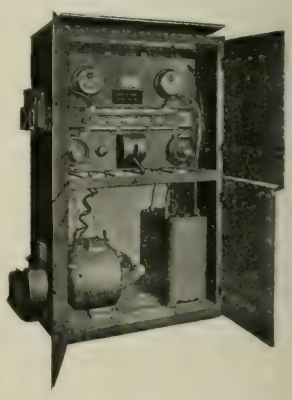

Fig. 1.—Electric Discharge Apparatus.

[1] A paper read at the York meeting of the British Association for the Advancement of Science.

Construction of Apparatus.

The apparatus as at present used is illustrated in Fig. 1, and consists of a steel plate case, in which is fixed an alternator, transformer, switchboard, ozonizer and sparking device; the whole being compact, self-contained and enclosed, occupies very small space and incurs no extra insurance risk. Besides the apparatus already mentioned, a small Roots blower is used for supplying air to be ozonized, and a small filter for cleaning this air. The action of the apparatus is as follows:

Pure air cleared from dust particles by passage through the filter is supplied by the blower to the ozonizer (situated behind the switchboard), where it becomes ozonized, and also to

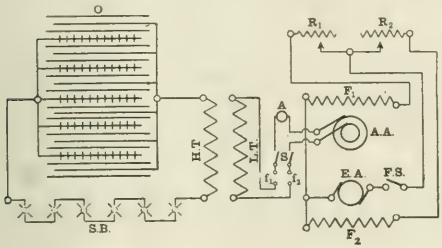

FIG. 2.—DIAGRAM OF CONNECTIONS OF THE APPARATUS, FIG. 1.

some extent sterilized. The ozonized air then passes on to the spark box—seen across the middle of the switchboard—where a very small proportion of oxides of nitrogen appears to be produced by the sparks between the points in the box. From the spark box the air passes through a valve flange (seen on the left-hand end of the case) straight to the reels through which the product to be bleached is passing.

The actual electric circuits are as shown in the diagram of connections (Fig. 2); where the high-tension and low-tension coils of the transformer are denoted by HT and LT respectively; O is the ozonizer of the Andreoli pattern; SB is the spark box, in which are a number of spark gaps in series; AA is the alternator armature and EA is the exciter armature; F_1 and F_2 are the field coils, and R_1 and R_2 the regulating rheostats. It will be noticed that the ozonizer and spark box are in series, and this is a great advantage in rendering the apparatus automatic to a certain extent. For every increase of current affects both ozonozer and sparks, so that within certain limits the proportions of the gas remain constant; though, of course, the yield becomes greater, as more fully discussed later

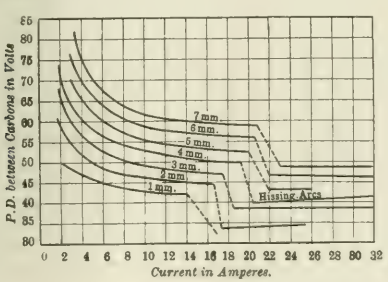

FIG. 3.—POTENTIAL DIFFERENCE AND CURRENT FOR DIFFERENT ARC LENGTHS (MRS. AYRTON). POSITIVE 11 MM., NEGATIVE 9 MM.

It may be stated here that with the normal amount of air used, which is approximately 100 cubic feet per minute, the resulting gas, which issues from a point some 3 feet beyond the end of the delivery flange of the iron case, is made up about as follows: Air, 40,000 parts by volume; ozone, three parts by volume; oxides of nitrogen, one part by volume (when four sparks are used, each 1-16 inch long, and the LT current is 8 amps.).

This result is the average of a large number of analyses made by Mr. F. S. Sinnatt, of the Manchester School of Technology, and will be seen to agree fairly well with Dr. Armstrong's rough estimate. It will, however, be clearly understood that variations in the above result are immediately introduced by change in any of the following factors: Frequency, air stream, distance apart of spark points and number of these, shape of points, temperature, current. Of these the last two influence the chemical phenomena chiefly. From a scientific point of view the interest lies in the question of how these various items affect not only the gas produced but also the type of discharge which produces it. We shall take the latter point first.

Forms of Electric Discharges.

Hitherto all types of alternating electrical discharges have been divided into three great classes: (1) the ordinary alternating-current arc; (2) the high-tension discharge between spark points; (3) the silent discharge. These three have usually been considered as separate and distinct phenomena, but they have never been individually defined. In reality they are not separate, but all stages of the same phenomenon—namely, the breakdown of air as an insulator when strained electrically. Roughly, the three types have been usually classified by the effect which they produce. In this way the arc has been recognized as that type of discharge which gives great heat and light. The sparking discharge has been recognized as that which is much less hot, though still luminous, and is known to produce Hertz vibrations. The silent discharge is usually referred to as that which produces ozone. Now, if a pair of metal points be insulated and separated in air by a small distance, and an electric potential difference be applied across the gap, we have the following series of events as the pressure is increased:

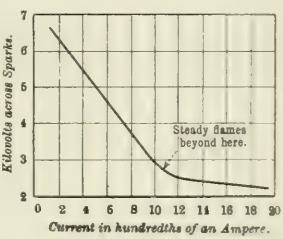

FIG. 4.—SPARKS ONLY THREE IN SERIES 1/16 INCH.

(*a*) The air is put into a state of stress

(*b*) Small conducting paths appear through the air as the resistance gives way, giving rise to small blue streamers issuing from the points.

(*c*) The blue streamers get thicker, eventually joining the points up.

(*d*) The thicker streamers become a discharge very like the ordinary arc. At the final stage the points heat up, and the current passing may be gradually increased till the discharge is practically the alternating-current arc

Throughout the changes given above the relation between current and e. m. f. across the gap is very remarkable. It has been known for a long time that an arc, either direct or alternating current, cannot be maintained on a constant potential circuit unless a steadying resistance or its equivalent is kept in series with it. This has given rise to discussion as to whether a back e. m. f. is present in the arc or not, though it is generally held now that change of air resistance alone is sufficient to account for the peculiarities observed. Whether a back e. m. f. exists or no; hardly comes into the province of this paper, but it is perfectly certain that the apparent resistance of the arc changes as the current through it changes. In the case of ordinary arc lights the results immediately met

with are generally very like those in Fig. 3, which represents the relationship between e. m. f. and current in the direct-current arc, as given by Mrs. Ayrton in her book *The Electric Arc*.

Now, the very same sort of relationship is found to exist with both the other forms of discharge. It will be recognized that the cycle of changes given in *a, b, c, d* above includes not only the spark discharge but also the so-called silent discharge. In Figs. 4 and 5 are shown curves connecting the e. m. f. and current for cases which include both the silent and the spark discharge. It will be seen that just as in the case of the arc the fall of potential is very rapid as the current increases, but finally settles down until it almost becomes proportional to the current.[2] The diagrams not only show the agreement between the shape of the curves of Figs. 4 and 5 and that given in Fig. 3, which is very important, but also emphasizes the fact that it is very difficult to draw any hard-and-fast lines between the different types of the discharge.

It should be explained that as it was desired to act upon

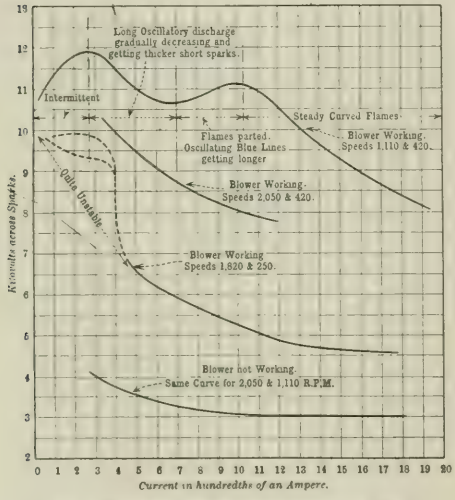
Long Oscillatory discharge gradually decreasing and getting thicker short sparks.
Intermittent
Steady Curved Flames
Flames parted. Oscillating Blue Lines getting longer
Blower Working. Speeds 1,110 & 420.
Blower Working. Speeds 2,050 & 420.
Quite Unstable
Blower Working Speeds 1,820 & 250.
Blower not Working.
Same Curve for 2,050 & 1,110 R.P.M.
Kilovolts across Sparks.
Current in hundredths of an Ampere.

FIG. 5.—FIVE SPARKS, EACH OF 1/16-INCH GAP.

air with the various forms of discharge a current of air had to be kept passing through the apparatus. This is why most of the curves above are taken with a certain air velocity, which is given in terms of the blower speed. Thus in Fig. 5, the numbers 420 and 250 refer to the speed of a 2½-inch Roots blower, and correspond to an air stream through the spark box of about 3,200 feet and 1,900 feet per minute respectively. The effect of this air velocity is (as shown) to increase the e. m. f. with a given current; but this will be discussed more fully directly. The earlier portion of the curves—that is, those parts in which the e. m. f. is high—usually refer to that discharge which produces chiefly ozone. Directly the curve has turned round, so that there is no longer a large fall in e. m. f. for a given current, very little ozone is produced.

It is clearly seen from the curves: (1) That the potential difference across the spark-gap gradually decreases as the current increases, showing clearly that the resistance of all electric discharges varies in a similar manner to that well known to exist in the case of the ordinary arc.

(2) That the character of the product of the discharge depends upon the voltage at the terminals for a given current; thus with the conditions shown in Fig. 4 practically no yield

[2] In all the curves connecting e. m. f. and current, where the e. m. f. is in kilovolts, the current is in hundredths of an ampere.

of oxides of nitrogen occurs until a point is reached marked with an arrow on the diagram. To the right of this arrow the discharge would be a small but rather hot arc, were it not for the presence of the air stream which blows the discharge into a short flame. In other words, up up to this point the discharge is to be considered practically as a silent discharge, afterwards as the so-called sparking discharge. The change, however, from one form to another takes place fairly gradually.

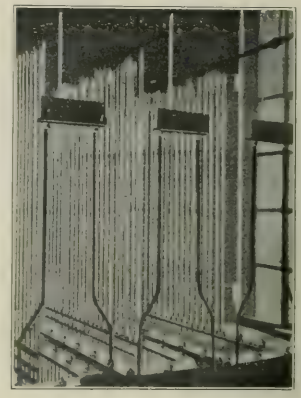

FIG. 6.—SCHNELLER OZONIZER.

(3) Any character of discharge may be obtained by simply regulating the fall of potential at the discharge points or terminals, so as to fulfil the following requirements: Since the resistance (or impedance) of the air gap across which the spark passes appears to depend not only upon its length but also on the current passing (being lower with a larger current) it follows that if there were a constant pressure on the arc, having once started the discharge, the current would rise till the product $c \times R = E$, where R is the resistance or impedance of the arc, E the applied potential difference and C the current. If R tends to decrease faster than C rises, the conditions are quite unstable and result in a long series of flashes across the points, which were observed often in the case of these experiments. These flashes are produced by a large rush of current which is only limited by the armature reaction of the generator. In consequence of this, beyond a certain point the e. m. f. and potential difference of the latter fall, so that the current almost instantly falls to zero. As soon as this has happened the e. m. f. rises again, the spark is re-established, another rush of current takes place with a similar result. Often this would take place as frequently as sixty times a minute. To steady the discharge then it is only necessary to arrange the circuit so that an increase of current shall produce at the spark terminals a fall of potential difference sufficiently great, and in this way, by proper adjustment of the circuit, a perfectly steady result will be obtained.

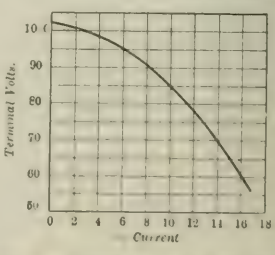

FIG. 7.—ALTERNATOR CHARACTERISTIC.

The only other conditions necessary for producing and maintaining the discharge are (1) that the maximum potential difference of the e. m. f. wave shall be capable of crossing the spark gap; (2) that the current ensuing shall be great enough to so lower the resistance of the gap that the vapor column set up shall not entirely die away before the zero point is crossed and the current re-established.

The adjustment of the circuit just shown to be necessary may be carried out in many ways. In the author's apparatus it is obtained by the use of a heavy synchronous armature impedance, but it may be almost as well obtained by resistance or choking coils. The higher up the scale the discharge the

greater must be the resistance employed; thus, if a silent discharge is required, the resistance drop will need to be from one-third to one-half of the terminal voltage.

An instance of such an ozonizer is the well-known Schneller ozonizer shown in Fig. 6, where the potential drop is obtained by means of a tube of glycerine, through which the current passes to those plates, between which the silent discharge takes place. If the ordinary constant potential arc is required, an impedance drop of about one-quarter the terminal voltage is used, while the characteristic of the alternator used by the authors for their purpose is as shown in Fig. 7. The amount of steadying resistance for any type of discharge is affected by:

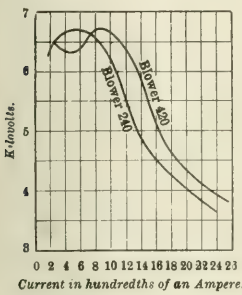

FIG. 8.—EFFECT OF AIR VELOCITY.

1. *The Frequency Employed.*—The higher the frequency the less likely is the discharge to be interrupted, and hence the lower is the pressure for a given current. This is well shown in Fig. 5. It is, however, curious that in some cases for very small currents the pressure required is actually raised by raising the frequency. In the apparatus as used by the authors, where the ozonizer and spark box are in series, though, as is shown later, the yield of ozone per amp. may be increased by raising the frequency, yet because the spark seems to have some difficulty in starting, and because the instability of the circuit does not seem to be reduced by higher frequencies, a practical limit is reached at about 150 to 200 periods per second.

2. *Air Velocity.*—The general effect of increased air velocity is to increase the apparent length, and hence, also, the apparent resistance of the spark, and with high air velocities the spark becomes blown out or extended into a flame. The general effects of the air velocity are shown in Fig. 5, and again in Fig. 8. In these diagrams the meaning of the figures 420, 250, 240 has already been referred to. Fig. 5, however, has a second set of figures upon it, viz.: 1,110, 2,050 and 1,820. These refer to the alternator speeds, and correspond to frequencies of 148, 273 and 243 respectively. In this way, while Fig. 8 shows the effect of change of air velocity only, Fig. 5 combines with this the effect of change of frequency. The obvious deductions from these curves are that higher frequencies conduce to lower the pressure for a given current, and higher air velocities conduce to higher pressure for the same current.

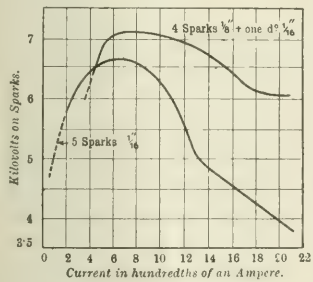

FIG. 9.—EFFECT OF SPARK LENGTH (BLOWER 240 REVOLUTIONS).

It may be observed that, besides these obvious results, it is also found that a high air velocity sets up very serious oscillations in the circuit, which have many times been responsible for breaking down the high-tension transformers used in these experiments One would almost expect this result from the fact that the higher the air velocity the more likely is the spark to be very unsteady and very often extinguished. These oscillations show themselves in the curves in giving [illegible] the unstable portion and causing it to exist at higher currents; and, finally, on account of their disastrous effects it was found necessary to limit the air velocity in the apparatus as at present used to a maximum of 2,500 feet per minute, with the distance between spark points, say 1/16 inch.

3. *Distance Apart of Spark Points.*—Many tests were taken with different spark lengths, as, for instance, five sparks 1/16 inch long in series were compared with four sparks of 1/8 inch long, and one of 1/16 inch. In the latter case [illegible] in the circuit are found to be greater than with the former, and this general result is true whenever the length of spark is increased. Again, the potential difference with the long spark gaps is always greater than in the case of the shorter, but this effect is not proportional to the number of sparks in series in the apparatus, nor so marked as the increase in oscillations, especially with low currents. So important is the latter that in certain instances it is not possible to measure the voltage across the sparks at all.

It might be expected that the voltage would be almost proportional to the number of sparks in series, but this is not found to be the case at all. On the contrary, Fig. 10 shows that the voltage required for three sparks (each 1/16 inch long) in series, is quite different to that required for one spark 3/16 inch long; further, Fig. 9 shows that four sparks, 1/8 inch long in series, and one of 1/16 inch long, do not take, with small currents, anything like twice the voltage of five sparks 1/16 inch long, as might have been expected. From a great number of experiments the phenomena above detailed seem to be largely explained by the following considerations:

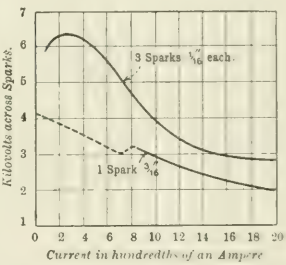

FIG. 10.—EFFECT OF SPARK LENGTH (BLOWER 265 REVOLUTIONS).

1. If the spark is a long one the air velocity has more effect upon the discharge, since the points protect it less.

2. Any capacity effect due to the points themselves is more marked when the points are close together.

3. If a vapor column is set up, the longer it is the more easily is it cooled.

These three causes, even taken with others referred to above, do not explain all the effects observed. We must assign to other causes the small increase of pressure for increase of spark length, for instance; also the comparative ease with which the discharges start if no air stream is present (see Fig. 5). The tests given below upon the question of ionization will perhaps afford a clue to these points. In the meantime we may mention that the surface leakage over the insulating porcelain has many times been responsible for starting one spark of a series before the others begin, so that it is usually found that four sparks 1/16 inch in series will start at a much lower voltage than one spark 1/4 inch long. But this fact alone will not explain the phenomenon that to restart the discharge after it has been running for some time requires a much lower pressure than to start the discharge at first, which must also be referred to the state of so-called ionization in which the gas exists.

Effect of Shape of Points.—The shape of the point changes very considerably the apparent resistance of the discharge, and also, in consequence, the type of discharge produced.

It is well known, in connection with high tension discharges that the sharper the points between which the current passes the lower is the pressure at which the discharge will commence

As an example of this we can give some results obtained at the Manchester School of Technology, by Mr. Bertram Scott. In these tests three types of electrodes were used, viz.:

1. Flat electrodes with rounded edges (in pairs).
2. Flat electrodes with sharp edges and spherical electrodes (one of each).
3. Sharp or pointed electrodes (in pairs).

The results, so far as they influence the pressures used in this apparatus, are as follows:

1. Flat electrodes with rounded edges.

Kilovolts.	*Sparking Distance in Cm.*
18	1.8
29.5	3.9
40	6

2. Flat electrodes with sharp edges and spherical electrodes.

Kilovolts.	*Sparking Distance in Cm.*
15	2
24	4
29	5
38	7
46	9

3. Sharp electrodes.

With these it was found that the effect of the sharpness was more marked the less the angle at the apex. The most marked results were obtained with needle points, and it will be noticed that, with such shaped points, the voltage required for a sparking distance of 2 cm. is about half that given in the first case. The figures are as follows:

Kilovolts.	*Distance in Cm.*
9	2.0
20	4.0
28	6.8
36	9.0
45	10.9

All the above results, of course, would change slightly with change of wave-form; in these cases an alternator was employed whose wave had a form factor of 1.54, so that the shape was approximately sinusoidal. Fig. 10A illustrates these readings graphically.

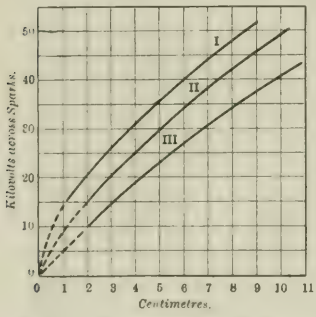

FIG. 10A.—SPARKING DISTANCE IN AIR.

The voltages given are not applicable to the actual apparatus under consideration, since no air current was present except that which would be produced by the actual heat of the discharge; and, since the results were taken when the points were vertical, this would have very little effect. When, however, an air current plays on the discharge, a result similar to the above is always experienced, but the voltage for a given sparking discharge is higher, the actual increase of voltage being of the order of that shown in Fig. 5. It was, therefore, most advantageous for this particular plant, where low voltage is desirable, that the shape of the points should be as sharp as possible. The final shape has been determined by the question of the life of the spark points—that is, they are so proportioned that they will dissipate the heat generated between them fast enough to necessitate no adjustment of the points, at any rate more often than once a month. The point used now is made from a screw ¼ inch in diameter, with an apex angle of 90°, very slightly rounded at the extreme point. It is found that up to 0.2 amp., with the normal air velocity as already given, six pairs of these points in a spark box do not materially change when at work night and day for a period of three weeks.

The foregoing discussion has been confined chiefly to those portions of the curves where the discharge produced is of the type known as "sparking." It is necessary now to refer to the upper parts, where, as has been already said, ozone is produced. The figures show quite clearly that in these steep parts the discharge is very unstable, and in consequence a large steadying resistance is necessary, for the current is, in any case, extremely small. This large steadying resistance (or impedance) in commercial ozonizers takes the form of either:

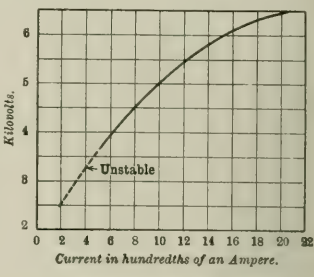

FIG. 11.—SPEED 1,200.

1. A dielectric between the points.
2. Very high resistance in series with the points, such as the glycerine tubes of Schneller.
3. A capacity in series with the points.

Examples of No. 1 are the ozonizers of Siemens, Andreoli, Elworthy, etc., which usually consist of two conductors, either plain or carrying points separated by mica or glass. In cases like this the fall of potential, which has been shown is necessary to maintain this silent discharge, takes place across the dielectric itself.

Where the points are placed upon sheets, and separated by sheets of mica (Andreoli), the ozonizer may have a large capacity, and in this case the effect of the breakdown of the resistance is very clearly marked. By the laws of condensers, if the ozonizer works as a capacity alone, the current would be proportional to the terminal potential difference; and in this case, therefore, the "curve" connecting voltage and current would be a straight line. If, then, such a curve be drawn for an ozonizer, and it is found to depart from the straight line, we should expect to find the yield of ozone increase as the curve turns over. This effect is clearly shown in Fig. 11, and it was always found that the yield of ozone increased as the curve departed from the straight line, until the vigor of the discharge caused the temperature to rise to such an extent as to partially destroy the ozone produced.

Ozonizers of the second type are not, as a rule, so efficient as the former. They are, however, easy to construct, and, as a rule, less bulky. An ozonizer of this type was constructed by Mr. Cramp, which consisted of a number of serrated strips of aluminium, placed with their serrated edges about ½ inch apart. It was found that the minimum air gap between these edges, at which a steady, silent discharge, producing large quantities of ozone, could be obtained, was about 7/16 inch; and in the experiment in question 9/16 inch actually gave about the best result.

In order to steady the discharge a glycerine resistance had to be included in the circuit, of such dimensions that the drop across the resistance was practically equal to the drop across the air gap. By gradually lowering this resistance the discharge could be changed from the silent distributed discharge to the located spark. A large fall of potential was always observed as the discharge changed from one form to the other.

Ozonizers of the third form are not made commercially, and the reason is probably that with any tendency of the air gap to break down, the condenser immediately discharges through the circuit, assisting the breakdown; hence the type cannot be

recommended, but it has been found possible to use, in series with the gaps described above, a small condenser which, by the fall of potential which it produced across the gaps, is enabled to a certain extent to steady the silent discharge. Of course, the capacity of the condenser must be of such a value that it cannot (by causing a leading current) raise to any extent the alternator voltage by means of armature reaction.

Summarizing now the general results of this investigation we have:

(*a*) All the types of electrical discharge in air behave similarly, and may be steadied by similar means.

(*b*) Between a pair of spark points in air any type of discharge may be obtained by suitable regulation—from the silent type producing ozone down to the alternating-current arc.

(*c*) Increasing the number of air gaps in series raises the voltage, but not proportionally to the number of gaps.

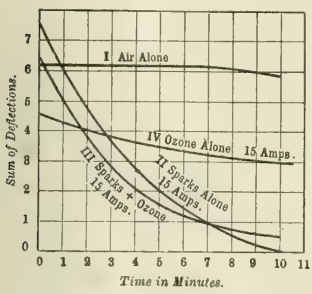
I Air Alone
IV Ozone Alone 15 Amps.
II Sparks Alone 15 Amps.
III Sparks + Ozone 15 Amps.
Sum of Deflections.
Time in Minutes.
0 1 2 3 4 5 6 7 8 9 10 11
0 1 2 3 4 5 6 7

FIG. 12.—IONIZATION EXPERIMENTS.

(*d*) An air current, impinging on the electric discharge, increases the mean length of path of the discharge; with large currents, blows it into a flame, renders the spark unable to pass without increased pressure and always sets up oscillations in the circuit.

(*e*) The effect of increasing the frequency is to lower the voltage required to maintain the spark, but this does not necessarily hold at starting.

Electrochemcal Experiments.

So many of the results obtained, both electrically and chemically, seemed inexplicable on ordinary grounds, that it was thought advisable to try the gases produced by the electrical discharge for *ionization*.[3] Accordingly, a series of curves were taken, which show the time required to discharge a good electroscope:

(*a*) By allowing the stream of air alone to impinge on the knob of the electroscope.

(*b*) By substituting for this a stream of air which had passed through the ozonizer.

(*c*) By substituting a stream of air which had been modified by passing through the spark box.

(*d*) By combining *b* and *c*.

The results are very clearly shown in Fig. 12. They indicate that the air, after having passed through the chambers in which electrification was proceeding, is always in a state of ionization. This result is much more marked in the case of air issuing from the spark box than in the case of air which has simply passed through the ozonizer. It is to be considered in connection with the chemical phenomena, but may also be discussed here with reference to the electrical effects.

Suggested Further Theoretical Explanation of Results Obtained.—It seemed almost impossible to imagine that these discharges, which require so small a current, could, by any possibility, produce enough heat to set up a vapor column sufficient to maintain a conducting path for the electric current; but, on the other hand, in the absence of the vapor-column theory, there seemed to be no reason why, first, the discharge should be more difficult to start when the air current was flowing, and, secondly, why it should be so much more difficult to maintain with an air current. Admit, however, that ionization is established and we immediately have an explanation for all the phenomena.

Thus, under electrical stress, the air is ionized between the points, and begins to tend to conduct. Small currents make their appearance as blue lines, but whether these blue lines are actually heated particles or not cannot be determined. Further increase of the current results in further ionization, with a better conducting path; but this conducting path is, under any circumstances, easily removed if a current of air is maintained past and around the points. Thus, the discharge would be more difficult to start and more difficult to maintain. Going still a step further we get towards the ordinary arc, and find that a vapor column is set up, which, of course, continues to behave very much as the ionized air column already suggested.

This is, in brief, the theory put forward by the authors to explain the facts already related, and it accords well with previous knowledge of the same subject. Of course, "ionization" would be immediately admitted by many as sufficient explanation for the whole of the phenomena, and we wish to insist upon the point that without some such theory some of the above facts are inexplicable, especially the point that the discharge starts more easily after it has been at work some time and then extinguished, than it does when first set to work. This characteristic is very marked with low air velocities and when the time between stopping and restarting is short. The same may be noticed with a high-tension electrostatic voltmeter. If it flashes over once, the case must be opened and fresh air introduced, or it will go over again at a comparatively low pressure. We think also that this phenomenon will almost certainly account for the extraordinary differences in so-called "sparking lengths," which have been so often observed and commented upon. Such differences have often been put down to damp and dust, but the authors' experience is that these factors will not altogether account for them. More especially the chemical results, the union of oxygen and nitrogen particularly, seem impossible without reference to some such theory, as will be seen presently.

Chemical Phenomena.

It has hitherto been usual to speak of the silent discharge as though it were a perfectly well known effect, producing a chemical result, well defined. The following experiments will dispel this idea: If an Andreoli ozonizer, such as that whose characteristic is shown in Fig. 11, be connected to alternate-current mains, and the pressure be gradually increased, it is found that a violet glow is set up between grids and mica which produces an excellent yield of ozone; increase of pressure increases the yield of ozone up to a certain point, beyond which the discharge becomes gradually more yellow, and finally quite a large percentage of oxides of nitrogen is produced, the yield of ozone being diminished. Whether ozone is really formed and afterwards destroyed, or actually never formed, is uncertain. Further increase of current still further increases the yield of oxides of nitrogen, though usually by this time long leakage discharges appear over the dielectric.

The effects are still more marked if an ozonizer without a dielectric other than air be used, as then the discharge can be gradually increased till practically no ozone is produced; thus the silent discharge becomes the sparking discharge, yielding chiefly NO_2; and however much the current be increased after this, even to the flaming arc, the gas produced is still NO_2. We have found no type of discharge, however silent, which yields no oxides of nitrogen.

We can offer no explanation whatever for the production of ozone by the initial types of discharge. But there are some suggestions we can make as to the formation of oxides of nitrogen. It has been generally assumed that oxides of nitrogen are formed electrically by nitrogen burning in the air, the heat necessary being supplied by the flame of the arc or spark, as pointed out in 1898 by Crookes. Now, the temperature of burning nitrogen is supposed to be about 2000° C; and though

[3] The term "ionization," as used throughout, indicates simply capability of discharging an electroscope.

it is impossible to see if this temperature ever exists in the sparking discharge, it seemed almost incredible that such could be the case. In order to try and settle this point, recourse was had to the following rough tests:

Temperature Effect.—A spark box with five pairs of points 1/16 inch apart was placed in circuit and the current adjusted to 0.1 amp. Tests were then made while the spark box was cold (20° C.), and again when heated to about 120° C., but no definite increase in the percentage of oxides of nitrogen was observed. It would therefore seem that the yield is not altogether a matter of temperature. With four 1/16-inch sparks in series, an air velocity of 2,000 feet per minute, and a current of 0.06 amp., the yield of oxides of nitrogen was 1 part in 40,000, as tested by Mr. Sinnatt many times.

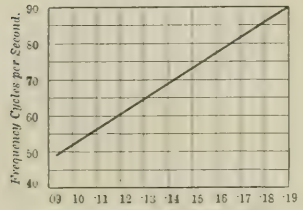

FIG. 14.—VARIATION IN YIELD OF OZONE WITH FREQUENCY.

Again, it is always found that when air with a small percentage of ozone is passed through the spark box the chief production of oxides of nitrogen does not take place in the spark box, but a little further on, in our case after the gas has passed through about 3 feet of pipe; pipe, too, which is perfectly cool. Or again, if in such a case the small sparks be replaced by white-hot Nernst filaments, no oxides of nitrogen are formed at all. And if further confirmation of the formation being independent of the temperature be needed, we can say that if pure oxygen mixed with 2 per cent of nitrogen be sent through a water-cooled Berthelot ozonizer, peroxide of nitrogen is still found to be present.

Current.—On the other hand, it is always found that the yield of oxides of nitrogen does not increase proportionately to the current, but more slowly. Indeed, in the spark box above referred to there is scarcely any change in the yield of oxides of nitrogen between 15 and 25 amps., which is very remarkable, inasmuch as not only is the current so much greater, but the temperature is also so much higher. On the other hand, between 5 and 10 amps. the yield is very much increased.

The reasons for these effects the authors consider to be as follows:

The best yields per ampere are always found to exist when very thin arcs are used, and this is confirmed by Bradley and Lovejoy, in America. That means, the air used must have easy access to the sparks if a large yield of oxides of nitrogen is desired; this condition accounts for the amount produced not being proportional to the current.

Now, in Fig. 12 it has been shown that the air modified by a sparking discharge is more highly ionized than that obtained by a silent discharge. Hence we suggest that ionization of the air allows of the combination of oxygen and nitrogen at fairly low temperatures to produce oxides of nitrogen, but it is very necessary that the air should be in intimate connection with the discharge to become thoroughly ionized.

This theory would seem to be well supported by the fact that replacing the sparks by apparatus which must surely be hotter—such, for instance, as a Nernst filament or a white hot electric radiator—has no effect at all.

With regard to the effects produced on the gas by electrical variations in the circuit, it may be said that a great number of experiments have been carried out, but only the general results can be given here. These are:

Increase of frequency increases the yield of ozone, as shown by Prépegnier (Fig. 14), and lowers the pressure for a given current through the spark box.

Increase of air velocity does not seem to lower the yield, but only the concentration.

Any increase in spark length increases the yield of oxides of nitrogen.

So much for the independent effects of different types of discharge. We come now to the most important part of the paper, viz.: the discovery that, though the gas produced by the sparks (N_2O_4) is valuable as a bleaching and sterilizing agent, and the gas produced by the ozonizer (ozone) is valuable in the same way, a combination of the two is far more valuable than either.

Reference to the first part of this paper will show the arrangement of the apparatus by which the Leetham gas is produced. It is obtained by passing air modified by an ozonizer through a box in which a sparking discharge is produced. An analysis of the gas produced has already been given, and it only remains to discuss its peculiar characteristics.

It has been pointed out as a curious fact that the chief modifications produced by the spark box are not discernible at the end of the spark box itself, but at a point in the pipe situated about 3 feet therefrom. This supports the theory already given for the production of N_2O_4. Beyond this 3 feet or 4 feet the gas analyses show no change, and it is an undoubted fact that, even after the gas has passed through many feet of pipe, ozone and oxides of nitrogen are still traceable. We regard this as very important, for eminent chemists (Sir James Dewar and others) have stated again and again that such a state of things is impossible; *i. e.*, that oxides of nitrogen and ozone cannot exist side by side. This simultaneous existence of two gases, well known to be mutually destructive, is probably the key to the whole extraordinary activity of the compound. As an instance of the great effect the gas has, it may be said that modified air, having a composition as shown in section I., will, if supplied at the rate of about 100 cubic feet per minute, bleach to a beautiful white color as much as 3 tons of flour an hour; a wonderful effect when the proportions of bleaching agents present are considered.

In order to try and discover the means whereby this electrical product yields such astonishing effects, very many experiments have been carried out both by the authors and Mr. J. S. Peachey, of the Manchester School of Technology. It is of interest to inquire, firstly, whether the bleaching action is due to oxidation. In order to test this, ozone, most carefully freed from oxides of nitrogen by thorough washing with pure concentrated sulphuric acid, was allowed to come into contact with flour. *Result: the flour was bleached.* An exactly similar effect was produced if the coloring matter of the flour was separated from the other constituents and treated with pure ozone. Therefore, the action may be ascribed to oxidation.

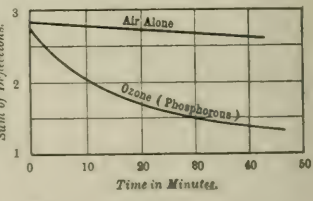

FIG. 13.—IONIZATION EXPERIMENTS.

Secondly, it is interesting to know whether the ozone and oxides of nitrogen act similarly as bleaching agents. So far this is not quite certain, but it is perfectly certain that the ultimate effect of the two bleachers is different. For ozone, however often applied, never has any other effect than that of bleaching, while oxides of nitrogen only bleach up to a certain point, beyond which they turn the flour darker and darker until it is a sepia color. The latter effect is probably due to the presence of nitric acid.

So far as we have yet discovered oxides of nitrogen and ozone bleach flour similarly, though the former has other effects besides that of bleaching.

Thirdly, we ask why are the two gases when electrically

produced so much more powerful than either gas alone.

Two theories are possible. (1) That the little proportion of oxides of nitrogen acts as a carrier between the ozone on the one hand and the flour on the other. It may be that by such means ozone is split up more easily, and the authors have certain reasons for inclining to this view. The action is then oxidation, but partially catalytic oxidation.

(2) That by passage through the spark box the ozone is rendered much more active because much more highly ionized. Fig. 12 shows how much more ionization takes place in the spark box than in the ozonizer. It was thought that this question might be settled once and for all by producing the ozone chemically instead of electrically. Pure ozone was, therefore, carefully prepared by Mr. Peachey by means of the well-known action of phosphorus, and this was found to bleach effectively; but on testing with the electroscope it was also found to be in a state of ionization, as shown in Fig. 13, so that the test was quite inconclusive. Work on this subject is still in hand, and the authors can only ask those chemists who have any experience of this kind of work to express their opinions of the possibility of each of the theories above suggested.

We may, however, collect, for the benefit of the commercial chemist, the results which have been obtained in this investigation and which will be of use to all those who are seeking to solve the problem of the electrical fixation of nitrogen or the economic production of ozone.

The conclusions to which our experiments point are:

(*a*) In order to obtain good yields of oxides of nitrogen from the air, it is not essential that a very high temperature should be used. Very thin electric discharges existing between many pairs of points (the latter being as far apart as insulation will allow and supplied by alternating current of high frequency) will give, probably, the most economical results. By this means long flames of burning nitrogen may be produced which do not appear to have a temperature approaching that of the arc, but yield, nevertheless, as much oxides of nitrogen with a smaller expenditure of power. Good results are to be looked for, not by increasing the current supplying those points so much as by increasing the number of pairs of points used.

The use of a current of air to extend these discharges is certainly to be advocated, but the limit of the velocity of this air current is determined by the oscillations set up in the circuit thereby; for above a certain point no commercial insulator will stand the effect of these latter.

The points used should be conical in shape, of such proportions that they do not readily burn away. Steel is, perhaps, the best material.

(*b*) In order to obtain good yields of ozone electric discharges should be allowed to take place between edges or points, these discharges being controlled by a high steadying resistance or impedance, so that they cannot possibly develop into that form of discharge which is so concentrated that it may be blown into a flame. An alternating current of high frequency should be used, and to avoid the production of oxides of nitrogen, the air supplied should be dry, clean and cold, and every precaution should be taken to keep the whole apparatus cool, as, with high temperature, ozone is immediately split up. The current may be increased so long as no yellow discharges appear, and no sensible heating of the apparatus takes place. In order to make the apparatus as economical as possible, the air gap used should be long, for then the loss in the steadying resistance is proportionately less, but the over-all pressure is proportionately higher. The authors consider that the observance of the above conditions will materially tend to economy in the commercial application of the electric discharge.

Biological Tests.—A number of tests have been made to prove the sterilizing effect of the gas, and the following are the chief results, as given by Dr. E. M. Blumenthal. The quantity of micro organisms in 1 gramme was:

Bleached Kyo Meal	*Unbleached Kyo Meal*	*Bleached Wheat Flour*	*Unbleached Wheat Flour*
1,000.	2,000	17	54

The flour is thus seen to be partially sterilized, and this fact is found to increase very materially the value of the process, since it improves so much the keeping properties of the material treated.

German Practice in Cleaning Blast-Furnace Gases and Coke-Oven Gases.

The fact that American engineers formerly did not pay sufficient attention to the cleaning of blast-furnace gases before delivering them to the gas engines, has undoubtedly been a source of trouble and one of the factors retarding in this country the industrial development of power production from the waste gases of blast furnaces. But it is now generally recognized that purification and simultaneous cooling is an indispensable condition, while the degree of purification required depends on the nature of the ore and also to a certain extent on the type of gas engine used.

In an editorial note on page 291 of our August issue we called attention to the great industrial change which is now taking place in this country in electric power development from blast-furnace gases, mainly due to the activity of the United States Steel Corporation. At this moment it, therefore, seems apropos to review the methods of gas cleaning employed in Germany, as the classical country of this industrial development.

An excellent review of the situation has been given by Mr. K. Reinhardt, of Dortmund, in his Iron and Steel Institute paper, which we have already abstracted on page 355 of our last issue. While this paper covered the whole field of the application of large gas engines in German iron and steel works, we give here that part only which deals with the purification of blast-furnace gases and coke-oven gases.

Blast Furnace Gases.

From what has been learned up to the present time it is clear that a thorough purification and drying of the gas is undoubtedly the principal factor in assuring a continuous and undisturbed working of gas engines. German gas engine manufacturers have from the very first considered the cleaning of the gas an essential condition, while, on the other hand, the Cockerill Co. considered it unnecessary.

As a matter of fact, at many places Cockerill engines were working satisfactorily without any cleaning whatever, while at other works this practice resulted in very disagreeable experiences; in one case, owing to the excessive wear of the working surfaces of the cylinder, and another time owing to premature ignition caused by the formation of a crust, chiefly on the piston ends, favored probably by excessive lubrication.

If at one works the engines gave satisfaction without the gas being cleaned, and at another works similar engines were unsatisfactory, this only proves that the gas, if not cleaned, contains different percentages of dust at various works as it issues from the furnace throat, and that it may become partially cleaned in the gas main. It proves further that the same quantity of dust does not everywhere have the same effect, since it may be composed in some works of very splintery coke which do not so quickly cause excessive wear of the working surfaces.

The design of the older Cockerill engines was [illegible] as regards the inlet-valves, not very sensitive to the effect of dust, the units most frequently constructed being those for a one single-acting cylinder, and in consequence the contents of the gas passages before the valve and of the valve itself were of rather large dimensions, and parts [illegible] to be worked by the dust were not present with the [illegible] of governing then employed.

The methods of governing and of mixing the gases in newer constructions, in which more stringent specifications for smaller variations of speed are laid down, are much more sensitive to the presence of dust, owing to their being combined with springs as delicate as possible, in order to keep the resistance of the governor and back pressure upon it as low as possible.

If the spindles or regulating slide valves are covered with a coating of dust, for instance, the springs are no longer sufficiently powerful to move these parts at all, or at the right moment, irregular working results and disturbances in working result. This also occurs if dust is deposited on the valves or slides, the positions of which are regulated by the governor according to the load on the engine. The valves and throttle-valves (manipulated by hand) of the gas main leading to the engine are also very sensitive to dust. The dust deposits on them very readily, and renders them difficult to move, and the areas at these places are for the time being unduly restricted, so that the engine does not receive sufficient gas to maintain its normal power. In all the above cases, in addition to the percentage of dust, the percentage of water contained in the gas when admitted to the engine also exercises an injurious effect.

It is easily understood that moist dust adheres with greater facility to the surfaces with which it comes in contact than dry dust, the greater part of which passes through the engine without being deposited.

Great trouble is experienced with moist and dusty gas when the engine does not run continuously, but stops working on Sundays, for instance. It may then happen that the deposit of wet dust, which, while the engine is continuously working, does not offer very great resistance to the motion of the valve-gear, dries to a hard crust while the engine is not running, and causes these moving parts to become jammed, rendering the starting of the engine impossible.

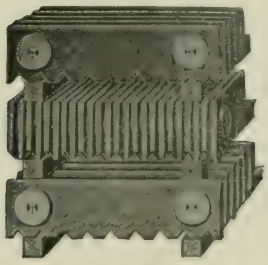

FIG. 1.—ZSCHOCKE SCRUBBER.

The circumstances mentioned above are the result of the gas not being sufficiently purified or dried, as well as of the greater consumption of oil necessitated, and the consequent increase of dirt inside the engine, and, as a matter of fact, are the cause of most of the troubles experienced in working. For this reason in all new plants great importance is attached to the effective cleaning of the gas.

Before the introduction of blast-furnace gas engines the washing of blast-furnace gas was considered necessary or advantageous for blast heating and steam raising purposes; the presence of dust diminished the efficiency of the combustion and of the heat transmission, and rendered frequent cleaning of the hot-blast stoves necessary, although cleaning the gas for the above purposes did not have to be carried out to the degree necessary for the working of engines.

The whole of the gas from the blast furnace is now subjected to a certain amount of washing, determined by experience, while the gas destined for utilization in the engines undergoes still further purification.

For a standard type of purifying plant for blast-furnace gas the following may be observed:

The gases on leaving the blast furnace are led through a series of so-called dry purifiers, and hence through long pipe lines into the coolers or scrubbers, and from these into the so-called centrifugal purifiers (Theisen apparatus or fans with water spray). After leaving the above plant the purification of the gas should be complete, so that before being admitted into the engine the gas has only to be dried in filters or in capacious tanks.

In several plants, by drying and by passing through a long main to the engine, a further noteworthy purification of the gas takes place.

With regard to the construction and manner of working of the various apparatus, the following remarks may be made:

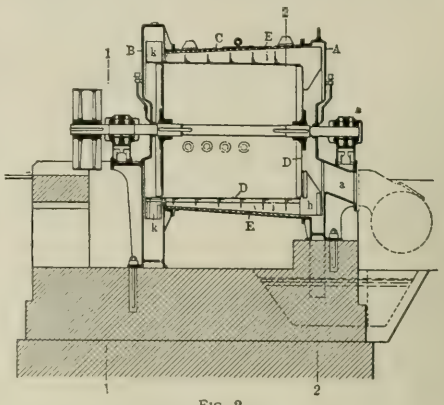

FIG. 2.

THEISEN APPARATUS.

The dry purifiers consist generally of a combination of cylindrical vessels, in which the gas is led downwards with a rapid motion and upwards with a slow motion. During this movement, and especially during the change of direction of the stream of gas, the coarsest particles of dust are separated. The pipes leading from the above should be made as long as possible with as large a section and as many sudden changes of direction as possible, in order that the gas may be further freed from coarse particles of dust.

The coolers or scrubbers are vessels in which the gas flows from the bottom to the top, and the water from the top to the bottom. The water must be finely sprayed in order to moisten the dust, and thereby increase its weight and cause it to settle to the bottom. At the same time the gas is cooled in the

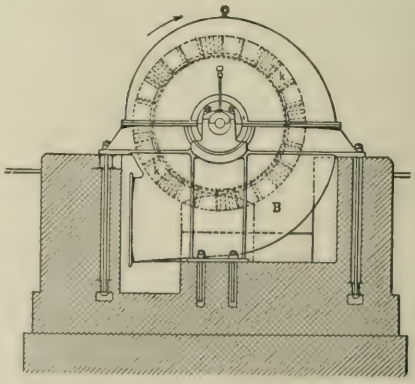

FIG. 3.

THEISEN APPARATUS

scrubbers, in which the water vapors are condensed, and the dust is deposited.

The vessels are either empty, in which case the water is finely divided by spraying nozzles, or the interior is arranged

with sieves, wire netting, coke or wooden trays, as in the Zschocke scrubber (Fig. 1).

The interior of the Zschocke scrubber consists of a series of wooden trays, one above the other, intended to reduce the velocity of the falling water, and by reason of their special

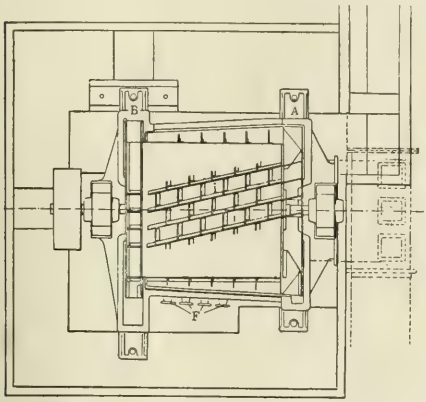

Fig. 4.
THEISEN APPARATUS.

form to divide the water into fine streams, so that the large surface exposed may effect a satisfactory cooling of the gas. The precipitated dust is removed at the bottom of the scrubber.

In centrifugal purifiers the further separation of the dust is effected by centrifugal action on the wet dust. The application of this apparatus renders it possible to attain a satisfactory purification of blast-furnace gas.

The first centrifugal purifier in Germany was the Theisen apparatus, patented by Mr. Theisen, of Munich. In Duedelingen it was subsequently discovered accidentally that an ordinary fan, with water injection, was also very efficacious for gas cleaning.

The Theisen apparatus (Figs. 2, 3, 4, 5), according to a description given by one of the makers, the Dingler Maschinenfabrik, consists essentially of the following parts: (1) The suction chamber A; (2) the pressure chamber B; (3) the middle chamber C; (4) the drum with shaft and bearings D; (5) the grating E.

Water enters at F tangentially to the casing of the middle chamber C, and leaves the apparatus through the pipe G.

The manner of working this apparatus may be described as follows: After the gas has been cooled and charged with water vapor, it is drawn in by the vanes *h*, and the coarse dust is separated in the suction chamber. Through the action of the fans at both ends of the drum D the gas is then drawn through the space between the drum and the casing. The outer circumference of the drum (Figs. 2 to 5) is provided with a number of inclined spiral vanes *i*, so that the gas has also to travel a long way in the form of a spiral. Hence, by injecting water at the same time through the pipes F a high degree of purification of the gas takes place, and the accompanying water vapor is simultaneously condensed. The dust is then projected against the spiral meshes of the coarse grating E fixed to the interior surface of the casing. By centrifugal action the water entering in a tangential direction is at the same time distributed over the surface of the grating, which prevents it from becoming clogged and incrusted with the separated dust. In addition to this, the surface of the water is broken up, thus favoring cooling and condensation. This washing of the gas absorbs carbonic acid and sulphurous gases.

The purified gas then enters the discharge chamber B, from which the water is thrown out by the vanes *k*, and the gas is forced to the engines with a pressure of from 50 to 100 millimeters of water. The washer reduces the dust from 3 or 4 grammes per cubic meter of gas to 0.02 or 0.03 grammes, with a consumption of 0.8 to 1.5 liters of water per cubic meter. The washer is generally driven by a direct-coupled electromotor, and the smaller sizes by a belting, at a speed of from 300 to 450 r. p. m. The sizes generally used [illegible] 6,000 to 33,000 cubic meters per hour, and the power required from 50 to 150 effective horse-power.

Theisen imputes the useful action, during purification, in his washer to the steam present in the blast-furnace gas and to that formed by contact with the injected water, and on this account recommends his apparatus to be placed, not behind the scrubber, but without such apparatus by introducing simple gas premoisteners immediately behind the dry purifier, in order that the gas may be as hot as possible at the entrance into the apparatus. On the other hand, Prof. Osann, in an exhaustive investigation of the purification of blast-furnace gases, chiefly by the action of cooling surfaces for the water vapors and the deposition of dust, considers it preferable to [illegible] the gases previous to their being introduced into the Theisen washer, so that the latter has only to remove the finer particles of dust which are otherwise difficult to separate. He hopes by this arrangement to effect a saving of power.

The fans employed for the purification of gases, as constructed by R. W. Dinnendahl, at Steele (Fig. 6), only differ from ordinary air fans in the construction of the vanes and bearings, which are of a much heavier construction, to cope with the injection of water and the higher temperature of gas. They are provided with a water inlet at the suction opening, and with an arrangement, as in disintegrators, for pulverizing the water, so that a sort of water curtain is formed through which the dust has to pass. The cohering particles of dust and water are separated by centrifugal action through which these particles are thrown against the inner circumference of the fan casing. The under portion of the fan casing opens into a tank A, from which the separated slimes flow away and the purified gas escapes at the top outlet. The method of purifica-

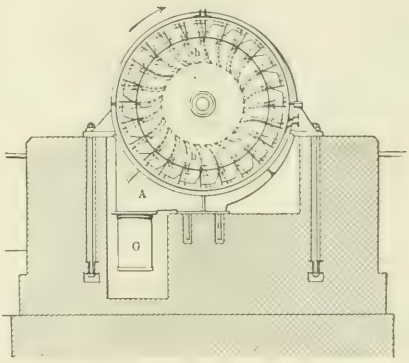

Fig. 5.
THEISEN APPARATUS.

tion resembles that of the Theisen apparatus, except that in the former the passage for the opposing action of the gas and water is not so long.

The usual sizes of gas cleaning fans, according to Dinnendahl, are from 15,000 to 70,000 cubic meters of gas per hour, requiring from 40 to 110 hp. The circumferential velocity of the impellers is up to 56 meters per second, with a diameter of from 1.1 to 1.75 meters. For 1 cubic meter of gas from 1½ to 2 liters of water are required, and the dust is reduced from 3 grammes to 0.2 grammes; as a rule, the percentage of dust is reduced to one tenth of the percentage before washing.

When two or more fans are arranged parallel to one an-

other for the purification of large quantities of gas, it is often difficult to obtain outputs equal in quantity and quality. It is, therefore, advisable to provide regulating dampers behind the fans, and above all, to make the mains, both before and after the branches to the fans, of large diameter, so that they can at the same time act as air vessels. As a certain preventive of the above difficulties, which are often of a very annoying character, the author can only offer one suggestion, namely, to drive the fans and the electromotors alike with the same speed in such a manner that their axes could be connected with friction couplings, so that the fans produce equal differences of pressure.

Of the other purifying apparatus employed only the Bian cooler may be mentioned. This consists of a horizontal shaft turning within a cylindrical casing and carrying a number of discs of wire netting. The lower halves of the discs dip into water, and the gas passes through the meshes of the upper halves. The purification of the gas is continued in centrifugal apparatus until the desired degree of cleanliness is attained, after which it has only to be dried. This is effected by forcing the gas through a series of layers of wooden fiber or wool in large cylindrical casings, to which it yields its water. Naturally the resistance caused in passing through the layers of wool requires a large expenditure of power, and the renewal of the wet wool, together with the cost of attendance, necessitates

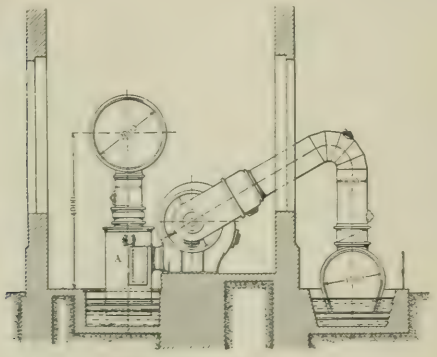

FIG. 6.—DINNENDAHL FAN.

the installation of a spare drier. Large vessels containing various materials through which the stream of gas is forced with frequent changes of direction, are employed for the separation of the water, and these vessels are further aided by long pipes with frequent changes of direction. If a large gas holder is erected between the cleaning plant and the engines, in addition to its quality as a pressure regulator, it does excellent service in the separation of water, and renders the previous drying of the gas and the expenditure for attendance on the plant and the power superfluous.

It must here be mentioned that in several iron works it was not found possible to reduce the percentage of moisture in the gas arriving at the engine to the point of saturation at the corresponding temperature of gas. In such cases, after the supply of water to the scrubbers had been cut off, so that they were only employed as dry coolers and purifiers, the gas has not so perfectly cleaned, but was drier and worked with less harmful results in the gas engines than before.

Coke Oven Gases.

A few remarks concerning the purification of coke-oven gas for motive power in gas engines may usefully be added.

The gas at disposal for this purpose has already been so far purified by the recovery of by-products that, as a rule, only the remains of tar, and also sulphur and cyanides, have to be removed. The tar residues are removed in so-called tar separators, which consist of high cylinders of boiler plate in which a number of platforms or ledges are arranged alternately to the left and to the right, so that the gases pass through in a zigzag direction and the tar is deposited on the ledges. Other apparatus work in a similar manner, the main stream of gas being divided into a large number of smaller streams and by the resulting sudden alterations of direction, and also by impinging on the plate walls, the gas is freed from tar (Pelouze apparatus). Further, rotary cleaners are in use, which serve for the separation of ammonia, naphthaline, cyanide and sulphuretted hydrogen, and according to the form of the rotating surface are arranged as hurdle, brush or ball washers (patented by Zschocke[1]). The Theisen washer can also serve this purpose; but, as far as the author is aware, it has not as yet been so employed. The inventor hoped to obtain good results, especially in the separation of tar.

The separation of sulphur and cyanide is, according to Prof. Baum, best obtained by filters. The filtering material employed consists of Laming composition, a mixture of bog-iron ore and wood shavings. The composition in layers of 6 to 8 inches deep is carried by plates or gratings; the gas passes through two to four such layers, one after the other, and the iron combines with the sulphur to form iron sulphide, and with the cyanide to make iron cyanides (Prussian blue). The composition is from time to time taken out of the filter and exposed to the air, by which means the sulphur is oxidized and the composition regenerated and ready to be used again.

In passing through the filter not only the sulphur, but also the tarry liquors, water and heavy oils remain behind. For this reason plants which do not require the removal of sulphur often employ filtering apparatus, the Laming composition being replaced by sawdust or wood fiber. Gasometers, which are frequently placed as near as possible to the engines, and as in the case of blast-furnace gas, at the same time regulate the pressure, also serve to dry the gas.

Results Obtained in German Plants.

With reference to the purification and its influence, the following may be seen from the answers received upon a systematic circular inquiry of the Society of German Ironmasters. All smelting works have centrifugal apparatus in use for removing the fine dust, and, indeed, about half of them have scrubbers or Bian coolers and fans, and the rest scrubbers with Thiesen apparatus, Thiesen apparatus alone or fans alone. The respective merits of the various apparatus or processes cannot well be ascertained from the information received from the iron works, as it is not easy to reduce the results to a common basis. The following results, nevertheless, are perhaps of interest:

The power expended in cleaning 1,000 cubic meters of gas per hour varies mostly between 6 and 13 effective horse-power. Accordingly the power expended, in cleaning varies from 1.8 to 4 per cent of the power obtained by the purified gas.

The amount of water used for cleaning varies greatly. It requires on an average from 3 to 8 liters per cubic meter of gas, and is naturally dependent on the temperature of the water. Generally speaking, the water used with centrifugal apparatus alone is less than when it is employed in combination with scrubbers. Similarly the cost of cleaning varies considerably, and includes interest and depreciation of the purifying plant 0.03 to 0.06 pfennig, or 0.0075 to 0.015 cent per cubic meter.

The percentage of dust in the gas after the dry purification, is on an average 4 to 6 grams per cubic meter. In a few cases, however, it is only 1 to 1.5 grams. In most instances the gas for working the engines is reduced to a percentage of 0.015 to 0.03 grams of dust per cubic meter, in a few works even to 0.005 to 0.001 grams per cubic meter.

All these remarks concerning the percentage of dust are to be judged from the point of view that the determination of the

[1] Baum, Glückauf, 1901, No. 17, page 457.

same at one and the same iron works if not absolutely correct, will always be proportionately exact; but that this latter will perhaps not always be the case of tests carried out by different iron works. It would, therefore, be of importance to adopt a standard method for the determination of the percentages of dust and water, so that all results could be exactly compared.

If the purification effected by the Theisen apparatus is compared with that by fans, it will be found that, according to the manufacturers, the Theisen apparatus cleans in the proportion of 140 : 1. Thus for 1,000 cubic meters of gas cleaned per hour there is required 5 effective horse-power, and per cubic meter 1.15 liters of water on an average.

With a fan the cleaning is on an average 10 : 1, the power required being 2.2 hp. and the water used 1.75 liter.

In order to obtain a similar result, two to three fans would have to be placed one behind the other, which would require perhaps 5 to 6 hp. per 1,000 cubic meters gas per hour, and a consumption of about 4 liters of water per cubic meter of gas.

From the information supplied by the iron works only the total result can in most cases be reviewed; however, in a few cases the result of the cleaning by each apparatus is given, and from this the author concludes that a single Theisen apparatus cleans better than a single fan, since with the former the proportion of cleaning is between 90 : 1 and 25 : 1, with about 6.5 effective horse-power per 1,000 cubic meters gas, and with a fan the proportion is about 12 : 1 and the average effective horse-power 2.3. From two fans, one placed behind the other, a proportion of cleaning from 50 : 1 to 200 : 1 and power employed from 6.5 to 10 effective horse-power per 1,000 cubic meters per hour has been attained. Without taking the consumption of water into consideration one Theisen apparatus is approximately equal to two fans.

With one exception, all iron works possess apparatus for drying the gas as described above.

In no case does the gas contain any suspended water—that is, no water above the quantity at the point of saturation at the corresponding temperature.

This temperature is in most cases the same as the temperature of the air, or only a few degrees higher. In a few cases the percentage of water is even lower than that corresponding to the point of saturation at the temperature of the gas, but this is only possible when the water used for cooling is at a very low temperature, and the gas is cooled to below the temperature of the gas arriving at the end of the gas main.

A further cooling of the gas would be of great utility, favoring the separation of water and purification, and thereby assuring the continual working of the gas engines without disturbance.

The particulars which the author received from the collieries are not so complete as those received from iron works, owing to the collieries not yet having so much experience.

Of fifteen collieries which were questioned, two had no special plant for the purification of gas, but only plant for the recovery of the by-products; four collieries have plant for the separation of sulphur and tar; six a similar plant for sulphur only, and three a plant for tar only. The power expended is only that necessary to overcome the resistance of the gas passing through the purifier, which is on an average about [illegible] per cent of the power developed. The other working expenses consist only of the renewals of the filtering material, which amounts on an average to about 0.03 pfennig (0.0075 cent) per cubic meter, whilst the expenses of the purification plant itself increases greatly with the sulphur in the gas.

Only traces of tar have to be removed by the purifier, but it is much more important to remove the sulphur, which attacks the cylinders, piston rings, piston rods, and stuffing boxes.

In one case it is stated that the percentage of sulphur was reduced from 5 grams to 0.7 grams per cubic meter.

The heating value of coke-oven gas varies from 2,500 to 4,600 calories per cubic meter.

[illegible] gas available for gas engines [illegible] approximately, ranging from [illegible] per cent according to the quality of the coal used and [illegible] according to the [illegible] coke-oven.

[illegible] replies to the questions published by the author, a certain doubt be called to the fact that there are [illegible] gas holders between the [illegible] engines. The capacity of the holders in proportion to the gas consumption varies considerably. One [illegible] plant is a gas holder of smaller size, arranged as an equalizer of pressure, before each engine.

The pressure of the gas at the engines is on an average from 2 to 4 inches, but in many plants it is 8 inches and over. The variations in the gas pressure naturally depend on the number of gas engines at work and of furnaces in blast, and on whether the blast-furnace tops are provided with a double seal or not. As a rule, it is recommended that the gas pressure be maintained as regularly as possible, and not much above the pressure of the atmosphere (about equal to from 1¼ to 2½ inches water). This can, of course, only be done by using a gas holder, which, besides being an excellent separator for water, possesses the advantage of preventing a reduction of speed or even the stopping of the gas engines when the supply of gas is suddenly interrupted for a short period, as may happen when only a small number of blast furnaces are at work. Long gas mains of large section also serve as a reserve, although not so effectively, and for a short period tend to equalize the pressure.

The intervals at which the engine or its several parts have to be cleaned vary greatly. From information received from iron works, it may be concluded that with gas well cleaned (0.015 to 0.3 grams of dust per cubic meter), and at the same time well cooled and dried, the inlet gear—that is, the parts before the cylinder of the engines—must be cleaned at intervals of two to three months, and a complete internal cleaning must be undertaken every six or eight months.

In a few plants using gas which is specially clean, the engines require less frequent cleaning. In others the inlet-gear, throttle valves and other similar parts require cleaning at periods of fourteen days. At the same time, when the lubrication is not excessive, and even when the gas is not well cleaned, an internal cleaning of the engine every two or three months is sufficient.

The parts before the cylinder require for cleaning on an average from 6 to 20 hours, according to the size and build of the engine and the number of men employed, and the internal cleaning requires from two to eight days.

The quantity of water used for cooling the cylinders and pistons averages 8.8 to 11 gallons per hour and per effective horse-power, of which 2.2 to 2.6 gallons are for the pistons. The consumption of oil in most plants is reckoned at 1.0 to 1.25 grammes per hour per effective horse-power. The consumption of gas has not yet been sufficiently tested to compare the various systems.

According to trials made at iron works, the heat employed by the engines varies from 2,500 to 3,300 calories per hour and per effective horse-power. Most iron works are at present not yet in a position to determine the consumption of gas in their engines, and content themselves with testing the exhaust gases and thereby determining the completeness of the combustion in the motor.

From the answers received from the collieries, engines using coke-oven gas require cleaning after similar periods to those using blast-furnace gas.

Generally speaking, however, at present the collieries have not sufficient experience to answer this question decisively, and not freely. The traces of tar in coke-oven gas which is difficult to remove, and to have, up till now, necessitated more frequent internal cleaning, and above all, the sticking rings, stuffing boxes, oil holes, and that similar parts, to [illegible] greater attention.

Electro-Decolorization—A Study in Optical Sugar Analysis.

By F. G. Wiechmann, Ph. D.

The Problem: Saccharimeters are designed to return a reading of 100° Ventzke, representing 100 per cent of sucrose when the chosen normal weight of chemically pure sucrose is dissolved in the chosen volume of water.

The ideal clarifying reagent would be one which would remove from a normal aqueous solution of raw sugar everything except the sucrose and the water, and which in so doing would cause no change whatever in the original optical rotatory power of the sucrose.

It is well known and generally admitted that the basic acetate of lead solution as at present used—with the sanction of the International Commission for Uniform Methods of Sugar Analysis—for the clarification of sugar solutions, is not above criticism.

In the first place, the use of this reagent causes an error by its mere presence: the precipitate it forms in clarifying the sugar solution occupies a part of the space which should be taken up by the solution, and in consequence, that solution is unavoidably concentrated.

In the second place, this reagent forms chemical combinations with at least some of the optically active bodies which are frequently present in raw sugars without, however, necessarily removing such bodies from the solution. It is a well known fact that such chemical combinations formed by this reagent with these optically active bodies, sugars and non-sugars, possess optical properties different from those originally exhibited by the bodies, and that it is, therefore, practically impossible to make a proper allowance for their optical influence.

Seeking for a method by which the difficulties indicated might be at least partially avoided or overcome, attention was naturally turned to electrical energy as the agent capable of bringing about chemical changes without necessarily introducing material reagents into the solution under examination. Making this consideration the point of departure, several series of experiments were planned to determine what degree of decolorization might be secured in sugar solutions by passing electric currents through them respectively from insoluble and from soluble electrodes.

Four experimental series were thus made:

Series A—Electrodes, platinum; solvent, water.

Series B—Electrodes, platinum; solvent, water and hydrogen peroxide solution.

Series C—Electrodes, lead; solvent, water.

Series D—Electrodes, anode, lead-peroxide; cathode, lead; solvent, water.

In all instances the work was done on *normal* sugar solutions, that is to say, on sugar solutions made up to the proper degree of concentration *before* decolorization, and in each of these methods, therefore, if feasible, the precipitate error would be entirely obviated.

Source of Current: The electrical energy used in these experiments was secured either from a motor generator, or from the direct lighting current available in the laboratory.

The motor generator was a low-voltage (electrolytic) generator with a range of from 3 to 15 volts, run by a motor on a 110-volt direct current.

When the lighting current was used directly the current density was regulated by means of a bank of electric lamps made by arranging a number of lamp sockets partly in parallel, partly in series, upon a slab of slate, and inserting the necessary number of lamps properly grouped to give the desired result.

Current Measurement was effected by means of a Weston voltmeter and a Weston milli-volt meter, used as an amperemeter.

The voltage was measured directly across the electrodes, the ammeter was inserted in the main circuit.

Electrodes: As previously stated, both insoluble and soluble electrodes were used in the experiments. Carbon and platinum were tried as representative of the first group; aluminium, lead and lead-peroxide were used in the tests with soluble electrodes.

Various forms of electrodes were experimented with: foils, flags, annular rings and bands, open and closed cylinders, wave plates and wires.

In some experiments both electrodes were stationary, in others, one of the electrodes was kept in motion. In short, a great many preliminary experiments were made in order to ascertain the current normal density N D_{100}; *i. e.*, the current strength for 100 square cm. of electrode surface, best suited for the work to be done.

At first a number of tests were carried out in which both electrodes were placed in one compartment; later it was determined to be distinctly preferable to work with the two electrodes in separate compartments.

This naturally brought the problem of septa up for consideration.

Septa: A number of experiments were made with septa of clay and of porous earthenware, but finally parchment was selected as best adapted to the needs of the situation.

Experience taught that considerable caution has to be exercised in the selection of the parchment for the purpose. Some of the parchment first experimented with diffused an optically active, copper-reducing body into the solution, and, of course, vitiated the experiment.

It was soon learned that all of such parchment-sugar can be entirely removed by repeated boiling with distilled water, and all of the experiments to be described hereafter were made with parchment thus treated and subsequently dried and ironed to perfect smoothness. Each batch of parchment so prepared for experimental work was, of course, submitted to preliminary examination, both optical and with Fehling's solution, to ensure the removal of every trace of the parchment-sugar.

Electro-Cells: Two forms of cell were adopted; one, cylindrical in form, consists of a cell frame of hard rubber, designed to support a cylinder of parchment. This cell is placed within a cell or beaker of glass. The one electrode, a cylinder with closed ends, is placed within the parchment cell; the other electrode, an annular band, is fitted tightly against the wall of the glass cell.

The whole apparatus can be enclosed in a water jacket, through which a current of water can be kept coursing during the experiment, should an absolutely uniform temperature of the cell contents be maintained.

The distance between each electrode and the parchment is 5 mm. A platinum wire from each electrode is connected with the copper leads which supplied the current. Experiments were made with this circular cell in which the N D_{100} values ranged from 1.11 to 4.02.

The square cell is also made of hard rubber. It consists of two plates, each 125 mm. square, and rimmed on three sides with raised edges 12 mm. thick. The receiving chambers, which have been hollowed out of the rubber plates to a depth of 5 mm., are 100 mm. x 110 mm., and are each provided at the base with a discharge opening.

To use the apparatus the parchment septum is placed between these two rubber plates, these are tightly locked together by four pegs placed at the corners, and the whole is then firmly clamped in a wooden frame; after the two electrodes have been slipped into place on either side of the parchment the apparatus is ready for use.

The distance from each electrode to the septum is 5 mm. The electrodes have each an active surface of 100 square cm.,

* Abstract of an address given before the International Commission for Uniform Methods of Sugar Analysis at Berne, Switzerland, Aug. 3, 1906.

and the amperes employed, therefore, at once express normal current density values.

Optical Determinations: All of the polarimetric observations were made by means of a double-wedge compensation saccharimeter built by J. Peters, of Berlin, examined and certified to as correct by Prof. Dr. Herzfeld, of the Institute of Sugar Industry at Berlin.

The correct adjustment of this instrument was controlled before each experiment by means of a standard quartz plate and by observations of the zero point. Furthermore, the observation-tube which was used in each instance was examined as to any possible optical activity of its cover glasses before being filled with the solution to be examined. The light was in every instance obtained from a 50-candle power, 110-volt, spiral-shaped filament, and the light so obtained was passed through 3 centimeters of a 6 per cent solution of potassium dichromate; the temperature of the saccharimeter was held as close to 20° C. as possible. Naturally, only carefully standardized metric flasks and weights were used. The solution in each case was made up by dissolving 26 grams of the sugar to a volume of 100 metric C.C.

Tintometer Observations: The amount of decolorization secured was determined by means of a Lovibond tintometer. A description of Lovibond's method will be found in the *Journal of the Society of Chemical Industry*, Vol. XIII., No. 4, April 30, 1894.

The method adopted to determine the amount of color removed consisted in matching the color of both the original solution and of the treated solution against the Lovibond glasses, subtracting the value representing the treated solution from that representing the untreated solution, and calculating the percentage of color removed.

Record of Experiments: As an illustration of the manner in which the experimental work was done the record of a single experiment is here given:

Number of solutions prepared = two, A and B.
Sugar used, first beet sugar.
Solvent used, water.
26.0 grams up to 100 metric C.C.
Solution A, clarified with 2 C.C. basic lead acetate solution. Int. Com. Standard.
Room temperature = 20° C.
Solution temperature = 20° C.
Saccharimeter temperature = 20° C.
Saccharimeter in adjustment at zero point and at 98.6°.
Polarization readings in 200-mm. tube.

Observers:	*A*	*B*
	94.7	94.8
	94.7	94.7
	94.7	94.6
	284.1	284.1
	94.7	94.7

Average polarization = 94.70.

Solution B—No preliminary clarification; solution placed in electro-cell; current passed for 5 minutes.

Electrodes:

Anode = lead 10 x 10 c.m.
Cathode = lead 10 x 10 c.m.

Temperature data:

Original solution, before passing electric current, 20.0° C.
Anode solution, after passing electric current, 20.0° C.
Cathode solution, after passing electric current, 20.0° C.

Reaction:

Original solution—acid to litmus.
Anode solution—acid to litmus.
Cathode solution—{alkaline to litmus. / alkaline to phenol-phthalein.

Distance between electrode and parchment in each chamber = 5 mm.

Electric Data.	*Ampere.*	Volts.
Start.	0.25	4.0
After 30 seconds	0.26	4.6
" 60 "	0.27	4.3
90 "	0.26	4.6
120 "	0.24	4.2
150 "	0.24	4.6
180 "	0.23	4.75
210 "	0.21	4.65
240 "	0.20	4.75
270 "	0.20	4.75
300 "	0.20	4.80
Average N D_{100} =	0.23	4.45

Decolorization:

	Per Cent of Color Removed.
Original solution = Glass 24	....
Anode solution = " 3	87.50
Cathode solution = " 18	25.00

Polarization readings of anode solution in 200-mm. tube.

Saccharimeter in adjustment at zero and at 98.6°.
200-mm. tube, empty, optically inactive.
Room temperature 20.0° C.
Solution temperature 20.0° C.
Saccharimeter temperature 20.0° C.

Observers:	*A*	*B*
	94.60	94.80
	94.90	94.85
	94.80	94.80
	284.30	284.45
	94.77	94.82
	94.77	
	94.82	

Average polarization = 189.59 ÷ 2 = 94.79.

Polarization of solution, clarified with basic acetate of lead	94.70
Polarization of anode solution after electric treatment.	94.79
Difference =	0.09

Experimental Data: As previously stated four experimental series were carried on, A, B, C and D.

The results obtained in series A and B were not satisfactory.

Series C: In Series C both electrodes were made of sheet lead and no hydrogen peroxide or any other reagent was used. The following data were obtained:

Experiment No.	*Sugar Used.*	*N. D_{100}*	*Anode Solution: Percentage of Color Removed*	*Polarization: Determined in*
0	Sand. Island I.....	.25	86.84	200 mm.
28	" "	.25	94.85	200 "
29	" "	.16	93.10	200 "
30	" "	.13	87.85	200 "
1	" " II.....	.25	94.12	200 "
25	Matanzas Centrifugal..	.25	93.58	200 "
26	Porto Rico Muscovado	.25	85.20	200 "
24	Java Centrifugal.....	.17	96.78	200 "
14	Beet I.....	.25	87.50	200 "
12	"	.25	93.44	200 "
27	"	.25	91.37	200 "
31	"	.33	93.54	200 "
20	"	.24	88.24	200 "
32	"	.25	85.28	200 "
21	"	.26	88.00	200 "
33	"	.23	87.80	200 "
18	" II.....	.26	92.80	200 "
80	Sand. Island II.....	.25	93.40	200 "
90	" " I.....	.13	92.30	200 "

A glance at these results shows that the decolorization obtained was very satisfactory; considering all of the twenty experiments recorded on raw sugars of various origin and grades an average decolorization of 90.56 per cent was attained.

Series D: In oxidation processes the energy with which the reaction proceeds at the anode is determined by the anode potential. Frequently the oxidation of certain substances can thus be effected by a peroxidized anode that cannot be secured by a purely metallic anode having a lower potential pressure, and it was, therefore, determined to try some experiments in which the cathode should be of lead, but in which the anode of lead should receive a coating of lead peroxide, to learn whether possibly a still more efficient decolorization might be secured.

The peroxide plate was prepared in the following manner: Three lead plates were thoroughly scoured with sand and water, then placed into a hot and fairly concentrated solution of caustic soda, and then after remaining therein for a few minutes were thoroughly rinsed with clean water.

These three plates were then hung as electrodes into a 20 per cent solution of sulphuric acid, the central plate was made the anode and the other two were linked as cathodes. An electric current, N D_{100} = 2.0 amperes, voltage = 2.5 volts, was then passed for 30 minutes. Should it be desired to coat only one side of the lead plates with the lead peroxide, then two such plates can be prepared in one operation by making those two plates the anodes, and the central plate the cathode in the above sketched arrangement.

When peroxidized the plate is washed thoroughly with water, then placed for 3 or 4 minutes in boiling water, thoroughly rinsed with alcohol and finally dried.

The data found in this set are given in the list appended:

Experiment No.	Sugar Used.	N. D_{100}	Anode Solution Percentage of Color Removed.	Polarization Determined in a Tube of:
42	Beet I.....	.25	94.00	200 mm.
43	Sand. Island I.....	.25	96.40	200 "
44	" " II.....	.22	94.44	100 "
45	" " III.....	.24	46.36	Not readable.
46	Matanzas Centrifugal..	.22	86.11	100 mm.
47	Sand. Island I.....	.24	96.00	200 "
48	Porto Rico Muscovado	.23	86.68	100 "
49	Java Centrifugal.....	.12	97.50	200 "

With few exceptions the decolorizations obtained compare favorably with the results secured in Series C, yet, the additional work necessary to prepare peroxidized electrodes appears to outweigh, or more than outweigh the slight advantage gained.

Of the four experimental series tried it would thus seem that Series C is the most simple and effective. The following table contains the polarization values found in this series, and also the polarization values obtained on the same sugars after clarification with a minimum amount of basic lead acetate solution:

Experiment No.	Sugar Used.	—Polarization by— Lead Acetate Solution.	Electro-Method.	Result by Electro-Method is +
6	Sand. Island I	97.15	97.66	+ .51
28	" "	97.29	97.60	+ .31
29	" "	97.29	97.43	+ .14
30	" "	97.29	97.58	+ .29
4	" " II.....	96.49	96.37	- .03
25	Matanzas Centrifugal	92.79	93.00	+ .21
26	Porto Rico Muscovado	90.81	91.05	+ .24
24	Java Centrifugal.....	97.77	98.00	+ .23
14	Beet I	94.23	94.40	+ .17
12	"	95.69	96.23	+ .54
27	"	95.86	96.32	+ .46
31	"	96.15	96.39	+ .24
29	"	93.72	94.15	+ .43
32	"	93.75	94.17	+ .42
21	"	94.75	95.20	+ .45
33	"	94.17	94.49	+ .32
18	" II*	91.06	91.09	+ .09

*(Current on 10 minutes.)

Attention is invited to the last column of this table, which shows the difference between the polarization values found by the two methods.

While it were idle to expect perfect agreement in duplicate analyses made by one, let alone made by two methods, yet it is peculiar that the results of the one method should in every instance be higher than the results of the other.

Careful experimental study showed that this discrepancy was not caused by kataphoresis nor by a concentration of the solution—due to evaporation or electrolytic decomposition of the solvent; furthermore, it was experimentally determined that the optical value of the sucrose and of the invert sugar were not directly affected by the electric currents used.

The question, however, whether the rotary power of some other optically active constituents in the sugar solution was altered by the process had still to be settled.

In all of the experiments made record was kept of the chemical reaction of the solutions before and after the passing of the electric current, and it was found that in every instance the anode solution—the solution used for polarization after the electric treatment—reacted acid, whatever the reaction of the original solution might have been.

Now, it is well known that some organic non-sugars which are optically active, experience a change, in some cases even an absolute reversal of their optical rotation value according to the reaction of their solutions, acid or alkaline.

Asparagine and some of its derivatives, for instance, act in this way. Asparagine, which when decomposed yields ammonia and aspartic acid, exhibits levo-rotation in aqueous solutions, an increased levo-rotation in alkaline, but a dextro-rotation in acid solutions.[1]

The high polarizations observed might hence have been caused by this substance; a probable indication of its presence was the strong ammoniacal odor noticeable in several instances in the cathode solutions during the electro-treatment, and the marked deepening of the color of these cathode solutions.

Asparagine is present in beets to the extent of possibly 0.1 to 0.2 per cent, it also occurs in cane products. To settle the question whether asparagine was the cause of the discrepancy or not, Champion and Pellet's method[2] was tried.

After electro-treatment and after a subsequent polarization of the anode solution, 10 per cent of an 8 per cent aqueous solution of acetic acid were added to the anode solution and the same re-polarized.

In case asparagine or one of its derivatives should prove to be the cause of the increased polarization, lower values were to be expected after the use of the acetic acid.

The following table embodies the results obtained:

Experiment No.	Sugar.	Polarization Determined by: Basic Lead Acetate Solution.	Electro-Method.	Electro-Method + 10% Acetic Acid Solution.
1	I Beet.....	95.70	96.58	95.03
2	Java Centrifugal.....	97.60	98.00	97.64
3	" "	97.90	98.17	97.57
4	Matanzas Centrifugal..	92.59	93.14	92.55
5	Porto Rico Muscovado	90.66	90.87	90.61
6	Sand. Island I.....	97.21	97.56	97.13
7	" " II.....	90.48	.....	90.46

From this table it appears that the differences between the results of the polarizations obtained by clarification with basic lead acetate solution and by the electro-method after the use of acetic acid, are practically identical, the difference on an average being less than 0.05° Ventzke.

The practical working of the electro method for decolorizing sugar solutions may hence be summarized as follows:

Dissolve 26,000 grams of sugar with pure water up to a

[1] A. Rumpler, Die Nichtzuckerstoffe der Rüben, 1898, pages 270, 271.

[2] Compt. Rend., 1876, Vol. 82, page 819. Zeitschrift d. Vereins f. d. Rübenzucker-Industrie, 1876, Vol. 26, page 819.

volume of 100 metric cubic centimeters, at a temperature of 20° Centigrade.

Place about 35 C.C. of this solution into each of the two chambers of the electro-cell previously described, and pass an electric current of approximately N $D_{100} = 0.25$ through the solution for 5 minutes.

Discharge the solutions separately. Filter the anode solution perfectly clear into a graduated receiver, add to this solution 10 per cent of its volume of an 8 per cent acetic acid solution, and polarize. To the reading observed on the saccharimeter add 10 per cent to compensate for the dilution caused by the addition of the acetic acid solution; the result is the polarization sought.

In these experiments lead electrodes have been used. On theoretical grounds possibly insoluble electrodes might be preferable, and additional experiments are now in progress in the writer's laboratory in which platinum electrodes and platinum electrodes coated with platinum black are used in their stead.

The amount of lead which passes into the solution in these experiments is, however, trifling, on an average only 7.14 per cent of the amount used in the clarification by the basic lead acetate solution.

In the electro-method, of course, the precipitate error is entirely obviated, as the lead enters the solution only after the same has been made up to its proper volume, and in this respect, at least, the method entirely fulfills the purpose for which it was devised.

Gaseous Fuels.

(Translated from Baron Hanns Jüptner von Jonstorff's Chemical Technology of Energies.)

By Oskar Nagel, Ph. D.

The gaseous fuels have, like the liquid fuels, the advantages of burning up without residue, of easy transportation to the place of combustion, and of convenient regulation of temperature. Furthermore, the length of the flame can be varied within certain limits, and for complete combustion a considerably smaller excess of air is required than with solid and liquid fuels. The gaseous fuels, therefore, have a higher temperature of combustion, and generate a smaller quantity of gaseous products of combustion than other fuels of same composition, whereby a better utilization of the generated heat can be accomplished. Another advantage is that in this case not only the air for combustion but also the gas can be preheated.

Such gaseous fuel occurs in nature and is then called natural gas. The average composition of natural gas of Pennsylvania is:

Methane	67%
Hydrogen	22
Nitrogen	3
Ethane	5
Ethylene	1
Carbon dioxide	0.6
Carbon monoxide	0.6

As the occurrence of natural gas is limited, similar gases are artificially produced for industrial use by the following methods:

1. Dry distillation of substances containing carbon, as coal, ignite, peat, wood, fat, etc., whereby gases of distillation (illuminating gas) are obtained. According to the raw material used the manufactured gas is called coal gas, peat gas, wood gas, fat gas, oil gas, etc.

2. Incomplete combustion of coal with insufficient amount of air, whereby generator gas, also called producer gas or air gas, is obtained.

3. Decomposition of water (steam) by glowing coal or combustion of coal by means of steam, whereby water gas is obtained.

In special cases other methods are used for producing fuel gases, as for instance:

4. Incomplete combustion of coal by simultaneous action of air and oxides, the latter thereby being reduced. This reaction takes place in iron blast furnaces and furnishes a gas of high fuel value, low in nitrogen and high in carbon monoxide, which is called blast-furnace gas. If water is used as oxide, semi-water gas or Dowson gas is obtained.

5. For getting high temperatures or high luminant power, acetylene C_2H_2 is sometimes used, which is obtained by reaction of calcium carbide and water:

$$CaC_2 + 2H_2O = Ca(OH)_2 + C_2H_2$$

We, therefore, have the following summary of methods for the

Production of Fuel Gases.

1. By dry distillation:
   - From coal, coal gas.
   - From peat, peat gas.
   - From wood, wood gas.
   - From fat, fat gas.
   - From oil residue, oil gas.
2. By incomplete combustion of coal:
   - (a) With air alone, producer gas (air gas).
   - (b) With air and oxides of metals Fe_2O_3, etc., blast-furnace gas.
   - (c) Air and steam, Dowson gas.
   - (d) Air and carbon dioxide, regenerated combustion gases.
3. By decomposing carbides with water:
   - Mainly calcium carbide, acetylene.

Leaving aside the acetylene and the blast-furnace gas, which is only of local importance, the following industrial gases have to be mainly considered:

(1) Gases of distillation, obtained by dry distillation of carbonaceous substances.

(a) Illuminating gas made in retorts. It is used for illuminating, heating and for internal combustion engines.

As an example, the composition of French illuminating gas is given below, which is identical all over France:

Weight of cubic meter = 0.523 kg.
Thermal value of 1 cubic meter = 5600 Ca.
Weight of 22.42 liters = 2 grams
Thermal value of 2 grams = 125 Cal.

Analysis in per cent by weight:
Carbon, 43.2 per cent.
Hydrogen, 21.3 per cent.
Oxygen and nitrogen, 25.5 per cent.

Analysis in per cent by volume:

51.0%	H_2
33.0	CH_4
8.8	CO
1.8	CO_2
1.0	$O_2 + N_2$
1.1	C_nH_n
3.3	absorbable CnH_{2n}
100.0	

(b) Gases of distillation, produced as by-product in the coking or charing of fuels, mainly coke oven gas.

(2) Generator gas, air gas, or producer gas, is properly the name of such gas only, which is made from carbon (charcoal or coke); i. e., from a coal free from hydrogen and oxygen and using dry air for the incomplete combustion. In practice, however, we comprise under the classification "generator gas" any gas generated in certain apparatus (gas producers) by leading air without steam through a glowing layer of fuel of sufficient height. The air never being dry, we get in practice always a mixture of generator gas and water gas, and also gases of distillation if crude, uncoked fuel is used.

(3) Water gas is used for illuminating and fuel purposes.

(4) Semi-water gas or Dowson gas is used for fuel and power purposes, and is prepared by leading a mixture of air and steam through a coal layer in a producer.

Generator Gas or Producer Gas.

If air is led at moderate speed through a layer of pure carbon (in practice charcoal or coke), incomplete combustion

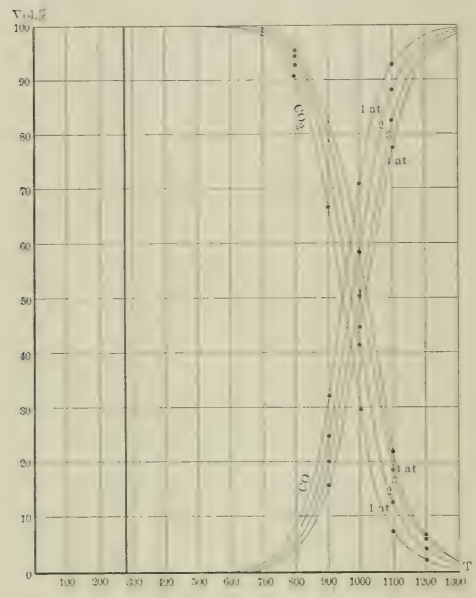

FIG. 1.—IDEAL COMPOSITION OF GENERATOR GAS FROM PURE OXYGEN.

takes place; *i. e.*, by the reaction of oxygen on the glowing coal, formation of carbon monoxide occurs:

$$C + \tfrac{1}{2}\,O_2 = CO$$

Supposing the air to contain 4 mols nitrogen to 1 mol oxygen, which is probably correct, we can write the reaction:

$$C + \tfrac{1}{2}\,O_2 + 2N_2 = CO + 2N_2$$

and we get a gas which theoretically contains 2 mols N_2 to 1 mol CO, and should have the composition:

CO 33.3% by volume
N 66.7 " "

This gas ought to yield per 22.42 liters if burned at constant volume $0.333 \times 67.9 = 22.61$ Cal. If burned at constant pressure $22.61 + 0.5 \times 0.54 = 22.88$ Cal. The thermal value of the same at constant pressure would be per cubic meter 1020.5 Cal.

The thermal value of 1 gram of gas is calculated as follows: According to the equation the gas has for every gram-matom of carbon

12 grams carbon / 28 grams carbon monoxide.
16 grams oxygen
56 grams nitrogen.
Sum: 84 grams.

As 84 grams of gas contain 3 mols (CO plus $2N_2$), 22.42 liters of the same at 0° C. and 760 mm. are equal to 28 grams, and, therefore, 1 gram of gas generates 817 Cal.

This reaction, however, only takes place at very high temperatures. At lower temperatures a second reaction occurs simultaneously, and the extent to which it occurs increases with decreasing temperature. This reaction is

$$C + O_2 = CO_2$$

or if the air is used instead of oxygen

$$C + O_2 + 4N_2 = CO_2 + 4N_2$$

Between these two reactions there exists a certain equilibrium for every temperature and pressure. If we subtract the equation

$$C + O_2 = CO_2$$

from

$$2C + O_2 = 2CO$$

we get

$$2CO = CO_2 + C$$

which reaction actually takes place at fairly high temperatures and determines the proportion of the two first reactions. It is reversible:

$$2CO \rightleftarrows CO_2 + C$$

That is, while pure CO within certain temperatures is decomposed into CO_2 and C, we find that under similar conditions CO is produced by reduction of CO_2 by means of C. Therefore, there exists necessarily an equilibrium between CO, CO_2 and C, which depends on the temperature and concentration (gas pressure).

Since out of two volumes CO only one volume CO_2 is formed, and since the reaction, according to our equation (from left to right), takes place without decrease of volume, it is clear that an increase of pressure facilitates the formation of CO_2, while a decrease of pressure favors the formation of CO. Therefore, the primary air (wind) in a gas pro-

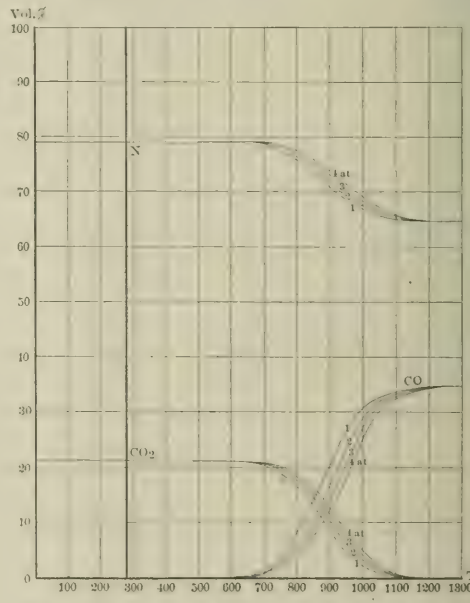

FIG. 2.—IDEAL COMPOSITION OF GENERATOR GAS FROM DRY AIR.

ducer should be of low pressure if a gas high in CO is desired.

The influence of temperature on the equilibrium is shown by the balance of the reaction heats:

$$C + O_2 = CO_2 + 94{,}310 \text{ Cal.}$$
$$2(C + O) = 2CO + 53{,}400 \text{ "}$$
$$2CO = CO_2 + C + 40{,}910 \text{ "}$$

i. e., the decomposition of 2CO into CO_2 and C takes place under generation of heat. Therefore an increase of temperature facilitates the formation, a decrease of temperature the decomposition of CO. Thence it is clear that the gas will be the richer on CO with higher temperature.

Table A.

Ideal Composition of Producer Gas (Generator Gas) Produced with Pure Oxygen.

Air Pressure. Volumetric Composition at a Temperature of		1 Atmosphere.		2 Atmosphere.		3 Atmosphere.		4 Atmosphere.	
		CO	CO_2	CO	CO_2	CO	CO_2	CO	CO_2
227° C	500° abs.	0.004	99.996	0.0028	99.9972	0.0023	99.9977	0.002	99.998
327°	600°	0.123	99.877	0.087	99.913	0.0711	99.9289	0.061	99.939
427°	700°	1.427	98.573	1.011	98.989	0.826	99.174	0.716	99.284
527°	800°	8.794	91.206	6.303	93.697	5.177	94.823	4.469	95.531
627°	900°	32.542	67.458	24.809	79.191	20.408	79.592	17.945	82.055
727°	1000°	70.35	29.65	58.105	42.259	51.788	48.212	47.017	52.983
827°	1100°	92.75	7.25	87.198	12.802	82.72	17.28	78.977	21.023
927°	1200°	98.445	1.555	97.00	3.00	95.65	4.35	94.315	5.685
1027°	1300°	99.50	0.50	99.00	1.00	98.97	1.03	98.67	1.33

All these observations are of importance for the state of equilibrium. Whether this is reached in practice or not depends on the height of the coal, porosity of same, velocity of wind, etc. It is, however, of the greatest importance for the theory of the gas producers as well as for the practice, to know the equilibrium for all the different conditions, since the only way to judge the quality of a gas producer process is to compare the results obtained in practice with those corresponding to the theoretical equilibrium.

We, therefore, give in Tables A, B and C the ideal composition of generator gas at different temperatures and pressures.

Table A gives the ideal composition of producer gas, produced with pure oxygen. Fig. 1 shows the content of this table graphically.

Table B gives the ideal composition of producer gas, produced with dry atmospheric air. The data of this table are graphically shown in Fig. 2.

Table B.

Ideal Composition of Producer Gas (Generator Gas) Produced with Dry Atmospheric Air.

Gasifying Temperature.		Partial Pressure of $CO+CO_2$ in atm.	Composition in Per Cent. by Volume.		
°C.	T° abs.		CO_2.	CO.	N_2.
		Air Pressure = 1 Atmosphere.			
227°	500°	0.21	21.00	.....	79.00
327°	600°	0.21	21.00	.....	79.00
427°	700°	0.2145	20.31	1.14	78.55
527°	800°	0.24	16.40	7.60	76.00
627°	900°	0.29	8.75	20.25	71.00
727°	1000°	0.334	2.14	31.26	66.60
827°	1100°	0.344	0.47	33.93	65.60
927°	1200°	0.346	0.14	34.46	65.40
1027°	1300°	0.3465	0.01	34.65	65.35
		Air Pressure = 2 Atmospheres.			
227°	500°	0.42	21.00	.....	79.00
327°	600°	0.42	21.00	.....	79.00
427°	700°	0.4228	20.39	1.01	78.60
527°	800°	0.466	18.14	5.82	76.70
627°	900°	0.555	11.94	17.09	72.25
727°	1000°	0.6535	4.31	29.56	67.32
827°	1100°	0.6865	0.83	33.74	65.67
927°	1200°	0.692	0.21	34.44	65.40
1027°	1300°	0.693	0.10	34.56	65.35
		Air Pressure = 3 Atmospheres.			
227°	500°	0.63	21.00	.....	79.00
327°	600°	0.63	21.00	.....	79.00
427°	700°	0.6305	20.51	0.81	78.68
527°	800°	0.686	18.14	4.76	77.00
627°	900°	0.8075	11.94	14.08	73.08
727°	1000°	0.957	4.31	27.50	68.10
827°	1100°	1.025	0.83	33.37	65.80
927°	1200°	1.0305	0.21	34.34	65.45
1027°	1300°	1.04	0.10	34.56	65.34
		Air Pressure = 4 Atmospheres.			
227°	500°	0.84	21.00	.....	79.00
327°	600°	0.84	21.00	.....	79.00
427°	700°	0.851	20.59	0.71	78.70
527°	800°	0.905	18.52	4.11	77.37
627°	900°	1.056	12.73	13.67	73.60
727°	1000°	1.258	5.00	26.46	68.55
827°	1100°	1.359	1.13	32.85	66.02
927°	1200°	1.381	0.28	34.25	65.47
1027°	1300°	1.385	0.13	34.50	65.37

Table C.

Ideal Composition of Producer Gas (Generator Gas) Produced with 50% Oxygen.

Gasifying Temperature.		Partial Pressure of $CO+CO_2$ in atm.	Composition in Per Cents. by Volume.		
t° C.	T° abs.		CO_2.	CO.	N_2.
		Air Pressure=1 Atmosphere.			
227°	500°	0.50	50.00	.....	50.00
327°	600°	0.50	50.00	.....	50.00
427°	700°	0.502	49.40	0.80	49.80
527°	800°	0.522	43.40	8.80	47.80
627°	900°	0.568	29.60	27.20	43.20
727°	1000°	0.633	10.10	53.20	36.70
827°	1100°	0.66	2.00	64.00	34.00
927°	1200°	0.663	1.10	65.20	33.70
1027°	1300°	0.6655	0.35	66.20	33.45
		Air Pressure = 2 Atmospheres.			
227°	500°	1.	49.56	.....	50.00
327°	600°	1.	45.65	.....	50.00
427°	700°	1.0035	34.03	0.61	49.83
527°	800°	1.0295	15.50	5.83	48.82
627°	900°	1.1065	34.03	21.30	44.67
727°	1000°	1.23	15.50	46.00	38.50
827°	1100°	1.308	3.80	61.60	34.60
927°	1200°	1.326	1.10	65.20	33.70
1027°	1300°	1.3305	0.43	66.10	33.47
		Air Pressure = 3 Atmospheres.			
227°	500°	1.5	50.00	.....	50.00
327°	600°	1.5	50.00	.....	50.00
427°	700°	1.5045	49.55	0.60	49.85
527°	800°	1.538	46.20	5.07	48.73
627°	900°	1.6345	36.55	17.93	45.52
727°	1000°	1.814	18.00	41.87	39.53
827°	1100°	1.9455	5.45	59.42	35.15
927°	1200°	1.986	1.40	64.80	33.80
1027°	1300°	1.9955	0.45	66.07	33.48
		Air Pressure = 4 Atmospheres.			
227°	500°	2.	50.00	.....	50.00
327°	600°	2.	50.00	.....	50.00
427°	700°	2.0053	49.00	0.54	49.80
527°	800°	2.0443	46.08	4.43	48.80
627°	900°	2.1615	37.89	16.15	45.96
727°	1000°	2.384	21.20	38.40	40.40
827°	1100°	2.588	5.00	58.80	36.20
927°	1200°	2.6435	1.74	64.35	33.91
1027°	1300°	2.6605	0.46	66.05	33.49

Since it is not improbable that in future a mixture of 50 per cent oxygen and 50 per cent nitrogen may be used in gas producers, the data for this case are given in Table C. Fig. 3 gives the results graphically.

The following important general conclusions may be drawn from these tables and diagrams:

1. In all cases the CO_2 content of the ideal generator gas at low temperature is a maximum, which is practically constant up to 400° C.

2. With increasing temperature the CO_2 content is decreasing; between 800° and 1,000° C. no CO_2 is present.

3. No CO is found up to about 400° C.

4. With increasing temperature the CO content is increasing and is reaching a maximum at 800° to 1,000° C.

5. At constant temperature the CO_2 content is increasing with the pressure, and therefore also with the oxygen content of the primary air.

6. CO shows the opposite property.

7. At low temperatures the absolute CO_2 content is increasing with the oxygen content of the primary air.

8. At high temperatures the absolute content of the gas on CO is increasing with the oxygen content of the primary air.

Therefore, the following facts have to be considered for getting a generator gas of the highest possible thermal value and also rich in CO.

1. The oxygen content of the primary air being the same, the gasifying temperature has to be high. In practice a temperature of 700° to 900° C. is sufficient, as at this temperature the maximum CO content is practically reached.

2. At high gasifying temperatures the quality of generator gas; *i. e.*, the content of CO, is increasing with the oxygen content of the primary air.

3. High air (wind) pressures are unfavorable, as thereby, under otherwise constant conditions, the CO_2 content is increased. If, however, it is desired to generate the largest possible quantity of CO_2 in the producer, which is sometimes the case in the hot blowing period of the water-gas process for the purpose of rapidly increasing the temperature, a very low

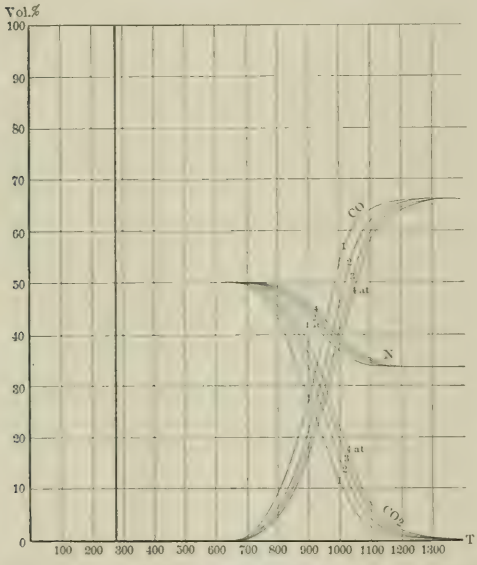

FIG. 3.—IDEAL COMPOSITION OF GENERATOR GAS FROM A MIXTURE OF 50 PER CENT O AND 50 PER CENT N.

temperature has to be kept during the process if the equilibrium is to be reached. This is easily understood, as with increasing temperature the quantity of the CO formed is rapidly increasing, and the quantity of CO_2 is decreasing. If in the producer the equilibrium is reached, the temperature of the producer must not get high if it is the intention to get a high yield of CO_2. These conditions are not changed by increasing the oxygen content of the primary air.

From the above facts we can calculate the volume proportions of CO_2 to CO, of CO_2 to CO + CO_2 and of CO to CO + CO_2, also the quantity of carbon gasified by a certain volume of air, the quantity of air necessary for gasifying a certain quantity of carbon, and also the quantity of carbon and air required for generating a certain volume of ideal generator gas.

We have so far treated the ideal generator gas; *i. e.*, a gas which is produced by the action of dry primary air on glowing coal, under the supposition that in the process of combustion the state of equilibrium is reached.

We now have to consider the case in which the equilibrium is not reached, this case occurring very frequently in practice.

Every single layer of coke consists of pieces of coke and air spaces between. The larger the pieces of coke the larger the air spaces. With coke of fist size, the air spaces amount to one-quarter to one-fifth of the total volume, and these spaces allow the air to pass through the producer.

Every piece of coal, therefore, is surrounded by a layer of air varying in thickness from a few millimeters to a few centimeters. The reaction between the oxygen of the air and the coal takes place only on their contact points, and the question arises which reaction will occur first. The law of the gradual reactions states, that wherever several reactions might take place, the first reaction is that one which corresponds to the least stable state, then the next stable, and at last the most stable.

In our case we have but two possible reactions: The formation of CO_2 and CO, and we have to find out which one of the two is more stable. We, therefore, have to consider the free energies of formation of the two compounds.

Under the supposition that the concentration of the free oxygen is one atmosphere, we find that the curves of the two energies of formation go through the same point at a little below 1,000° abs. (about 700° C.), and that at lower temperatures the free energy of formation of the CO_2 is the larger one at higher temperatures, that of CO. We find the same relation in the stability of the two compounds, and, therefore, at the beginning of the reaction at low temperatures first of all CO, at higher temperatures first of all CO_2 will be formed. In rising upwards the gases will further react with the upper layers of coal and with the air contained in the interior part of the gas current.

The reaction of the outer part of the gas current with coal consists either in combustion of coal by means of CO_2 or in formation of carbon from carbon monoxide ($2CO = CO_2 + C$). Since at low temperatures first of all CO, if formed, the most plausible reaction under such condition is the decomposition of the CO and formation of C. The reaction, however, between the inner and outer parts of the gas current counteracts this decomposition, since the O of the inner part would burn any C which was deposited from the CO. The velocity of diffusion and mixture between the inner and outer parts of the gas current being sufficiently large, no C will be deposited, on the contrary, the CO formed will be burned to CO_2, and the oxygen going to the outer part will oxidize some more carbon. Therefore, the average composition of the gas will approach more and more the equilibrium.

At higher temperatures at first CO_2 is formed, and this will, by contact with the higher layers of coal, oxidize some C to CO. On the other hand, the oxygen of the inner part will tend to oxidize the CO present to CO_2.

In both cases we have two effects counteracting each other. At low temperatures the reaction between coal and outer layer of gas tends to prevent the reaching of the equilibrium, while the reaction between outer and inner layers favors the approach to the equilibrium. At high temperatures, however, we find that the reaction between gas and coal favors the equilibrium, and the reaction in the gas current works against it.

The conditions become still more complicated, if we consider that the actual velocity of the gas current at different points of the generator varies according to the unequal dimensions of the air spaces, and that also the temperature throughout the generator is not at all uniform. If the generator is working with the fire on top (maximum temperature in the upper parts of the charge), the state of equilibrium of the rising gas current is getting more and more favorable to the formation of CO.

The reverse is true with the maximum temperature in the lower parts of the producer. The location of the maximum temperature of the producer, however, changes during the

operation. In starting the fire the upper layer of the generator will be cold, and will allow the formation of CO_2. They are gradually heated up by radiation of heat from the combustion gases to the coal, and the hot zone will therefore extend from the bottom further upwards. After continued blowing we can imagine a coke column which has the combustion temperature of the hot carbon in cold air.

As will be seen from the above considerations the research of the generator process is extremely difficult, and we have but a few scientific investigations on this subject. One of the best is by O. Boudouard, even this being not free from objectionable points. He passed air at different speeds through a tube filled with charcoal and analyzed the gases obtained. He found at 800° C. the results given in Table D:

Table D.

	Velocity in Liters per Minute.				
Per Cent. by Volume.	0.10.	0.27.	1.30.	1.4055.	3.20.
CO_2	18.2	18.43	18.02	10.9	10.4
CO	5.2	3.8	1.88	1.83	0.93
O_2	....	0.47	0.04	.....	0.03
N_2 (difference)	76.6	77.30	78.26	78.27	78.74

The analysis corresponding to the equilibrium at this temperature is

CO_2	0.02 per cent. by volume.
CO	34.32 per cent. by volume.
N	74.76 per cent. by volume.

It will be noticed that the gases from Boudouard's experiments are very high in CO_2 and very low in CO. In three cases they also contain free oxygen. This is in accordance with the fact that at 800° C. CO_2 is less stable than CO, so that, therefore, CO_2 must be formed first and the gas composition is approaching the equilibrium but gradually.

To better understand these conditions we are going to decompose the gases into the elementary components. We have in 22.42 liters of gas the amounts given in Table E.

According to the law of gradual reaction in the beginning, a thin layer of CO_2 is formed, which then oxidizes the coal layer through which it passes. It will, therefore, pretty nearly be correct to suppose that the outer layer (surface) of the gas current will have, shortly after its entrance into the tube, the composition which corresponds to the equilibrium. In this case the ratio of CO_2 to CO_2 + CO must be equal to 0.0261, and there must have been formed the amounts given in Table F:

Table F.

Velocity in Liters.	*Vol. CO_2.*	*Vol. CO.*	*Oxygen in Same.*	*Corresponding Amount of Air.*
0.10	0.61	22.70	12.01	57.19
0.27	0.58	21.05	11.41	54.33
1.30	0.54	20.26	10.67	50.81
1.465	0.54	20.19	10.64	50.67
3.20	0.53	19.80	10.43	49.67

If we deduct the air volume actually used for the original combustion from the volume of primary air, we get the surplus quantity of air from which we can figure by a simple way the surplus air given in Table G and Fig. 4:

The following consideration will be still more useful for the practical regulation of this process:

We suppose again that in the first moment the least stable gas is formed, but that in a short time on the surface area the equilibrium corresponding to the actual gasifying temperature will be reached. In the further course of the process this equilibrium will, however, be disturbed by the gradual mixture of the outer gas layer with the inner air volume, by the fall in temperature resulting therefrom, and by the combustion of a part of the original CO to CO_2, due to the surplus oxygen.

Referring again to Boudouard's experiments at 800° C., we can calculate from the free oxygen content of the gases the corresponding amount of air, deduct the latter from the composition of the gas, calculate the temperature of equilibrium corresponding to the gas mixture obtained, and compare the temperature of equilibrium with the actual gasifying temperature (800° C. = 1,073° abs.). We obtain thereby the results given in Table H:

As may be seen from Table H and from Fig. 5 the "ideal"

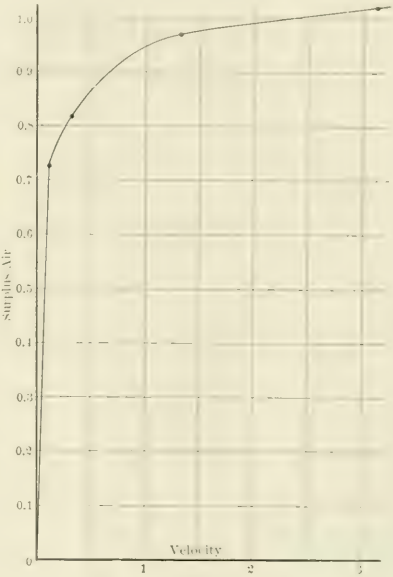

Fig. 4.—Curve of Surplus Air.

(or apparent) gasifying temperature corresponding to the actual composition of the gas is clearly below the actual, and the curve of this difference of temperatures consists of two practically straight branches, which are connected with each other by a short, sharply bent piece of curve. In the one branch, which is practically vertical, the velocity of reaction is the main factor, while in the inclined branch the velocity of the wind is of main importance.

Naturally, the position and shape of this curve depends not only on the gasifying temperature, but also on the size of coal used, and on the height of the fuel layer. Under conditions, however, which can be compared with each other, these additional factors will have the same character, and the position of the bending point of the curve seems a very suitable characteristic point for the conditions.

With increasing gasifying temperature, the velocity of reaction increases, and the bending point of the curve will move to the right. Increase of the fuel height and diameter and the coal size will have a similar effect. In the latter cases, how-

Table E.

Velocity in Liters per Minute.	*Gramatoms C in* CO_2	*CO*	*Total.*	*Gramatoms Oxygen in* CO_2	*CO*	*Free.*	*Total.*	*Nitrogen.*	*Temperature.*
0.	0.02	34.32	35.24	0.02	17.02	.....	18.54	69.70	81.37
0.10	18.2	5.2	23.4	18.2	2.6	.....	20.8	77.96	1,[illegible]
0.27	18.43	3.8	22.23	18.43	1.9	0.47	20.8	77.30	1,[illegible]
1.30	18.02	1.88	20.80	18.02	0.94	0.04	20.8	78.26	1,[illegible]
1.405	10.9	1.83	21.73	18.9	0.92	.....	20.18	78.27	1,[illegible]
3.20	10.4	0.93	20.33	10.4	0.47	0.03	21.20	78.74	1,[illegible]

TABLE G

Velocity in Liters per Minute.	In 100 Volumes Generator Gas Primary Air.	Volumes of Air for Original Combustion.	Surplus Quantity of Air.	Of 100 Volumes Primary Air. For Original Combustion.	Surplus Air.	N Times Surplus Air.
0.10	97.40	57.19	40.21	58.72	41.28	0.737
0.27	98.10	54.33	43.77	55.38	44.62	0.805
1.30	99.06	50.81	48.25	51.29	48.71	0.949
1.465	98.45	50.67	47.78	51.46	48.54	0.943
3.20	99.04	49.67	50.27	49.70	50.30	1.012

TABLE H.

	Velocity in Liters per Minute.					
	0	0.10	0.27	1.30	1.465	3.9°
Free oxygen, per cent. by volume	.....	.....	0.47	0.94	.....	0.93
Corresponding amount of air, per cent. by volume	.....	.....	2.24	4.48	.....	4.43
Composition of the gas free from air, per cent. by volume. CO_2	0.92	18.2	18.85	19.81	19.9	20.20
Composition of the gas free from air, per cent. by volume. CO	34.32	5.2	3.89	1.98	1.83	0.97
Composition of the gas free from air, per cent. by volume. N_2	64.76	76.6	77.26	78.21	78.27	78.74
Gasifying temperature (absol.), corresponding to the composition	1073°	763°	749°	732°	729°	7002
Difference between the latter and the actual gasifying temperature, which is higher by.	0°	307°	324°	341°	344°	373°

ever, some other influences have to be considered, such as friction between gas current and coal pieces, heating of the upper layers by the rising gas, location of the maximum temperature in the generator, etc.

The following figures are given as practical results of generators that were charged with carbonized fuel.

Ebelman gasified at Audincourt small-sized charcoal in a

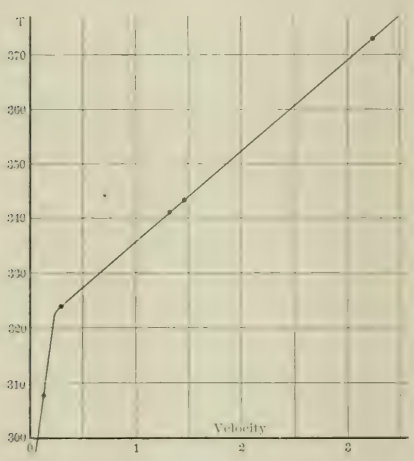

FIG. 5.—DIFFERENCE BETWEEN ACTUAL AND APPARENT GASIFYING TEMPERATURES.

pressure producer, which had the shape of a small blast furnace, and he obtained a gas of the following composition (per cent by weight):

CO	34.1%
CO_2	0.8
N	64.9
H_2	0.2
	100.0

In a gas producer at Pous l'Eveque, which was charged with coke, he obtained a gas of the following composition:

CO	33.8%
CO_2	1.3
N	64.8
H_2	0.1
	100.0

Notes on Electrochemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

THE IRON AND STEEL INSTITUTE MEETINGS.[1]

The second day's gathering at Great George Street was opened by Capt. ROBERT HUNT's presidential address to the American Institute of Mining Engineers. The speaker pointed out that the inception of the body he was addressing and the introduction of the Bessemer process into the United States were practically coincident. The address was, therefore, mainly descriptive of the history of the rise of the process in the United States of America, and the various statistical, economic and mechanical features of its development recited.

The conclusion of the address seemed to imply that in Capt. Hunt's opinion the Bessemer process had probably reached its zenith. Although the production of basic open-hearth steel rails was small, it was "the writing on the wall" and therefore significant. As to future developments in American metallurgy in general, he would place Mr. Gayley's work as next in importance to the gas engine using furnace gases. On behalf of his Society he had to present diplomas of honorary membership to Mr. Hadfield and Mr. J. E. Stead, as the highest honor that it was in their power to bestow on two distinguished English metallurgists. After suitable acknowledgment from the recipients the discussion of papers was commenced.

The first paper presented was that by Mr. ALBERT LADD COLBY, on "Comparisons of American and Foreign Rail Specifications," of which the author read an abstract. The paper was mainly devoted to a comparison of various specifications from the American standpoint.

The discussion was opened by Mr. Windsor Richards, who described phosphorus as the most undesirable constituent. The Bristol Engineering Standards Committee had fixed 0.07 per cent of phosphorus as the highest permissible proportion. Formerly the orthodox practice had been to specify 0.06, but the poorer ores now being used necessitated an increase. The manufacturers had asked for 0.08, but this was not conceded. Even Mr. Colby, able advocate though he was could not have extorted 0.08 from the engineering standards committee. According to English ideas, Mr. Colby's standard of 0.10 or 0.11 was far too high, and such rails would not be laid down here.

Mr. Price Williams suggested that a copy of the paper should be presented to the engineers of English railways. What was wanted was practical information on the subject—there had been too many micro-photographs and chemical analyses. The serviceable life of the rail was more important than chemical constitution. What was really necessary was definition of the tonnage actually required to wear

[1] This report will supplement our former reports on pages 349 and 362 of our last issue.

an ordinary rail down one-sixteenth of an inch.

Mr. Harbord urged that specifications should be so drafted that they could be lived up to, and the British standard of 0.07 phosphorus was one which manufacturers here could live up to. The hardness varied with the carbon present, and also with the process acid or basic. With the Bessemer process higher percentages of carbon could be used, a proportion of 0.6 and 0.7 had recently given excellent results.

Mr. Hadfield then congratulated Mr. Colby on his paper, and remarked on the difficulty which Mr. Colby would have had if he had attempted to convert the engineering standards committee. As to rails with 0.1 per cent of phosphorus which had not broken, Mr. Colby had not mentioned the proportions of carbon and manganese present; low carbon and manganese permitted durability with high phosphorus. He suggested that Mr. Colby should send over several samples of different rails to be tested on the alternating-stress machine at the National Physical Laboratory.

Dr. Raymond then read two communications. The first was from Mr. E. F. Kenny, and declared that very convincing testimony had been furnished that nearly every American railroad now suffered from broken rails. The brittleness was undoubtedly due to high phosphorus, which was an unmixed evil. So long as its proportion was near 0.1 per cent it would be impossible to get rails which would wear well. The arguments advanced in favor of such high proportions were not sound. Further, the straightening was often carelessly done. He advocated a maximum cumber before straightening; rails in excess of the fixed quantity ought to be classed as No. 2 quality and sold as such. Another was the length which should be discarded from each rail in order to limit segregation. The second communication was from Mr. W. E. Freir, and referred to corrugations and roaring rails. These were a nuisance and source of monetary loss, the cause being shrouded in mystery, there being explanations galore, none of which held water.

Mr. J. E. Stead pointed out that in regard to phosphorus chemists and metallurgists were all agreed that if the carbon was raised the effects of phosphorus became more pronounced. He regarded 0.1 phosphorus as a maximum, but manufacturers liked to have a working margin above the figure of the specification, which latter they liked to treat as an average and not as a maximum. He was personally prepared to advocate the occasional acceptance of isolated rails up to 0.1, if these were only few in number, and if the carbon and manganese were low. He believed that high manganese caused more broken rails than high phosphorus.

Mr. J. E. York then remarked that the physical treatment of rails was of much importance. Several internal stresses were often occasioned by the irregular operation of mills. The reason why English rails did not break like American rails was because English rails were delivered at a uniform speed.

Capt. Hunt stated that he was absolutely convinced American processes would be forced to change, and that the limitation of high phosphorus would lose its present significance. Mr. Colby then wound up the discussion by asking permission to reply in writing to the many points raised. He reminded the audience that the specification which he had submitted required a vigorous drop test, which was a safe check on brittleness.

A paper by Mr. R. H. Lee on "The Gas Producer as an Auxiliary in Iron Blast Furnace Practice" then elicited a very brief discussion.

The third and last day was a joint meeting of the two societies, Mr. Hadfield presiding. Mr. James P. Roe's paper on "The Development of the Roe Puddling Process" (page 353 of our last issue) was read in abstract, and the discussion opened by Dr. Raymond, who stated that the invention had provided a revival of the puddling furnace after its funeral had been announced and obituary eulogies delivered. Many complaints were now being made by farmers about their wire fences. These had given much trouble, and in many parts had given way. Careful investigations into the matter had been going on for some time, and it was thought that the breakages were due to the manganese in the steel. The old wrought iron wire, it was said, stood much better than steel. He thought that so long as manganese was put in steel—and it must be done—then there would be an opening for wrought iron in certain cases. If engineers were again going to resort to the puddle furnace, then mechanical puddling must be adopted if the quality of the metal was to be as good as large as in small quantities.

Mr. J. E. Stead complimented Mr. Roe on his ingenuity and dogged determination in the face of difficulties apparently insuperable. Mr. Sahlin explained that he had seen the process at work, but when he first heard of it he regretted that so good a worker should risk his reputation on an attempt to resurrect an old process. Seeing it in successful operation, he had noted with pleasure the solid homogeneity of the product. The machine, too, was good as an engineering work, none of the penurious economy often found in experimental work being noticeable. Wrought iron manufacturers competed with steel makers for cast iron, the latter securing the higher grades.

Mr. E. S. Cook pointed out that wrought iron manufacturers were important to blast furnace owners, as customers for lower grades of cast iron; within certain limits the chemical constitution of pig iron did not trouble the puddler. Mr. Hartshorne praised the Roe process as a practical and technical success, and was sure it would be a commercial success. Mr. Kerr, as an old puddler, welcomed the machine, and wanted to know whether a high silicon iron would be treated in the same way as a low silicon iron. Mr. Paul attributed the elimination of silicon to the low temperatures Mr. Roe employed. He was surprised, however, that the development had not been carried a stage further, and the regenerative process adapted to the furnace. Had the Danks furnace been worked entirely with gas it would have been alive to-day. As to corrosion in iron and steel, much emphasis had been laid on manganese. Was not corrosion attributable to the galvanic action of the metalloids? The greater the number of the elements they introduced the greater the risk of galvanic action.

Mr. E. P. Martin pointed out that the most difficult question in puddling was that of the maintenance of the bottom of the furnace, which Mr. Roe had solved for his furnace. Prof. Bauerman remarked on the exceeding interest of the paper, and how rarely it was that inventors took audiences so fully into their confidence. As to rotary furnaces, these were generally spoken of in the past tense, but some still survived, a Creuzot modification of the Danks furnace being in use at Terni, in Italy. The original Danks furnace was weak in construction, and gave too large a ball for manipulation. The author now proposed to give a bloom double the weight of Danks. He did not feel sure of the propriety of this. Possibly a smaller mass would give better results. Prof. Turner supported Prof. Bauerman's view as to the large blooms, and said that the future of wrought iron would depend upon a good quality of material. Fiber and texture were of the utmost importance, and it had yet to be proved that the larger bloom would give a good enough texture.

A vote of thanks, moved by Mr. Hadfield, wound up the discussion on this paper.

The last item in the way of discussed papers was that by Mr. James E. York on "Improvements in Rolling Iron and Steel." As to what was said, a leading English engineering paper characterizes the discussion as "short, uninteresting and irrelevant." This description lacks the word "farcical." The author was frankly and humorously autobiographical, and the walls, covered with paintings of solemn and revered past presidents of the Institution of Civil Engineers, echoed to the laughter of the audience. Uninteresting, well, yes, if the low-

comedy touches in a tragedy are uninteresting. Irrelevant! well, yes, if the wit of the grave-digger in *Hamlet* is irrelevant. But in any case unusual, unaccustomed and undignified for such an assembly.

Metallurgical Papers at the British Association.

This year's British Association meeting, at York, was considerably ahead of its predecessors in the high standard of excellence of the papers secured. Section G, which is devoted to engineering matters, had a very strenuous time, the metallurgical papers alone being quite numerous. These included "The Deformation and Fracture of Iron and Steel," by Mr. W. Rosenhain; "Segregation in Steel Ingots and Its Effect in Modifying the Mechanical Properties of Steel," by Mr. J. E. Stead; "Structural in Nickel Wire at High Temperatures." by Dr. H. C. H. Carpenter; "A Magnetic Indicator of Temperature for Hardening Steel" and "Electro-Positive Coatings for the Preservation of Iron and Steel," by Mr. Sherard Cowper-Coles.

Also of indirect interest was Prof. T. Hudson Beare's account of "The New Engineering Laboratories at Edinburgh University and Their Equipment," as the contents of the testing laboratory was described, this containing a 100-ton Buckton machine, a 60,000-pound Riehle machine, and a 100-ton Amsler machine, specially designed for compression and bending work. Sir William Preece's paper on "Glow Lamps and the Grading of Voltages" was interesting on account of the large number of metallic filament lamps alluded to in the discussion.

Market Prices During August.

The general activity of all manufacturing concerns, particularly those of the engineering trades, has resulted in higher prices for the chief raw material. Cleveland pig iron has advanced during the month from 51s 2d to 54s 6d per ton. Copper prices have been well maintained. There have been several bear attempts to depreciate prices on account of increased production. Nevertheless, the price has advanced from £82.5s per ton to £84.15s. Hand-to-mouth-buying is still going on here. Tin has advanced again during the month, having advanced from £175 to £182.10s. Friends in the Straits Settlements tell me of considerable activity in the tin mining industry. Lead is firm, English being quoted at £17.10s for prompt delivery. Antimony is easier at £105 per ton.

Platinum is quoted at 91s 3d per ounce.

Copper sulphate is firm at £25.5s. Ammonia sulphate is also 5s dearer. at £12.2s 6d. Prices of the coal tar products are fairly well maintained, liquid carbolic acid 97-99 per cent fetching 11d per gallon. Cyanides 98 per cent, minimum are fetching 8½d. Caustic potash has advanced 5s to £19.15s per ton, and prussiate yellow ¼d to 5¾d per pound. On the other hand, Montreal potash in store, Liverpool, is quoted £4 per ton lower at £31. The only change to be chronicled in regard to the soda products is the higher price of £11.5s for nitrate of soda.

London, Sept. 1, 1906.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

Electrometallurgy.

Preparation of Molybdenum in the Electric Furnace.—Dr. C. Lehmer, in *Metallurgie*, Aug. 22, gives an account of the experiments which he undertook with molybdenite in the electric furnace, for the purpose of producing metallic molybdenum and molybdenum alloys. The equation which he attempted to realize for obtaining molybdenum was $MoS_2 + 2CaO + 2C = Mo + 2CaS + 2CO$. After several trials with varying amounts of material, and increasing the quantity of lime considerably above that required theoretically, he smelted a charge of 1,500 grams MoS_2, 1,400 grams CaO and 385 grams C, and obtained a yield of 85 per cent, of a metal of the following composition: Mo 88.3 per cent, Fe 2.89 per cent, Si 2.95 per cent, S 0.04 per cent and C 5.6 per cent. The melting was done in a furnace of the Héroult type. For producing ferro-molybdenum he smelted a charge of the following proportions: MoS_2 2,500 grams, Fe_2O_3 2,150 grams, $CaCO_3$ 4,160 grams and C 900 grams. The analysis of the metal obtained was: Fe 46.56 per cent, Mo 48.72 per cent, Si 2.61 per cent, S 0.13 per cent and C 1.06 per cent. The metal produced formed a coherent regulus and had a silver-gray color, but was still very hard, notwithstanding the small amount of carbon it contained. For the production of chrome-molybdenum, the author used a charge of 1,218 grams of Cr_2O_3, 960 grams MoS_2, 1,200 grams $CaCO_3$ and 600 grams C. The analysis of the metal obtained showed Cr 53.44 per cent, Mo 34.10 per cent, Fe 3.71 per cent, Si 0.21 per cent, S 0.09 per cent, C 7.4 per cent. The charge was briquetted with tar. The yield of metal was 83 per cent. The next experiment was made with the addition of CaO, the carbon being only in the form of tar residuums, obtained by repeated briquetting and ignition. The charge contained 900 grams Cr_2O_3, 480 grams MoS_2 and 896 grams CaO. The analysis of the metal gave Cr 57.21 per cent, Mo 36.10 per cent, Fe 3.82 per cent, Si 0.30 per cent, S 0.07 per cent and C 2.1 per cent. The yield was 79 per cent. The color of the alloy was silvery white, the hardness about No. 9 of the scale, and it could easily be broken with a hammer. The experiments thus far described were made in a furnace lined with magnesite. The next experiments were made in order to produce nickel-molybdenum from a charge of 1,195 grams NiO, 640 grams MoS_2 448 grams CaO and 280 grams C, the charge having been briquetted with a small amount of tar. The yield was 89 per cent. The color of the alloy obtained was that of pure nickel, and on account of its soft nature it could be easily flattened out with a hammer. It had the following composition: Ni 69.20 per cent, Mo 26.15 per cent, Fe 3.12 per cent, Si 0.30 per cent and S 1.23 per cent. The comparatively large amount of sulphur is to be charged to the quantity of lime being only the theoretical amount. An experiment with a larger amount of lime, in a charge composed of 1,195 grams NiO, 640 grams MoS_2, 760 grams CaO and 300 grams C, gave the following alloy: Ni 68.67 per cent, Mo 26.05 per cent, Fe 2.61 per cent, Si 0.27 per cent, S 0.07 per cent and C 2.10 per cent. The yield of metal was 85 per cent. Another charge made up of 472 grams NiO, 640 grams MoS_2, 760 grams CaO, 250 grams C, composed of material of 2 to 3 millimeter size and not briquetted, gave an alloy of the following composition: Ni 48.53 per cent, Mo 49.10 per cent, Fe 1.86 per cent, Si 0.25 per cent, S 0.18 per cent and C none.

Metallurgy—Copper

Copper Converter Melting its Own Matte.—Owing to the reconstruction of the blast furnace plant of the British Columbia Copper Co., at Greenwood, B. C., it becomes necessary to convert a quantity of copper matte and to melt it in the

converter. According to the *British Columbia Mining Record*, July, the converters at the plant are of the horizontal trough type, 84 inches in diameter by 126 inches long, with about 4,500 cubic feet of free air available at 10 pounds pressure. In order to melt the matte, a fire was first lighted in the bottom of the converter, and when this was thoroughly alight, about 1,500 pounds of coke were dumped on it and a light blast of air put on. When the fuel became red hot all through, some 3 tons of cold matte were placed on top of the brightly burning coke, and the full pressure of air was turned on. The matte quickly melted and more matte was immediately added to increase the volume of the charge. When this was melted in turn, the slag was skimmed off and still more matte was added, the operation being repeated until there was sufficient matte in the converter so that the charge could be properly converted. An average of 3½ hours was required from the time of commencing to fire the cold shell until the blister copper was poured. About 15 tons of 40 to 50 per cent matte was converted in one stand in one ordinary shift. It was found that a 45 per cent matte could easily be handled in the manner described above and finished hot. Matte up to 55 per cent was also successfully converted, but with this more care was required to maintain the requisite heat, and occasionally it was found necessary to add a few hundred pounds of coal to prevent freezing of the charge. The operation, of course, is not economical, but shows what can be done in an emergency.

Lead.

The Huntingdon-Heberlein Installation at Nelson, B. C.—The new installations for the treatment of complex lead ores at the above mill by the Huntingdon-Heberlein process are described by H. Harris in the July issue of the *British Columbia Mining Record*. The process, as is well known, consists in partially roasting at a high temperature a mixture of fine crushed lead sulphide ore with carbonate of lime, the granular product being subjected to the action of an air blast in a cast-iron hemispherical pot. Practically, all the ore used has to be pulverized to a 6-mesh, and it is, therefore, crushed in a large Gates gyratory crusher, sampled, passed through rolls and brought into the bins. Automatic feeders are provided at the foot of these bins, by means of which the ore is fed onto a conveyor belt and discharged into a cylindrical mixer. This in turn discharges into a bin from which the roasters are charged. The new roaster installation contains one Merton roasting furnace, converted and enlarged to adapt it for use with the new process, and one 30-foot Huntingdon roaster, provision being made for a second. This machine has 50 tons capacity per day, and differs from most rabble-stirred automatic roasters in that the rabbles are fixed to the roof, and the circular bed, which is 26 feet in diameter, revolves, thus pushing the ore against the rabbles. The partially roasted ore from the mechanical roasters drops against a water spray into a large bin, whence it is conveyed to small bins located above the converters. The latter are six in number, 9 feet in diameter, and of a hemispherical form. They stand 17 feet above the dumping level, and are supplied with air by a No. 7 Connellsville blower. They are surmounted by hoods connected by telescopic pipes to a fan 7 feet in diameter, which serves to draw away the said fumes. One converter holds from 12 to 15 tons of charge, and the desulphurization requires from 8 to 12 hours. At the end of that time the converter is inverted and the contents are dumped on the floor below, where the mass is broken roughly by means of a drop weight operated by an electric hoist and by means of hand labor. The broken sintered product is pushed down a chute into a side dump-car, which automatically runs to and discharges its contents into storage bins situated on the railroad track, whence they are transferred to bins alongside of the blast furnace, where the material receives its final treatment.

Gold and Silver.

The Pohle-Croasdale Process at Mayer, Yavapai County, Arizona.—The Pohle-Croasdale process depends essentially upon the chlorination of the precious metals and their volatilization as the chlorides and subsequent condensation. The ores are crushed dry, mixed with salt and subjected to an oxidizing atmosphere of 1,000° C., as above. The chemical equation $2NaCl + S + 2O_2 = Na_2SO_4 + Cl_2$ shows the necessity for sulphur being present. As the amount of chlorine must exceed the theoretical amount required, the percentage of salt to be added must, of course, be greater than the theoretical required by the equation. With an average ore the limits are said to be placed at from 5 to 12 per cent of salt, but with higher lead and copper contents more will be required in order to volatilize these metals. The relative proportion of salt to sulphur, as expressed by the equation given above, is 10 per cent salt to 2.3 per cent of sulphur. In practical tests it was, however, determined that when over 3 per cent of sulphur is present, roasting is necessary, as otherwise the amount of salt consumed is too high. The practical application of the above process at the new plant of the Rigby Mining & Production Co., at Mayer, Yavapai County, Arizona, is described by Mr. O. H. Fairchild in *Mining and Scient. Press*, Sept. 1. This is the first application of the process on a large scale. The ore is crushed and dried and thoroughly mixed with salt in a square box made of sheet steel, patterned after a cement mixer. It then is fed into the furnaces, which consist of wrought iron cylinders with ½-inch shells, 5 feet 6 inches in diameter at the feed end, and 7 feet 6 inches inside diameter at the discharge end. They are 36 feet long, and lined with 6 inches of fire-brick. They have a series of standard brick projecting 3 inches beyond the inside surface of the lining, which serve as rabbles. The fire-box is situated at the discharge end of the furnace, while at the feed end is a smoke-box 4 feet x 6 feet clear on the inside. The ore after treatment is discharged at the fire-box end into a conveyor, which takes it to the dump. The fumes are conducted by a pipe, 4 feet in diameter, from the smoke-box to the inlet collection chambers. The latter are of brick, 12 feet 8 inches long by 5 feet 2 inches wide, the inlet chambers being 6 feet high, while the outlet chambers are 5 feet high. Midway between them is a chamber 5 feet square and 5 feet high, on which rests the center portion of the M pipe. The chambers are so built that from the inlet where the gases are introduced, they pass through an A pipe 20 feet high at the apex, rising and descending to the outlet chamber. The gases then return to the inlet through an M pipe of equal height, and then pass back from the inlet to the outlet chamber again through another A pipe. These pipes have a total length of 200 feet and a diameter of 3 feet; they are built of sheet steel. The fumes are then forced by a Root blower into the condensing chambers, which latter are located in a building 96 feet long by 44 feet wide. One half of which is 28 feet high, while the other half, containing the bag houses, is 40 feet high. The condensing chambers are 21 feet x 16 feet, and are built 6 feet high of brick, and plastered on the sides and bottom with smooth cement. At the height of 6 feet from the floor is a sheet-iron floor, which contains apertures for fifty-four 18-inch flannel bags, 30 feet long. The materials obtained in the collection chambers, condensing chambers and bag houses are taken to the filter house, where the copper is separated. From there the solution goes to the precipitation house, which contains tanks for the precipitation of the copper, the only commercial metal contained in the solution. It is stated that 50 or 60 minutes will volatilize the gold contents of an ore with a furnace temperature of 1,000° C. or above. The gold volatilizes as chloride or possibly as double chloride with sodium, but these break up soon after leaving the furnace and metallic gold is deposited. The silver sublimate is deposited as chloride or sub-chloride, insoluble in water or dilute acids. The lead sublimate is a mixture of both the chloride and sulphide salts. The copper volatilizes readily, but the time required is usually 2 to 3 hours, if a considerable amount of copper is present. The sublimate is composed of

cupric, cuprous and oxy-chlorides principally. Zinc compounds volatilize very imperfectly, so that 50 to 70 per cent of the original zinc is left in the residue. Nickel and cobalt appear to behave much like the zinc.

Cyanide Practice with the Moore Filter.—The first instalment of an article by R. Gilman Brown in the *Mining and Scientific Press*, Sept. 1, deals with the practice at the Standard mine at Bodie, Cal., in the treatment of the slimes. The ore consists of quartz, iron oxides and clay, the latter a result of the decomposition of the feldspar in the country rock. This on an average is 33 per cent. The gold is partly coarse and partly very fine, and the latter portion amalgamates very badly. The proportion of silver is high, and the ore is difficult to treat, although all minerals ordinarily classed as deleterious are absent. In conjunction with the introduction of the Moore method weak cyanide solution was substituted for water in the 20-stamp mill, the grade of the plates being increased from 1½ to 2½ inches per foot, for the purpose of cutting down the proportion of liquid to keep the plates clear. This steep grade, combined with the hastening effect of the cyanide on the amalgam resulted in lowering the extraction by amalgamation from 60 to 50 per cent. The strength of the cyanide solution was found by experiments to give the best results at 2 pounds per ton, but lately the lower plates where the amalgam lays the thinnest have shown signs of wasting, and some have been renewed. Ten pounds per ton of lime are added to the ore before it goes to the crusher. After passing over vanners, the pulp is conveyed to the slimes plant, where it goes to two cone separators. The latter are of wood, approximately 5 feet deep, and have a 60° slope. The underflow, which contains the coarse stuff and a certain portion of adhering slime, passes to the tube mill, and is mixed with sand and slime from the ponds, automatically fed into the stream. The mill makes 26 r. p. m., and is charged with about 12 tons of Greenland concretionary flints, which charge fills the mill a short distance above the middle. This charge seems the best for grinding and most economical for power, according to the author, but the difference is not great, and the charge can wear to half this quantity with small diminution of grinding. After unsatisfactory trials with cast iron and wooden, as well as silex linings, wrought iron plates were finally adopted. These plates are ⅞ inch x 8 inch, cut into 7 and 15-foot lengths and bolted through the shell. The average life of these is stated to be over 100 days, and the duty 4,800 tons of sand, ground to 200-mesh. The consumption of pebbles is given as 6.47 pounds per ton, the wear of the lining being 2.44 pounds per ton. The wrought iron lining can be worn down thin without breaking, and it can be inserted in 10 hours. The power required for driving the mill is approximately at 50 hp. when running and 100 hp. when starting. The maximum grinding capacity of the single unit is stated not to have been definitely reached, but it may safely be placed at 60 tons of sand per 24 hours, the duty of a horse-power-month thus being 36 tons. The normal cost of electric power in California, the power cost comes to 17 cents per ton, while at the Standard mine, which produces its own power, it is 5⅓ cents. The author states that experiments have also been made with a modified pan for fine grinding. It was found that a single 5 foot pan had a capacity of about 5 tons per 24 hours, and consumed 10 hp. Thus, from eight to nine pans were required to handle the coarse product of twenty stamps at a marked increase of power cost. No determination was made of the wear of metal, but judging from experience with grinding concentrates, it would be high, while the labor of attending to a battery of pans would also prove high. The author, therefore, is of the opinion that, together for a small plant as far as the experience goes, three smaller tube mills, any two of which could do the whole work, would be preferable to pans, and probably cheaper in operation than two large units.

Device for Cleaning Battery Plates Without Stopping the Stamps.—In order to save the loss of time occasioned by stopping the stamps when it becomes necessary to dress the battery plates, Mr. W. Beaver, at the Globe & Phœnix Co. mill, Johannesburg, Transvaal, constructed an appliance which he describes in a paper before the Chemical, Metallurgical and Mining Society of South Africa, which is published in the journal of the Society, June, 1906. Fixed to the top of the chuck block, and immediately beneath the screen frame, he arranges a catch plate of sheet iron, 8 inches wide, which deflects the pulp outwards, and thus allows of a satisfactory sample being taken. When running in the usual manner, the pulp falls from this catch plate on a baffle-board, which deflects it back onto the mortar lip. When, however, it is intended to dress the main plate, this baffle-board is removed, and the pulp deflected into a sheet-iron launder, resting on chains fixed to the sides of the table frame, which carries the pulp from the catch plate to one side of the plate. There it enters a portable launder, 14 feet x 6 inches x 6 inches, which conveys it to a permanent launder running the full length of the mill. This permanent launder discharges into a distributing box placed at the head of an extra plate, 12 feet x 5 feet, situated outside of the mill. The plate has a lock-up cover and is dressed each time immediately before dressing the main plates. When a plate is to be dressed the amalgamator first puts the portable launder into position and then adjusts the sheet iron trough, after which the pulp flows directly to the extra plate described above. The screen is covered with a piece of canvas to prevent splashing, and two pieces of sheet iron pushed into saw cuts in the screen frame at each end serve to throw the pulp from the corners of the screen frame towards the center of the catch plate. Splashing and leakage onto the main plate are therefore prevented. The author states that the device has been quite satisfactory, the main plate being quite dry and clean during dressing and steaming. The extra plate only runs 4 hours and 20 minutes, but it behaves quite normally. He figures that the time saved in this mill, which would have been been consumed by stopping the stamps, amounts to a total of 9 days and 2 hours per year, which in his forty-stamp mill represents an extra crushing capacity of 1,816 tons, with practically no extra cost except the steam for the stamps. The saving in cam sticks alone is considerable.

Treatment of Zinc Gold Slimes Before Smelting.—As a substitute for the ordinary treatment of zinc slimes by the sulphuric acid process, Mr. C. E. Meyer, in the *Journal* of the Chemical, Metallurgical and Mining Society of South Africa, June, 1906, proposes a treatment with ammonium carbonate, for which he claims various advantages. The important feature of the process consists in the previous oxidation of the slimes. They are taken from the precipitation boxes and placed in a vat, where they are brought to complete dryness by steam pipe attachment. During this procedure oxidation sets in, and any incomplete oxidation is hastened and perfected by moistening the slimes with nitric acid. Not much acid will be necessary, as the slimes oxidize easily, and any excess of the acid must be expelled by heating. This process of oxidation must be carried out with care, and an excess of acid must be avoided, as this will cause an unnecessary expenditure of time and money. The easily decomposed cyanides are thus oxidized while those cyanides which are only decomposable with difficulty, will be acted upon by the subsequent treatment with ammonium carbonate. The next step in the process consists in the addition of a preferably saturated solution of ammonium carbonate, the quantity required being about ten times the volume of slime under treatment. This ammonium carbonate throws down metals such as zinc, silver and copper as precipitates, dissolving in the excess of the reagent. Experiments on the same quantity of slimes, according to the author, resulted when the sulphuric acid process was used, in the production of a very heavy filtrate of a viscous nature which filtered slowly, the residue on the filter containing a considerable quantity of zinc and copper. On the other hand,

the ammonium carbonate process gave a light, clear filtrate, while the bulk of the residue on the filter was considerably less and showed no zinc and copper. He claims, therefore, that the process has the advantage of superior filtering qualities and a complete elimination of the base metals, beside greater ease in handling. The filtrate from the filter press is kept and distilled over with powdered limestone or dolomite in a large boiler, by which means the ammonium carbonate is driven off and a by-product of a matte consisting of silver, lead and copper is left. The ammonium carbonate is obtained in solution by arranging a spraying device in the condensation chamber.

Observations on Tube Milling.—Tube milling of gold ores is a subject which engages considerable attention in metallurgical circles all over the world, and a good deal of important work is being done to ascertain the most favorable conditions for running the tube mill. A recent contribution to the subject is a communication of Mr. E. J. Laschinger, before the Chemical, Metallurgical and Mining Society of South Africa, in the Society's monthly journal for June. He investigates the causes of the wear of the silex linings and comes to the conclusion, based upon microscopic examination, that the grinding action of the pebbles on the lining is of much greater importance than the chipping action. By chipping action is not meant the loss of weight which the blocks suffer by flakes being chipped off their sides, but rather the impact of the pebbles striking anywhere on the surface of the liner and causing the loosening of small particles of the material. For the same duty, the wear must be in some measure inversely proportional to the area of the effective wearing surface. If it is assumed that the cement in the joints is not an effective wearing surface, it follows that the greater the percentage area of joints the less the effective area and the greater the rate of wear. Carefully measured, it would appear that the ordinary silex blocks, as used at the Robinson Deep Mill, which measure on the average 7½ x 4½ x 2½ inches, when placed as closely together as possible, have, when set flat, an effective wearing surface of 95.4 per cent of the circumference of the tube mill, while, when set on edge, the effective surface is only 86.1 per cent. These are in the ratio of 111 to 100, and if the wear be inversely proportional to the effective surface, the rates of wear will be in the above ratio. It thus is evident that blocks should be made as large as possible, with due regard, of course, to other practical considerations, and that the joints should be made as close as possible when setting the blocks. The pulverizing effect of the tube mill is the greater the less the amount of water introduced with the feed. As the chief cause of reduction is probably the crushing action on particles caught between individual pebbles falling on each other, this may be explained by the cushioning of the blows by the water. The author thinks, therefore, that there may be two ways in which the efficiency of wet grinding can be increased and the life of the liners be prolonged, and that is (1) by introducing as little water as possible with the feed, and (2) by allowing any surplus water introduced into the mill to drain out quickly, thus leaving the particles to be ground as dry as possible. He suggests that this second method of increasing efficiency might be attained by having the discharge plate or screen of the tube mill in the shape of a disc, and perforated all over its area, so that the water could drain out as quickly and thoroughly as possible. Two other important factors in the life of liners may be the pebble load and the speed of the mill, and even small differences may account for large differences in the life of the liners and the wear of the pebbles.

Tube Mill Practice.—Mr. S. S. Osborn, in the journal of the Chemical, Metallurgical and Mining Society of South Africa, June, contributes some remarks on the practical operation of tube mills. He lays stress again on the importance of a good classification of the pulp, and thinks that the present arrangement of dewatering spitzkasten on the Rand fields is not altogether satisfactory. The pulp falls from too great a height into the center of a spitzkasten [illegible] the consequence that a larger proportion [illegible] carried over the side instead of passing through the tube mill, thus causing an increase in the quantity of pulp in circulation. This difficulty can be to a certain extent overcome by placing a sleeve around the inlet pipe and baffle below, as is done in the slimes collecting tanks. When ores are [illegible] charged with marcosite, as is the case in the Glen Deep and the Robinson Deep mines, fine grinding has to be resorted to, and according to the author every effort should be made to grind all pulp sufficiently fine to allow of its passing through a 60-mesh screen. The author states, that as far as the feed end of the tube mill is concerned, where the outlet of the mill is much larger than the inlet, nothing need be added to prevent the pulp from flowing out at the inlet end. On the other hand, however, where there is not much difference in the size of the inlet and the outlet, there is a tendency for the pulp to pass out at the feed end, and, therefore, the gland and top feed were adopted in the Allis-Chalmers mill. He thinks that there seems no necessity for any elaborate fixture on the feed end, as a thin disc, with a 5-inch hole in the center, attached to feed end, prevents the pulp from flowing out, while it is not a difficult matter to feed in the required amount of pebbles through the 5-inch hole. At the discharge end of the tube mill, excellent results have been obtained at the Glen Deep mill by replacing the screens supplied by the makers by a plain disc of white iron, 1½ inches thick, and perforated with holes tapering from 1 inch to ⅜ inch. Broken pebbles cannot become lodged in such a hole. As far as the pebbles are concerned the author maintains that the secret of low pebble consumption is to pass a large tonnage of pulp through the mill, so as to insure having a cushion of that material between the pebbles when they are striking one another in the mill. To substantiate this statement he gives the following results:

Liners.	Life Days	Hrs	Battery Screening 200	150	Tons Crushed in Battery	Pebbles Fed In	Tons per 24 Hours Through Tube Mill	Pebble Consumption per Ton Crushed in Battery.
Chilled Iron	20	21	100	64	2,186	6 541	0.5	3 lbs.
Silex.....	51	1	64	100	11 727	31,040	3.0	2.65
Manganese Steel.....	97	13	100	200	25 810	30 7.0	522	1 42
Silex.....	48	—	100	200 400	27.354	16,250	917	.59

In most cases the tonnage passing through the tube mill was larger than the total tons crushed by the stamps, owing to having a larger quantity of pulp in circulation.

Filter Pressing for Clarifying Gold-Bearing Solutions.—The results obtained by using two small Johnson filter presses for clarifying the precious metal-bearing solutions coming from the slimes plant and going to the precipitation boxes, are given by Messrs. S. J. Truscott and A. Yates, in a paper read before the Chemical, Metallurgical and Mining Society of South Africa, printed in the July issue of the journal of that Society. The presses had each thirty-four cells, 30 inches square and 2 inches deep. Previous to the utilization of the presses the solutions from the slimes plant, amounting to some 300 tons per 24 hours, passed along a launder, where a little slump slime was added at intervals. This launder discharged into a 13 foot x 5-foot settling tank, which overflowed into an 8 inch x 5 inch tank, and this again fed another one of similar size, from which the solution was pumped up to the extractor house. There it passed through a 16-foot x 6-foot tank before entering the zinc boxes. Slime settled in all these tanks and was from time to time cleaned out, but much also settled in the extractor boxes and fouled the precipitate, though the solution had but little appearance of turbidity. At the present time, as the two Johnson presses are used, the addition of lime is dispensed with. The solution is pumped from the first settling tank into the presses, from which it issues crystal clear. The pump used is an Evans steam pump of 5 inch diameter and 12-inch stroke, operated by compressed

air, which serves also as a reserve solution lift pump. The presses are opened twice a week, and the clothes well scrubbed to free them from fine slimes. The filter cloths which have already been in use two months in the zinc gold slimes filter press, and have become hard in that service, are still effective for another month in clarifying the solutions. The authors remark that the beneficial effects of this filter press clarification upon the solution were at once felt in the smelting. The precipitate became clear and was easily reduced. Thus, for instance, in January, 1905, with the settling tanks, and a total recovery of 2,269.8 ounces of gold and 17,853.7 ounces of silver, the cost of smelting per ounce of fine gold recovered was 27.2d and the cost of smelting per ounce of fine metal recovered was 3,07d. On the other hand, in Dec., 1905, with the clarifying presses, and a total recovery of 3,336.8 ounces of gold and 19,510.4 ounces of silver, the cost of smelting per ounce of fine gold recovered was 6.01d. and per ounce of fine metal recovered 0.87d. The authors think that one press would be sufficient if more frequently opened, and if that press were of the inside-closed channel type. the force pump which sent the solution into the press would also force it into the extractor boxes wherever they were.

Treatment of Silver Lead Tailings by the Cyanide Process.—A description of the method used for cyaniding silver lead tailings at the Winters mine, in Douglas County, Nevada, is given by the manager, Mr. E. J. Sweetland, in the *Eng. and Min. Journal*, Aug. 25. The low-grade ores, which remain after sorting out the shipping ore, are treated by crushing in quadruple issue mortars, the pulp being passed over New Standard concentrators. The value of the ore is principally in lead and silver, the lead being commonly present as carbonate, though galena is also present. When the concentrators are properly adjusted the galena is found in the outer streak of concentrates, and is followed by a well defined streak of lead carbonate, running well in lead and silver and almost free from sand. Inside of this is a darker streak, carrying lead and silver and some sand. the percentage of which gradually increases as it approaches the tailings end of the table. It was found that the most economical point for separating the concentrates from the tailings was at the center of the streak last mentioned. The tailings are run into one of three reservoirs, and are allowed to settle. The tailings average 1.25 per cent lead and 0.11 per cent copper, with traces of zinc, arsenic and antimony. The copper is largely present as carbonate, which is readily attracted by potassium cyanide. The average value is 0.41 in gold and 7.6 ounces silver per ton. It was found that it was possible to treat the tailings without an excessive consumption of cyanide, on account of the silver existing largely in the form of cerargyrite or chloride. Thus a fair amount of silver could be dissolved with a solution so dilute as to have little effect upon the salts of copper and lead. A series of laboratory experiments were run in order to determine the best form of treatment. It was found that on account of the selective action of the cyanide for chloride of silver, practically as much is dissolved by a solution of 0.1 per cent KCy as by a stronger solution, while the base metals are not as strongly attacked by the weaker solution. It was also ascertained that the extraction could not be materially improved by combined contact with a weak solution. The laboratory experiments were confirmed by practical results. The cyanide plant consists of four leaching vats, each 17 feet in diameter, with 4-foot staves, fitted with a false bottom of slats covered with a layer of burlap and 6-ounce duck. The burlap and duck are held in place by a ¾-inch hemp rope driven in a groove between the staves and the false bottom. There are two 10 x 10 inch bottom discharge doors in each vat. The vats are connected to two collecting tanks by a double line of 1½ inch pipe. There are sixteen circular sheet-iron boxes in two rows of eight each, 16 inches in diameter and 18 inches high, each provided with a screen 2 inches from the bottom, and a baffle-board to direct the flow under the screen. The author states that though this seems to be a very small zinc-box capacity for a plant treating silver ores, he found it to be ample. The solution after reaching the sump is returned to the stock tanks by a Gould rotary pump, actuated by a 3-hp. Fairbanks-Morse gasoline engine. The vats are filled by a wheeled scraper and leveled, and are charged with solution from below. Immediately when the solution covers the surface of the charge, the flow is stopped from below and turned on at the top, leaching being commenced at once, omitting the customary period of soaking. The solutions are allowed to percolate unrestrained, but when a solution leached too freely on account of lack of slime, it was found of advantage to cut down the flow, as the passage of too great an excess of solution increased the consumption of cyanide. The clean up of the zinc slimes is affected in the usual manner, the solution being carried out in a redwood tank, which gave excellent satisfaction. The resulting bullion was about 750 fine.

Power.

Electrical Equipment of the El Oro Mill.—In a lengthy article on the recent devlopments in the installation of electrical machinery in the mills of the El Oro mines district, C. V. Allen, *Engineering Magazine*, September, gives some data regarding the use of electricity in the mill of the El Oro Mining & Railway Co. The total amount of power used amounts to some 4,000 hp. All the motors are three-phase, 50-cycle (6,000 alternations) machines, and as a rule they are 400-volt induction motors for 200 hp. and less, and 3.000-volt synchronous motors above 200 hp. For the older mill, No. 1, the main stamp shaft was formerly rope-driven from its engine, but it is now driven by a 350-hp. synchronous motor, making 187 revolutions. From this main shaft a belt drive transmits the power to the cam shaft of the batteries in groups of ten stamps.

This cam shaft makes close to 51 r. p. m., thus giving the stamps about 100 drops per minute. The stamps weigh about 1,000 pounds, and are lifted 6 inches. The duty per stamp is 4.7 tons per 24 hours. This motor, and all other synchronous motors, is started by means of a small variable-speed induction motor, by single gear reduction, the pinion on the starting motor shaft being on a feather-key. The motor can thus be cut out of service by a hand lever after the large motor is up to synchronism and connected electrically to its circuit. The new mill has been also provided with a steam plant, to be put into service quickly if anything happens to the electric power supply. Thus the 550-hp., 88-revolution, two-bearing synchronous motor which drives the 100-stamp battery and three tube mills, and the 260-hp., 88-revolution, two-bearing synchronous motor which drives the cyanide plant, were constructed with shafts extended on both ends to receive half couplings, the motors being installed between engines and shafts, to be driven with all shafts in each set in line. A flexible coupling of rubber cylinder type connects motor and driven shaft, and a special semi-flexible form of key-plate coupling connects motor and engine shaft. Normally the bolts and the middle or key section of the latter couplings are left out when the motors are driving. In 15 minutes or less they can be put into place, thus allowing the engines to drive the shafts, revolving the rotors of the motors at the same time. This has thus far, however, not been found satisfactory. Concerning the three tube mills at El Oro, the author remarks that smaller mills were adopted in the new installation, as they had proved to give the most satisfaction and least trouble in mill No. 1.

The "Danish" pebbles used cost $36.80 per ton at the mill, and the average pebble consumption is given as 5 pounds per ton of sand. The lining is composed of cast iron strips, the consumption being roughly ½ pound per ton of sand. The author states that for the slimes treatment the cost for steam drive was $0.90 per ton, while it is $0.50 at present for electric power.

Costs.

The Cost of Smelting Copper Ore.—In an article in the *Engineering and Mining Journal*, Sept. 1, G. F. Beardsley discusses in general the factors which go to make up the cost of copper smelting, so as to give a basis for figuring upon the question whether it pays to work a given low grade ore deposit. The first example he gives applies to the Mt. Lyell mine, of Tasmania, the ore of which is an exceptionally pure pyrites, which contains about 2.3 per cent of copper, 2 ounces of silver, and $1.40 gold. The average composition of the ore is: Iron 40 per cent, sulphur 46 per cent, silica 5 per cent, baryta 2 per cent, alumina 2 per cent. The ore is treated by pyritic smelting. The first product is an 8 to 10 per cent copper matte, which is concentrated by a second smelting to converter matte containing 35 to 40 per cent of copper. The deposit is worked by the open cut system, and therefore an item, "removal of overburden," is included in the following cost sheet. The item of smelting includes labor, coke, fluxes, power, heating blast, stores, water supply, flue dust, briquetting, laboratory and matte concentration. Labor is $5.44 per 24 hours, and coke $12.50 per ton. The cost of treatment per ton of 2,000 pounds is as follows: Mining $0.4938, or 10.1 per cent; removal of overburden* $0.4321, or 8.8 per cent; delivery of ore to works* $0.2619, or 5.3 per cent; sampling* $0.0393, or 0.8 per cent; smelting $2.3664, or 48.7 per cent; converting $0.3442, or 7.0 per cent; supervision $0.1259, or 2.6 per cent; disposal charges $0.7993, or 16.4 per cent; a total of $4.8638, or 99.7 per cent. The items marked with an asterisk * are those usually overlooked by the casual figures. Taxes, legal and administrative, are not included. The second example given by the author applies to the Tennessee Copper Co., where the ore is a pyrrhotite, containing about 2½ per cent of copper and traces of the precious metals. The analysis of the ore of the principal mine is as follows: Iron 38 per cent, sulphur 29 per cent, silica 11 per cent, lime 5.5 per cent, alumina 6 per cent, magnesia 1 per cent, maganese oxide 0.75 per cent. The ore is treated by pyritic smelting. Labor is $2.60 per 24 hours, and coke $3.50 to $6.00 per ton. The cost sheet for the treatment of a ton of 2,000 pounds is as follows: Mines development* $0.1712, or 5.6 per cent; mining, hoisting, etc., $0.6861, or 22.5 per cent; crushing and sorting* $0.1060, or 3.5 per cent; railway, delivery at works,* $0.1279, or 4.2 per cent; smelting $1.2450, or 40.9 per cent; converting $0.2313, or 7.6 per cent; engineering and laboratory* $0.0534, or 1.7 per cent; freight, insurance and selling expenses $0.2366, or 7.8 per cent; taxes and general expenses* $0.1809, or 5.9 per cent; a total of $3.0384, or 99.7 per cent. The third example refers to an ore in Mexico, 50 miles from a railway, the oré to be practically the same as in the first example, as far as gold, silver and the other constituents are concerned, and the method of treatment the same. With labor at $1.50, gold, per 24 hours, and coke at $30.00, gold, per ton, the cost of treatment per ton of 2,000 pounds would be as follows: Mine development* $0.40, or 8.6 per cent; mining* $1.00, or 21.4 per cent; delivery to works* $0.10, or 2.1 per cent; smelting $2.03, or 43.5 per cent; converting* $0.14, or 3 per cent; engineering and laboratory* $0.10, or 2.1 per cent; freight and disposal charges $0.89, or 19.1 per cent; a total of $4.66, or 99.8 per cent. The freight on blister copper is $30.00, gold, per ton. The disposal charges in these various cost sheets amount to 10.4, 7.8 and 19.1 per cent respectively. As the refining charges are rarely constant, the ratio between them is roughly in proportion to the amount that has to be paid for freight on the blister copper from the works to the refinery or market. Roughly, mining and smelting costs vary in proportion to the ordinary labor cost; the establishment of the correct cost of a few items will give a fair approximation of the whole; for the reason that, with the exception of coke, coal and stores, the mine and smelter consumes practically no raw material costing cash, but only material costing labor.

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

Electric Furnace.

Iron Reduction in the Electric Furnace.—C. S. Bradley, 829,907, Aug. 28. Application filed July 31, 1903.

Lead and copper can be reduced by fusing together suitable proportions of their oxides and sulphides, the sulphur and oxygen combining to form sulphur dioxide, which passes off. If we want to employ the corresponding reaction in the case of iron, it is to be considered that the reaction between iron sulphide and iron oxide is endothermic and requires a large amount of heat. For this reason it is proposed to employ an electric furnace. As oxide, hematite or magnetite may be used; the carbonate may replace the oxide, since at the temperature in the electric furnace the carbonic acid gas passes off, leaving an oxide. As sulphide, pyrites or other sulphide ore or waste products may be used. The silica or other gangue associated with the ores is converted to a slag and the coalescence of the reduced metal facilitated by the addition of a suitable fluxing material, like limestone. The following example is given: Crushed hematite and pyrites are mixed in the proportion of, say two to one by weight, together with a flux, and raised in an electric furnace to a temperature considerably higher than is attained in the ordinary iron blast furnace, "probably in the neighborhood of 3,000° to 4,000° C., and possibly higher." After the mass has reached a molten condition, the current is increased, thereby raising the temperature. When the temperature of the reaction is attained the charge boils violently with copious evolution of fumes of SO_2, which pass off and may be collected. The iron collects at the bottom. The following two equations are given as representing the reactions:

$$FeS_2 + \text{heat} = FeS + S$$

$$3FeS + 2Fe_2O_3 = 7Fe + 3SO_2$$

The process is said to be also applicable to the production of alloys, like chrome-iron by melting an oxide of chromium with a sulphide of iron.

Crucible for Induction Furnace.—E. A. Colby, 830,208. Sept. 4. Application filed Nov. 23, 1905.

To facilitate charging and discharging of the crucible, the crucible opening is enlarged by making its walls not truly cylindrical, the chamber surface of one wall being vertical and cylindrical, that of the other wall frusto-conical. In order to protect the crucible from rupture, due to the compressive strain of the material solidifying in the crucible, a lining is used of any refractory material which is more friable, and hence more easily crushed than the solid refractory material of the crucible itself, or if the lining is of the same refractory material as the crucible, it is in some way made less dense or less resistant to the compressive strain.

Electrolytic Processes.

Electrolytic Production of Copper.—J. A. W. Borchers, P. R. Franke, F. E. Günther, 830,639, Sept. 11. Application filed Sept. 21, 1906.

The object is the direct electrolytic treatment of copper matter. The inventors refer to former commercial trials as follows:

"In the experiments at Stolberg it was discovered that in the electrolysis in acid solutions of copper sulphate of the kinds of copper matte formerly employed by the cooperation of ferric sulphate, $Fe_2(SO_4)_3$, or of SO_3 ions from the sulphides forming the anode mass, sulphur was deposited on the surface of the anode and the metals passed into the solution at first readily, but subsequently only with the consumption of considerable electromotive force the copper was transferred from said anode onto the cathode. Now, we have discovered that the reasons

for this are as follows: To overcome the affinity between Cu_2S and FeS and other sulphides requires of itself a larger expenditure of power than was allowed for in the calculations of previous experimenters. The amount per unit of volume and the mechanical nature of the sulphur that is separated out of FeS (which is the metallic sulphide occurring in greater part as a subsidiary constituent in copper matte) and that of the sulphur separated out of Cu_2S are materially different, as a glance at the values of the molecular and atomic weights will show:

1. FeS (88) = Fe (56) + S (32).
2. Cu_2S (158) = Cu_2 (126) + S (32).

Thus for the same residual amount of sulphur there disappear in the process of solution fifty-six parts, by weight, of metal in the first case, and 126 parts, by weight, of metal in the second case—that is to say, one atom of metal in the first case and two atoms of metal in the second case. The greater the percentage of contained FeS the greater the quantity and the denser the nature of the sulphur that remains behind per unit of volume on the anode. The greater the amount of sulphur and the more densely it is deposited the more rapidly does the mechanical resistance against the access of the active constituents of the electrolyte to the anode mass which is still to be dissolved increase. The more slowly therefore the products of electrolysis formed on the still active surface of the anode are able to pass away so much easier substances which give rise to disturbing or costly electromotive counteractions are produced underneath the covering of sulphur, which prevents the equalization of concentration."

To overcome these disadvantages the inventors melt the copper matte to a degree of concentration of approximately 78 to 80 per cent of copper by means of the well-known reverberatory or converter furnaces, cast the mass into plates and electrolyze the latter as anodes in an electrolyte compound of acidulated aqueous solution of copper sulphate opposite to pure copper sheets as cathodes. Under these conditions the electrical pressure in the bath between anode and cathode is said to remain generally below 1.0 volt at ordinary temperature and with customary circulation of the electrolyte even after layers of sulphur of some thickness have been deposited. Whether and how often the layer of sulphur must be scraped off from the anode during electrolysis will depend chiefly on the local conditions. The sulphur may thus be recovered.

Electrodeposition of Metals; Copper Refining.—J. A. Nussbaum, 832,024, Sept. 25. Application filed Nov. 28, 1905. (Assigned to Siemens & Halske A. G.)

The object is to get metallic deposits in thick, dense, coherent layers, and to prevent objectionable crystal formations. The latter are said to be due to the fact that after the setting free of the metal ions the metallic separation does not at once follow, but an unstable intermediate form still remains for a short time in solution, from which metal deposits first here and there, where the conditions favorable to its separation exist—such, for instance, at those places where crystals already exist, which crystals will, therefore, grow larger at the expense of adjacent portions of the cathode, taking on a form of needles, foliations, etc. "If conditions be established in the electrolyte which hinder or prevent the changing of place of the unstable intermediate form before the separation as coherent metal, these will also restrain or prevent the formation of dendrites. This advantage will be secured by diminishing the movability of the electrolyte in the boundary zone at the cathode surface." For this purpose a slimy or colloidal substances of that class is added, which has the property of wandering to the cathode. In this way an intermediary colloidal layer is formed at those places on the cathode where crystal points tend to grow by the local shifting of the metal separation, which causes at these places a strong current density, as a result of which an increased traveling of the colloids to those places occurs. "The colloidal layer acts as a diaphragm, since it checks diffusion and circulation. In the progress of the electrolysis these deposition centers shift from place to place, so that the process may be explained as a fluctuating diaphragm on the cathode." As particularly effective colloids are mentioned the seed mucilages (linseed, for example), roots or bulb mucilage (salep), vegetable gums, glutinous plant saps, albumen and their hydrolization and transformation products of animal and vegetable origin (glutin, condrin, mucin, etc.). It is claimed that by this method one is enabled to get good metal deposits from any salt solution, so that one is no longer restricted to certain salts, which are usually expensive. As an example, copper refining is mentioned, where copper sulphate is used, "notwithstanding that with the same current cuprous salts give double the yield of copper, since the electrochemical equivalent of the copper therein is twice as great. The reason why the cuprous salts—for example, copper chloride—have not been employed, is that the metal separates therefrom in present known processes in the form of crystals—that is, in a form most unsuited for technical purposes. By the employment of the above invention, however, a solid coherent crystalline deposit is obtained from a solution of copper chloride and sodium chloride. In this case it is sufficient to add to the solution about 0.2 per cent of a colloid which has the property of wandering to the cathode, for example, flax, or linseed mucilage or salep mucilage. The current density may be the same as in the present known process of copper refining, and also the content of the solution in copper may be the same as ordinarily employed."

Electrolytic Precipitation of Copper.—B. Comba, 820,555, May 15. Application filed Sept. 14, 1903.

The object is to precipitate copper from acid liquors containing it; for example, a solution of copper sulphate obtained by lixiviation of roasted copper pyrites or iron pyrites containing copper, by means of the substitution for the copper of another metal of less value, usually scrap-iron. This is done in a tank representing in principle a short-circuited double-fluid primary cell without diaphragm. Sheets of copper, or preferably lead, are suspended in the tank which is filled with the copper solution, and on these sheets the copper is deposited "in such a ductile condition as to be ready for rolling." On top of these sheets, and in electrical contact with them, a sheet of copper gauze is placed, upon which the scrap iron rests in a solution of slightly acidulated warm water. On account of its lower specific gravity, this solution remains on top above the heavier copper solution. The plane separating the acidulated water and the copper liquor gradually drops as the liquor deposits its copper and loses its color. Fresh copper liquor is then automatically introduced at the bottom.

Electrolysis of Zinc Sulphate.—V. Engelhardt, 831,843, Sept. 25. Application filed Sept. 14, 1904. (Assigned to Siemens & Halske A. G.)

The object is the production of a crystalline, dense, smooth, electrolytic zinc deposit, free from sponginess and roughness or irregularities of surface, by electrolysis of a zinc sulphate solution with insoluble anodes. This is accomplished by using an anodic current density of from twenty to fifty times that at the cathode. "Thus for practically obtainable zinc concentrations in which a current density at the cathode of about 150 amps. per square meter of the cathode surface is employed, the current density at the anode either for the whole surface or at single places of the same would be from 3,000 to 7,500 amps. per square meter of the surface." Either an anode of smaller dimensions is used or an anode of a surface equal to the cathode, but reinforced at several places, as, for example, by bands, ribs or wire insertions, so that at these places a high conductivity is produced, and therefore a higher current density obtained. The possibility of using an anode of very small dimensions permits the employment of platinum, which is advantageous, since it cannot foul the solution. The beneficial

effects of a high anodic current density are attributed to the fact that "the formation of ozone and super-sulphuric acid is materially promoted, and these have an oxidizing effect upon the cathode or zinc hydrogen."

Diaphragm.—I. L. Roberts, 831,474, Sept. 18. Application filed Dec. 23, 1904. (Assigned to Roberts Chemical Co.)

In Fig. 1 the cut or basket 1 of wire gauze is strengthened at top and bottom by metal rings 2, 3. Inside of the basket is a cloth bag or lining 4, within which is a layer of the "plastic or gelatinous material" 5. In setting up the cell, a layer 6 of the plastic material is first placed on the bottom of the cloth-lined basket, and then a disc 7 of a composition of asbestos and magnesium silicate (known commercially as "transite," "magnesia building lumber," etc.) is pressed upon the viscous material until the two are in close contact, leaving the gelatinous or viscous layer of the desired thickness, generally equal to or slightly greater than the space which separates the edge of the disc from the sides of the bag. On the disc is now placed a cylinder or tube 8, made of the same asbestos-magnesia composition, and the space between the cylinder and the bag is then evenly packed with plastic material.

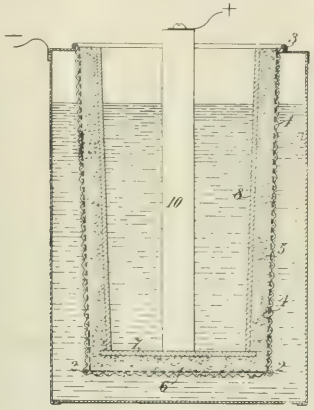

FIG. 1.—DIAPHRAGM.

Water Purifier.—F. R. Hinkson, 831,434, Sept. 18. Application filed Oct. 7, 1905.

"An electrolytic device for purifiers, comprising a plurality of closed compartments arranged in a vertical series, and adapted to successively receive the inflow-current of the fluid to be purified, said compartments being separated by bipolar electrodes and terminal electrodes adapted for connection to a source of current."

STORAGE BATTERIES.

Storage Battery Plate.—T. A. Edison, 831,269, Sept. 18. Application filed March 5, 1903.

In the well-known construction of the Edison battery the active nickel and iron masses are contained in pockets, and are maintained therein under pressure by means of closing plates, secured in place by overturning the metal around the edges of the pockets over upon the closing plates. The pockets, as well as the closing plates, are perforated. The plates are made of thick material (0.02 inch). For the purpose of economy in manufacture the perforations formed in the integral receptacles as well as in the closing plates are quite large, and the active material is prevented from exuding through these perforations by interposing very thin (0.0025 inch) sheets of nickel-plated material, and which are formed with extremely small perforations through which the active material cannot pass.

Chemical Effects in Gases.—D. R. Lovejoy, 829,876 and 829,877. Aug. 28. Application filed Jan. 27, 1906. (Assigned to Atmospheric Product Co.)

If union or chemical action of gases (the combination of oxygen and nitrogen) is to be accomplished by the agency of electric arcs, it is recommended to first subject the gases individually to the action of electrodes charged to a high electric potential, "in such a manner that the respective molecules of the two gases shall be given an electric charge of high potential, the molecules of one gas being given a positive charge, and the molecules of the other gas a negative charge previous to subjecting the gases, after mixing the same, to the action of electric arcs." (See also the patent abstracted on page 371 of our September issue.) In Fig. 2 the two gases first pass through the electrifying chambers 53 and 54, containing electrodes 55 and 56, charged to high potentials of opposite polarity from 57. The electrostatically charged gases are then passed into chamber 1, where they are subjected to arc discharges between the fixed electrodes 2 and the movable electrodes 4; 3 is an exhaust fan. If equal volumes of oxygen and nitrogen are treated by this process, the electrodes 55 and 56 are charged to about 50,000 volts, while the limits of voltage maintained between the arc terminals 2 and 4 may be about 10,000 volts; for instance, with a current of 0.1 amp.

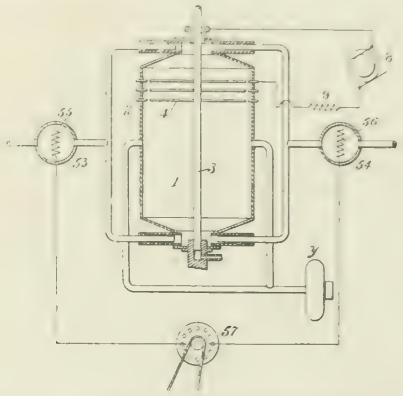

FIG. 2.—APPARATUS FOR TREATMENT OF GASES BY ELECTRIC DISCHARGES.

Production of Ammonia and Caustic Alkali.—G. F. Cassel, 830,209, Sept. 4. Application filed Aug. 22, 1904.

Electric discharges through air produce compounds of nitrogen and oxygen. By absorbing them by lime or carbonate of lime, nitrate and nitrite of lime are formed. These salts are subsequently treated with an alkali salt—for example, sulphate or chloride of an alkali, whereby a corresponding quantity of nitrate or nitrite of alkali is produced, which is subsequently subjected to electrolytic treatment with insoluble electrodes without diaphragm. In this electrolysis ammonia is produced, which passes off and is collected. The reduction, however, does not yield any appreciable output unless the solution contains a sufficient quantity of nitrites. In case a sufficient quantity of nitrites must be supplied, either by adding a suitable quantity of nitrites or a small quantity of a soluble salt of lead, for example, 0.5 per cent of nitrate of lead. These additions may be dispensed with, however, if a small cathodic current density is used, for example, not more than [illegible] per square centimeter.

On account of limitations of space a number of abstracts of patents had to be reserved for our next issue.

RECENT METALLURGICAL PATENTS.

Zinc.

Retort Construction.—To prevent the destruction of the bottoms in the retorts in zinc-distillation processes, R. Fiesing (818,070, April 17, 1906) constructs the retort with a false bottom, affording a protective shield and a ventilating passage beneath the floor proper of the retort. This floor is further provided with longitudinal ribs, or ridges interiorly of the ventilating passage, designed to stiffen or strengthen the structure, and also increase the radiating surface for cooling the floor of the retort to avoid its deterioration or injury.

Refining Zinc.—J. Callman and R. Bormann (827,418, July 31) patent a continuous process for zinc refining, the characteristic feature being that the zinc is distilled while in a liquid condition and flowing in a thin stream through a pipe, which is heated by a regeneration method and fitted with discharge pipes for the zinc vapor. A pipe of refractory material is connected with a vessel containing the zinc to be refined in a molten condition. This pipe is slightly inclined, and opens either into a vessel intended for the reception of the impurities melted during the distilling process or into a retort, which is also somewhat inclined and heated. While the outflow of the molten zinc from the above-mentioned vessel into the pipe is regulated by suitable appliances in such a way that the fluid zinc passes on through the pipe in a thin stream or layer and only flows into it as it evaporates, the pipe is best maintained by regenerative heating at such a temperature that at the in-flow part it is only slightly hotter than the melting point of zinc. On the other hand, at the mouth of the pipe the temperature should exceed the boiling point of zinc. The fluid zinc is accordingly gradually brought to the boiling point and evaporated as it passes through the pipe. While the vapor is being condensed in condensing chambers, the foreign metals, in a more or less molten condition, pass into the receiving vessel at the end of the said pipe intended for them and from which they can be taken away. A retort or crucible may be fixed between the pipe and the last-mentioned receiving vessel, the heat supply being regulated in such a way that the boiling point of zinc will only be reached in the retort, to which also the zinc will flow in only small quantities. While the zinc vapors escape through the neck of the retort, the foreign metals flow off through an aperture close to the bottom of the retort into the receiving vessel. In order to insure as far as possible a perfect distribution in the pipe and retort of the zinc to be distilled, the latter are filled with pieces of fire-resisting or inert materials, such as fire-brick and the like. Instead of fire-resisting or inert materials pieces of coal may advantageously be used, as these combine with the oxygen in the pipe and prevent the zinc from being oxidized. On the other hand, however, if dregs of zinc containing oxygen are treated by this process the coal will abstract the oxygen from these dregs and also from the oxygenous foreign metals, so that the zinc as well as the foreign metals are obtained in a pure metallic condition.

Flux for Extracting Metals from Ores.—A patent of A. Gutensohn (826,568, July 24) refers to the treatment of ores holding metals in silicious combinations and tailings mixed with silicious impurities. The salient feature is the use of a flux, consisting of 1 part of borate of manganese, 1 part ground anthracite coal and 2 or 3 parts of fluor spar, mixed with gas tar. The quantity of flux used is generally the estimated weight of the metal in the ore. It is stated that "this flux can be most advantageously used in the distillation of zinc out of ore, as it not only enables the metal to be obtained much quicker and at a lower heat than is now necessary, but prevents the metal's rapid destruction of the retorts." In the treatment of poor ores or low-grade material the quantity of the flux equals the estimated weight of the metal in the ore, and the metal when reduced will form into globules in the slag, which is then chilled in water with or without a small quantity of sulphuric acid or alkali; the chilled mass is afterward dried and ground to powder, when the globules, in consequence of their high specific gravity against the other material, can be easily recovered by any of the usual concentrating methods.

Silver and Gold.

Cyanide Process for Silver Extraction.—With reference to the developments at Guanajuato, Mexico, described in the article of Dr. J. W. Richards in our September issue, we noticed on page 371 two patents of F. J. Hobson on the treatment of silver ores by the cyanide process. A third patent (827,368, July 31) of the same inventor refers to the same subject, and may be best characterized by quoting claim 4: "As a selective solvent for silver when in combination with sulphur, a solution of potassium cyanide to which has been added mercurous chloride." and claim 6: "As a solvent for silver in the presence of sulphur, a solution of mercurous potassic cyanide, having the formula $KHgCy_2$, and of approximately 0.05 to 0.50 per cent in strength."

Cyanide Process.—C. W. Merrill (825,920, July 17) patents a modification of the cyanide process for treating that class of ores and tailings which contain reducing salts or minerals. After a preliminary treatment of the precious-metal-bearing material with a cyanogen-bearing solution, the latter is withdrawn from the interstitial spaces of the crushed material and a solution of chloride of calcium (which is obtainable, mixed with chloride of calcium, in the form of bleaching powder in sufficient quantities at low cost) is introduced into the interstitial spaces of the mass in the container, the effect of which is that such particles of reducing material as may be there present are oxidized. The hypochlorite solution is then displaced, and the particles within the container are subjected to further treatment with the cyanogen-bearing solution.

Slimes Treatment.—C. W. van Law (826,390, July 17), of Guanajuato, patents an apparatus for filtering slimes. A "slime-supply tank" contains a revolving agitating mechanism; from there the material flows to the "slime-pressure tank," until the latter has the desired amount of material stored therein to perform ore treating operation. The valve between both tanks is then closed, and air is admitted under pressure at the upper part of the slime-pressure tank, whereby the slimes are forced from the hopper-bottom of the slime-pressure tank through a pipe upwards into the hopper-bottom of the "filter tank." The latter is situated at a higher level, and contains a plurality of filtering frames of special construction, surrounded by wire-mesh and an enveloping canvass sack. These frames are placed side by side, but with free spaces between them, which spaces are gradually filled with cakes of the solid slime material, while the solution passes through the canvass into the frames and upwards through a pipe into a "recovery tank." There are also provided a "standard tank," containing cleansing water and a compressed air reservoir. A series of proper valves is, of course, provided in the pipes connecting the different tanks.

Separating Solution from Sand and Tailings.—W. H. Lomas, of Doornfontein, near Johannesburg, patents (825,331, July 10) an apparatus comprising an endless porous belt or band, which serves as the filtering medium upon which the solid matter is deposited and from which the separated solid matter is removed and collected. This belt or band is arranged to pass round a perforated suction-drum and round another perforated drum or pipe, from which a current of a fluid (gaseous or liquid) passes to displace or remove the deposited solid matter from the surface of the filtering belt. A suitable arrangement of guide and driving pulleys is provided for directing the belt over or round the greater portion of the circumference of the suction-drum. The suction drum is arranged to run in a trough or tank in which the mixture of liquid and solid is placed, and the interior of the drum is coupled up or placed in connection with a pump, for creating a partial vacuum in the drum, so that the liquid portion of the mixture is thereby drawn through the interstices of the filtering

belt into the drum, from which it may be removed by the same or an additional pump. In the interior of the suction-drum is arranged a damper, which serves for preventing the ingress of air or liquid around that portion of the circumference of the drum which is not covered by the belt.

Tin.

Detinning.—In our August issue, page 325, we commented on a new chemical process by Dr. Karl Goldschmidt and Mr. Josef Weber for detinning tin scrap. This process was, in its general features, like a process of Mr. F. von Kügelgen and Mr. G. O. Seward (see our September issue, page 344), since both processes use a solution of chlorine in an anhydrous liquid. Essentially different is a new process of Dr. Karl Goldschmidt and Mr. Josef Weber (831,223, Sept. 18), in which no anhydrous liquid is used. Its essential features consist in subjecting the tin scraps in a firmly compressed state in a closed vessel to the action of chlorine gas, and in altering the pressure in the vessel when the reaction of the chlorine on the scraps takes place. The chlorine gas is not employed in a pure state, but mixed with neutral or inert gases, like dry air. It is of importance to avoid in the process high temperatures in the mass of the tin scraps and to use the tin scraps in a well-dried state. The bundles are compressed, and of such a size that they can afterwards be directly used in steel works. They may conveniently be 40 centimeters wide, 60 centimeters long, and 8/14 centimeter high, the weight being about 50 to 60 kilograms. The packets are placed in baskets, which are put into the vessel. The process is so guided that the pressure in the vessel is increased, so that at the end of the operation the over-pressure—for instance, of one atmosphere above the ordinary pressure—is obtained. By the increased pressure of the vessel the chlorine gas comes in contact with all the surfaces of the tin scraps, so that even between the most closely packed surfaces of the tin scraps a sufficient action of the chlorine gas and formation of chloride of tin takes place. The iron waste remains perfectly detinned and with a fine, smooth gray surface. The process would seem to be of ideal simplicity for large-scale operation, when the mechanical difficulties of handling dry compressed chlorine gas are overcome. But it is known that such difficulties have been completely overcome in various European plants.

BOOK REVIEWS.

The Management of Electrical Machinery. By Prof. Francis B. Crocker, Ph. D., and Schuyler S. Wheeler, D. Sc. 223 pages, 131 illustrations. New York: D. Van Nostrand Co. Price, $1.00 net.

This excellent little book, which here appears in its second edition, is intended to give simple directions for the practical use and management of dynamos and motors. It is a thoroughly practical book, and it is remarkable what an enormous amount of useful information is here given in a comparatively small space.

The book is divided into five parts. The first deals with the selection and management of dynamo-electric machinery (principles of generators and motors, selection, installation, connections, operation). The second part deals with inspecting and testing (adjustment, friction, balance, noise, heating and sparking, measurement of resistance, voltage and current, speed and torque, power and efficiency). The third part, which should be greatly appreciated in practice, gives concise instructions as to the localization and remedy of troubles. The fourth and fifth part deal with constant-current (arc) generators and with railway motors, respectively.

Notes on Metallurgical Mill Construction. Edited by W. R. Ingalls. New York: *The Engineering and Mining Journal*. 256 pages. Price, $1.50.

The book, the price of which is remarkably low, is a reprint of a series of articles bearing upon the following seven general subjects: Brickwork and concrete, building construction, ore-crushing machinery, driers and drying, conveyors and elevators, disposal of tailings, miscellaneous.

The articles reprinted here have formerly appeared in *The Engineering and Mining Journal*, chiefly during the last three years. The book is, therefore, of a somewhat heterogeneous character, and in the nature of a scrapbook rather than a general complete treatise. But this has some decided advantages, especially since the individuality of the different authors has been preserved. A scrapbook is a most useful thing, if made up in the right way, and the editor of the present volume is to be congratulated on the excellent judgment he has shown in its preparation.

A New Canadian Custom Smelter.

By Martin W. Peschel.

During my presence in the northern parts of the provinces of Ontario and Quebec, I often spoke with mine owners and prospectors concerning the necessity and possibility of a smelting plant that would not only treat those exceptionally high-grade ores mined directly at and around Cobalt, but would also enable the miner of a more distant locality and mining a lower grade of ore, to dispose of his mine products at its market values without having to carry the burden of a freight rate of $9.00 per ton from Cobalt to New Jersey.

If one considers that the New Jersey smelters only treated the higher grade of ores, which were often penalized in an unjustified way, the poor and independent miner's outlook into the future could not be called very bright. Several schemes had been considered to overcome this great drawback, and a smelter was indeed planned. The reasons why such an enterprise, with such great prospects, did not realize, are unknown, although it was generally believed that the heavy treatment expenses of the cobalt-nickel ores would render profits too doubtful, while the amount of ore mined at the time was still too small to supply a smelter of even a moderate capacity sufficiently.

The latter reason for the failure of the erection of a reduction work might have been justified as far as the cobalt-silver ores of the township of Coleman are concerned. But as the Canadian mining regions of Ontario and Quebec are more or less in their infancy, and the mineral wealth of these provinces not confined to the township of Coleman, a broad-minded enterprising smelterman would have thought of the vast area these two provinces do cover on the maps of Canada.

There is beside those rich mines an abundance of calcspar-veins assaying from 5 to 200 ounces of silver per ton, and it is well known how easy those ores can be treated by simply adding them to the charge of a lead or copper furnace. Whether such ores would allow a freight rate of $9.00 per ton is another question, but they certainly could be disposed of if a smelter would be near at hand. Lately discoveries of big copper deposits with fair values of the precious metals have been made in the Temagami Forest Reserve, and further reports of discoveries of all kinds of ores are made continually.

On my return trip to North Bay I learned that a smelter at Front Lake, 3 miles distant from the town of North Bay, was in course of erection, but people had previously paid attention to various rumors which had not realized, and did not seem to take much stock in the "little native work" commenced outside of their town, and spoke of a "bluff." What I found at Front Lake, however, could no longer be called a bluff.

Long concrete foundations and walls had already been erected, with a large gang of men employed in grading, etc., for further buildings. The plan shown to me on the company's office spoke for itself, as well as the name of the designer, and that the Canadian Government fully realizes the necessity of

an independent custom smelter was proved by the support it has given from the very start to the project.

Mr. J. H. Brown is the organizer of the Montreal Reduction & Smelting Co., of Canada, Ltd., and the designer of the plant. All the members of the syndicate, with the exception of Mr. J. H. Brown himself, are prominent Canadians, and the attendance at the ceremony of the laying of the cornerstone, which took place on Sept. 10, showed clearly how highly the population of Canada is welcoming the new enterprise.

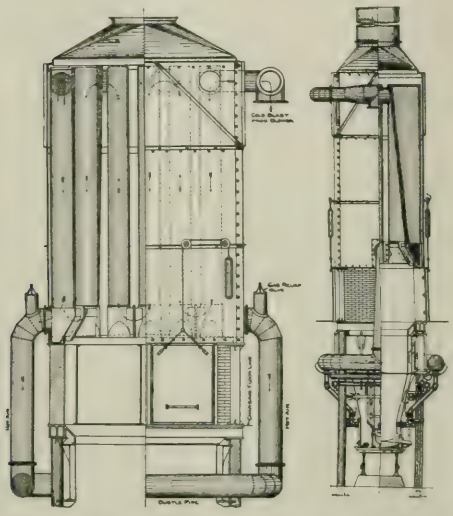

FIG. 1.—DIAGRAMMATICAL SKETCH OF GIROUX HOT-BLAST FURNACE.

There are plenty of men all over the world who can stumble over a rich ore vein, but there are only a few who can design and superintend a custom smelter of the kind they are in need of here, if the success hoped for shall not turn into failure.

It is intended to commence operations this year. The plant will have a daily capacity of 500 tons, and will treat all kinds of ores by concentrating, smelting, amalgamating, converting, cyaniding, etc., and will be up to date in its equipment.

The cornerstone was laid by the Minister of Mines of the Province of Ontario, amid the cheers of 3,000 onlookers, and in the presence of the Minister of Agriculture, Hon. Montieth, and the whole Board of Directors. The meeting was presided over by the Mayor of North Bay, who read an address, to which the Minister of Mines replied, pointing out the progress which the Western States had made since the erection of smelters. Mr. Leonard, M. P., and president of the company, presented then the deed to the company. Hon. Montieth spoke similarly, recalling the memory of Champlain, who had traveled through this part of the country on his voyages. Mr. Monk, M. P., made an eloquent speech, as the representative of the company, and Mr. J. H. Brown made the final address, stating that the plant would be the most up to date and complete.

It is to be hoped, according to the way the Canadian people and Government have supported the enterprise so far, that this plant will be to Eastern Canada what the great custom smelters are for the West. The facilities which it has, both by water and rail are excellent, and all conditions are in favor of the new enterprise, which, as a Canadian concern, but promoted and guided by an experienced American, will give employment to at least a thousand men.

Hot-Blast Smelting.

The National Metallurgical Co., of Matehuala, Mexico, is almost ready to put into service a new blast furnace, designed and built by the Traylor Engineering Co., of New York City. The furnace measures 42 inches by 100 inches inside the tuyeres, and will add 250 to 300 tons per day to the capacity of the plant.

One of the most interesting features of the construction of this furnace is that it is fitted with the Giroux hot-blast top. This top was invented by Joseph L. Giroux when he was manager of the United Verde Works, at Jerome, Ariz. This hot-blast system has been in use in the United Verde smelter for three years, and has there proved a great source of fuel economy, besides showing that the design is entirely successful in resisting the destructive effects of expansion and contraction, which has made other designs too expensive in maintenance. They have cost practically nothing for repairs. The tops are also in regular use at the works of the Giroux Consolidated Mines, in Ely, Nev. The top furnishes a blast heated to 400 to 500 degrees F.

Experience has shown that the new hot-blast top system gives a saving of 30 per cent of the fuel charge with some ores, while at the same time it increases the capacity of the furnace from 15 to 25 per cent. It allows the use of charges carrying from 2 to 4 per cent more silica, and the more uniform temperature seems to prevent, to a considerable extent, the crusting or building up of the walls of the furnace, so noticeable in ordinary practice.

FIG. 2.—END VIEW OF FURNACE WITHOUT HOUSING.

The Giroux hot-blast top is applied with advantage and economy to copper smelting furnaces where the process is largely oxidizing, and, as a result, the zone of heat is carried to a comparatively high line in the furnace.

The heat thus produced, instead of being wasted, is utilized for heating the blast. This not only makes a corresponding reduction in the amount and cost of the coke charge, but gives all the other advantages which are incident to the use

of the hot blast. For use where fuel is low in price, and a very hot blast is desirable, the Traylor Engineering Co. furnish a hot-blast stove, of a superior character, of its own design.

The Giroux hot blast top consists mechanically of two series of metal pipes arranged symmetrically around the inside of the top of the furnace, the lower ends of the pipes being just above the top of the charging floor. The vertical portion of the pipes is built of sheet steel. These are circular at the bottom where they join the return elbows, and oval and of larger section at the top. The material of the pipes makes them readily permeable to the heat, and their form gives them a large surface for absorbing the heat from the

Fig. 3.—Side view of furnace without housing.

furnace gases. This form also prevents dust from adhering to them and reducing their capacity for absorbing heat.

Near their tops the pipes are connected together in pairs sidewise by means of flanged cast-iron connections, and at the bottom each pair of pipes is connected to form a continuous series on each side of the furnace by means of cast-iron return elbows. These elbows have a V-shaped section, with one side of the V lying parallel with the side of the furnace top, and the other forming a slope for the hot furnace to impinge against.

Cast iron is used for the elbows because it is the best material to resist the destructive action of the impinging furnace gases. The form of the elbows insures the delivery of the hot gases, and its distribution along the sides of the upright pipes in the best manner to insure their giving up of their heat to the blast air passing through the pipes. The attachment is built so that it may be used on furnaces with either sheet metal or brick tops, but the use of the brick top is always recommended. The brick is best because it is a good non-conductor of heat and thus concentrates the heat of the furnace in the air pipes.

The blast from the blower enters the furnace at the top of the attachment at the two ends. It then circulates through the air-heating pipes in series, after traversing their length, passes down through the hot-air connection into the bustle pipe of the furnace, and from thence through the blow-pipes to the tuyeres. This top is mechanically perfect. There is no distortion produced by the heat as all of the heating pipes are entirely surrounded by the furnace gases, and the expansion is, therefore, uniform throughout their length. This prevents the distortion from heat which has made other arrangements intended for this purpose failures.

A similar hot blast attachment is now being constructed by the Traylor Engineering Co. for a furnace 36 inches by 288 inches, under construction for the Dominion Copper Co., of Canada. This furnace will be the largest ever built.

Fig. 1 shows diagrammatically the general arrangement of the hot blast furnace, while Figs. 2 to 4 are views of the hot-blast furnace for the Matchuala Works of the National Metallurgical Co. The photographs were taken in the erection yards at the works of the Traylor Engineering Co. in Allentown, Pa.

Fig. 2 is an end view without housing, showing the blast

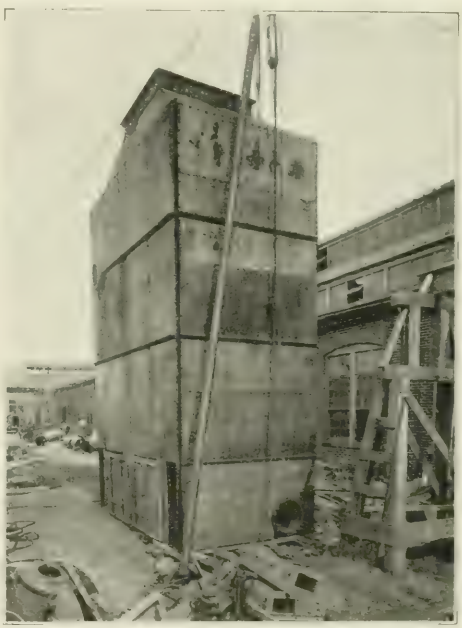

Fig. 4.—Furnace complete with housing.

connection at the upper left hand side of the top and bustle pipe connection at the lower right hand side.

Fig. 3 is a view without housing showing the furnace charging doors under one set of side pipes, and the V-shaped form of the cast-iron bottom connection for the pipes.

Fig. 4 is a view of the complete furnace with housing.

Lime Burning in Rotary Kilns.—The Pittsburg Reduction Co. will soon be burning lime in rotary gas-fired kilns. The capacity of the plant will be 350 tons in 24 hours. Before decided on rotary cylinders a trial was made in the New York Lime Co.'s plant at Natural Bridge, N. Y., which is the first plant to successfully burn lime in rotary kilns. The gas producer equipment is being supplied by the Morgan Construction Co., of New York. Besides 8 producers will be using 2 rotary kilns for burning lime and generating CO^2, 8 other producers are used for heating 6 rotary kilns for calcining magnesite.

Notes.

Historical Development of General Chemistry.—The July issue of the *School of Mines Quarterly* contains the concluding instalment of the series of lectures held by Prof. William Ostwald in the beginning of this year at Columbia University. This instalment contains the fifth lecture on equilibrium and chemical affinity and the sixth lecture on chemical dynamics, including catalysis.

The Goldschmidt Thermit Co. are about to vacate their present manufacturing premises at 179 Christopher Street, New York City, as they are insufficient for the largely increased business. They have bought ground at the corner of Cornelison and Bishop Streets, in Jersey City, and have erected there a large and commodious factory building, 165 x 75 feet, within easy reach of their down-town offices, and intend removing their manufacturing plant to the new location on or about Oct. 1.

Filter Presses.—Messrs. T. Shriver & Co., manufacturers of filter presses, have received one of the largest orders for presses ever given for installation at a single plant. The order is placed by the Pittsburg Reduction Co., and calls for forty-five large presses for use in their East St. Louis works.

Muspratt Laboratory.—The new laboratory of physical chemistry and electrochemistry, which has been presented to the University of Liverpool by Mr. E. K. Muspratt, will be opened by Sir William Ramsay, F. R. S., on Oct. 13, when a large number of English and Continental men of science will be present. Addresses will be made by Sir William Ramsay and Prof. Ostwald. A complete electrical equipment has been provided, including a 30-kw. motor alternator, a 30-kw. continuous-current motor generator, and a 10-kw. set for battery charging. The battery is composed of thirty-six Tudor cells. The new laboratory, which contains in all twenty-one rooms, is to be known as the Muspratt Laboratory of Physical aad Electrochemistry, and is under the charge of Prof. Donnan.

Attack of Glass Vessels by Chemical Solutions.—In last year's report of the German Reichsanstalt, reference is made to the suitability of quartz vessels for chemical operations, and the absorption of water by glass exposed to the action of the atmosphere and, particularly, of the sulphuric acid of accumulators. This absorption of water is accompanied by changes in the composition of the glass, consisting essentially in the loss of potash and soda; the water is absorbed by the silica, which undergoes molecular changes. When the glass is slowly heated most of the absorbed water is given off a little above 100° C., the rest at 500°, and the glass peels. When the heating is rapid the water escapes in very small bubbles between 400° and 500° C., and the glass does not peel, but turns white like porcelain. One kind of glass for accumulator containers had absorbed 5.8 per cent of water in fifteen years. For good accumulator glass the effect is not of importance.

Turbo Blowers.—In a paper recently presented at a meeting of the British Association for the Advancement of Science, by Gerald Stoney, the author described the development of the steam turbine for driving rotary air compressors of the turbine type, which are now being used largely for blowing blast furnaces. The advantages gained are light weight, small foundation, small consumption of oil and, above all, high economy of steam over the reciprocating types of blowing engines. The outfits described are generally for about 20,000 cubic feet of free air per minute and a pressure of 10 to 15 pounds per square inch. A slightly different type is made for about 30,000 cubic feet per minute, at about 1 pound per square inch pressure. These blowing equipments are being used in several large iron works for dealing with the waste gases from furnaces and for driving these gases through the recovering plant, etc., an important feature being that they do not clog with tar and other matters. Since it is nearly impossible to use economically low pressure steam at about atmospheric pressure in a reciprocating engine, the exhaust steam turbine becomes an important factor in those cases where there are non-condensing engines and other sources of exhaust steam. The American rights for the manufacture of Parsons turbine blowers, as well as Parsons land and marine turbines, have been secured by the Allis-Chalmers Co., of Milwaukee, and through an arrangement made with the English builders, the American company will have the benefit of the experience and data of the European experts.

Gold from the Sea.—With respect to this problem, which has been in the past a source of income to a few promoters, and only to them, the London *Electrician* writes as follows: "Recently nothing has been heard of the Electrolytic Marine Salts Co., U. S. A., which was floated for the purpose of extracting wealth in the form of gold from the sea. But in a recent issue of *L'Electrician*, Mr. A. Nodon returns to the subject, and gives his views as to how this desirable object might be attained. He proposes to use copper or lead cathodes, preferably lead on account of the expense, sheets 1 mm. in thickness being suitable. These electrodes would be surrounded by canvas or other suitable material, forming porous bags, in which any deposit not adhering to the electrodes would be collected. The anodes could be made of graphite, lead or cast iron. Of these, however, graphite would be expensive and lead would be attacked, so that gray cast iron would be the most suitable from the chemical point of view as well as that of cost. The electrolytic baths would be in the seashore, within reach of the tide, and would have a depth of 2 meters and a breadth of 10 meters. By having cement partitions every 40 meters, a battery of 100 such baths could be run together, and would furnish 150 grams of gold per day of 12 hours. During this time 3,000 cubic meters of water would be treated, and a current of 5,000 amps. at 2.5 volts would be required. Mr. Nodon puts the annual cost at $12,000 and the profit at $20,000, with a capital of $40,000, the value of the gold recovered being taken at 60 cents per gram, so as to allow of subsequent treatment."

American Electrochemical Society.—At the September meeting of the Board of Directors, Mr. H. J. Wolf, of Telluride, Col., was elected a member, while at the October meeting the names of the following gentlemen will come up for election: Messrs. Frank A. Decker, Albert J. Shinn and William P. Divine, all of Philadelphia, Pa.

Illinois State Geological Survey.—We have received Bulletins Nos. 1 and 2 of the Illinois State Geological Survey. Bulletin No. 1 is a geological map of Illinois by Dr. Stuart Weller; it is very neatly executed in different colors, and represents the best information available at this time, and should, therefore, serve as an excellent base for future work. In Bulletin No. 2 Prof. W. S. Blatchley gives a review of the petroleum industry of Southeastern Illinois.

Progress of Krupp Works.—According to Consul-General Guenther, the celebrated works of Krupp, at Essen, Magdeburg, Kiel, Annen, and at their ore and coal mines had, on April 1, 1900, in their employ 62,553 persons, of whom 5,065 were officials and clerks, against a total of 55,816 employees in the year before. The company's principal plant and accessories at Essen consumed as much water in said year as did the entire city of Dresden, which latter has a population of over 300,000 inhabitants. The gas made and consumed by the single cast steel plant at Essen exceeds the gas consumption of the city of Elberfeld. Besides this, the electric plant of these steel works supplies 1,051 arc lamps, 15,304 incandescent lamps and 763 electric motors. The average daily wage paid in 1905 to the workers in the cast steel plant was $1.22 per person, which is about 5½ cents more than was paid in the preceding year.

Portland Cement Industry.—The Portland cement industry in this country presents one of the most marvelous instances of growth on record, showing the amazing increase in the output in the United States from 42,000 barrels in 1880 to 35,000,

000 barrels in 1905, or over 800 times as much, whereas in pig iron production the output in 1905 was only about six times that of 1880. This marvelous growth of the cement industry, however, has in no way interfered with the growth of the iron industry. On the contrary, cement has come as an auxiliary to help maintain the vast building activity and prevent an iron and steel famine. It has also relieved the lumber situation through its use in many forms of construction where timber would otherwise have been essential.. . Many plants have sprung up within the last few years in various parts of this country, whose capacities run up into thousands of barrels daily. Contracts for cement-making machinery calling for an expenditure of hundreds of thousands of dollars are of frequent occurrence to the large cement machinery manufacturers. What is said to constitute the largest individual order ever placed for tube mills for the grinding of cement clinker is one recently placed by the United States Steel Corporation. This order calls for forty-seven tube mills, 5 feet in diameter by 22 feet in length. Twenty of these are to be installed in the plant of the Carnegie Steel Co., at Homestead, Pa., and twenty-seven are for an extension to the immense modern cement plant of the Illinois Steel Co., at Buffington, Ind. This entire order was awarded to the Power & Mining Machinery Co., Cudahy, Wis.

Gas Producers for Steel Plant.—Manufacturers have lately been stirred up by the largest contract for gas producers that has ever been given. Last week the Illinois Steel Co. closed for 140 large producers with gas flues and Dyblie reversing valves, to operate the twenty-eight open-hearth furnaces to be erected at Gary, Ind., constituting the largest steel plant in the world. For the past four months eight of the leading manufacturers in that line of business have been engaged in this contest, because of its great importance, not only on account of the large amount of work, but principally on account of the standing to be obtained by the successful type. A committee of engineers was appointed by the Steel Corporation to visit the various plants, investigate the claims and note actual results. The first approach to a decision came when the committee visited the plant of the Lackawanna Steel Co., at Buffalo, where they saw several of the leading types of gas producer in operation. The open-hearth furnaces, which are fired with gas from continuous gas producers having mechanical feeds, at that plant, are now turning out more steel with less coal consumption than any other plant in the world. When the order was placed the award fell to the Morgan Construction Co., of Worcester, Mass., who were represented in the contest by Mr. E. A. W. Jefferies, of 40 Exchange Place, New York City. This makes the fifth order for gas producers that the Illinois Steel Co. have awarded to the Morgan Construction Co. This company has also obtained a remarkable recognition of the merit of their work in England and continental Europe, where about 100 Morgan continuous gas producers have been sold during the past twelve months, this being the only American gas producer which has successfully invaded that field.

Tube Mill in Cyanide Practice.—Based on the results obtained from the operation of an experimental plant of half a ton capacity at Bodie, Cal., the Standard Mine designed a plant to which all the tailing from the Standard Mill might be applied in accordance with the Moore methods of treatment (which was described and illustrated in our Vol. III., p. 257). The new plant, which was placed in regular operation in 1905, has fulfilled the expectations of those who were responsible for its design. The ore at Bodie consists of quartz, iron oxides and clay. The gold is partly coarse and partly very fine, which amalgamates badly. The ore is difficult to treat in spite of the absence of all minerals ordinarily classed as deleterious. An Allis-Chalmers 5-foot x 22-foot tube mill does the regrinding. The under flow of two cone separators furnishes the feed. It contains the coarse stuff and a certain portion of adhering slime, and passes to the tube mill mixed with sand from the pond fed into the stream automatically. The outflow from the mill is returned to the cones. The tube mill, which makes 20 r. p. m., is charged with 12 tons of [illegible] flints. Wrought iron plates are used for liners. They are [illegible] inch x 8 inches, and are cut into 7-foot and 15-foot lengths, bolted through the shell. The power for the mill is approximately 50 hp. when running, and 100 hp. at starting. The grinding capacity is placed at 60 tons of sand per 24 hours.

Personal.

Dr. Karl Goldschmidt, partner of the firm of Theodor Goldschmidt, Essen-Ruhr, Germany, equally distinguished by their decisive pioneer work in detinning as by the aluminothermic process, is at present in this country. Some notes on the latest work of Dr. Karl Goldschmidt on detinning by means of chlorine gas, will be found in this issue, under the head of "Recent Metallurgical Patents."

Sir William Henry Perkin, the famous inventor of mauve, arrived in this country on Sept. 30, to attend the celebration to be given by the chemists of the United States in his honor on the fiftieth anniversary of the coal-tar industry. The program of the celebration will be found on another page. The distinguished chemist will be the guest of Prof. C. F. Chandler during his stay here. On Oct 1, Sir William will be the guest of Mr. George F. Kunz, the gem expert of Tiffany & Co. He will be escorted through various places of interest to chemists and the Museum of Art, the New York Botanical Garden and the Zoological Park will also be visited. On Oct. 2 the host will be Comptroller Metz, who is interested in one of the great German coal-tar firms. On Oct. 4, Sir William and his family will be the guests of Mr. H. H. Rogers, on whose yacht they will take a trip up the Hudson, while on Oct. 5 Sir William will visit the Laurel Hill Chemical Works, on Long Island, as the guest of Dr. William H. Nichols, President of the General Chemical Co. and the Nichols Copper Co., and Past-President of the Society of Chemical Industry. According to an interview, published in the New York *Times*, the distinguished inventor made the following statements to the reporter: "I was in the laboratory of the German chemist, A. W. Hofmann, when I discovered mauve. I was then 18. While in an experiment to find quinine I failed, and was about to throw a certain black residue away when I thought it might be interesting. The solution of it resulted in a strangely beautiful color. You know the rest. * * * * It is impossible, of course, for America to be ahead at the present moment in the matter of organic chemistry, in which I am especially interested. Germany has so many advantages, primarily cheaper labor. Besides that, German chemical manufacturers have established laboratories in which they place university graduates, whose business is nothing but to experiment and discover new things. This country, however, has greater opportunities for the future, as it is building its genius upon the genius of European centuries." When told that in the recent pure-food discussion the question of the possible injuriousness of aniline dyes in foods had been raised, he replied, "I would not like to take sides in that matter. It is probable that there have been abuses of the uses of aniline dyes in foodstuffs, in fact, I know there have been. But this is certain, that the amount of aniline dye necessary to color a food is so minute that if the same quantity of strychnine were used, it would be equally harmless."

Digest of U. S. Patents.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

Ozone.

No. 574,341, Dec. 29, 1896, M. Pridham, Philadelphia, Pa. Oxygen is cooled by passing it through [illegible] coil, sur

rounded by a refrigerating liquid, such as ammonium nitrate and water, or by compressing and expanding it, and then ozonized. The ozonizer comprises concentric tubes carrying electrode coatings. The outer tube is of glass, closed at the ends with caps having gas inlet and outlet pipes. The inner tube may be of a dielectric, but is preferably of metal, closed at the ends. A spray pipe for the cooling liquid enters through a central stuffing-box at one end and a waste pipe leads from the other end. The outer surfaces of both the inner and outer tubes are coated with a rough or granular layer of material adapted to produce a brush-discharge, such as powdered carbon or lead dust, held by an adhesive non-conductor. The ozonizer is preferably worked at temperatures below zero degree F.

No. 577,523, Feb. 23, 1897, G. J. Anderson and J. C. Dittrich, Brooklyn, N. Y.

Ozonizes oxygen or air. The oxygen, produced by heating potassium chlorate and manganese dioxide, is passed through wash-bottles containing solutions of caustic alkali, sodium hyposulfite and sulfuric acid, to remove carbon dioxide, chlorine and water. The ozonized oxygen is employed to saturate distilled and sterilized water for medical use, or vegetable or mineral fats or oils. The ozonized air is passed into an alkaline solution or water, to produce nitric salts or acid. The ozonizer consists of four Siemens ozone tubes, having a single supply and discharge pipe, supported in a glass jar. The inner tube of each ozonizer is filled with water, into which dips one terminal. The jar is also filled with liquid.

No. 577,030, Feb. 23, 1897, E. Andreoli, London, England.

The electrodes are vertically arranged in horizontal series, within a casing having a bottom inlet and a lateral top outlet, resting on a slitted false bottom. The electrodes consist of thin metallic boxes spaced apart from intermediate dielectric sheets by means of glass blocks. The opposing faces of the boxes may be provided with serrated metallic strips. Water is circulated through the boxes by separate reservoirs and supply and discharge pipes. Air entering the casing-inlet is distributed to the slits in the bottom by a baffle-plate or fan.

No. 580,244, April 6, 1897, C. J. Yarnold, London, England.

The air to be ozonized is purified by passing it through a caustic alkali bath, then dried and cooled to 32° F. The ozonized gas is passed through a vessel containing pumice and water, to remove the nitrogen oxides. The ozonizer has electrodes, each consisting of two sheets of corrugated glass inclosing a layer of silver foil. The foil sheets have tabs, which are connected to metal rods in insulating sleeves. The electrodes are supported vertically in horizontal series on the floor of a casing, their lower edges being separated by ribs and their upper edges by depending fingers. The casing has a hinged cover with a window, and expansion chambers at each end, receiving the supply and discharge pipes. A thermometer enters each expansion chamber.

No. 587,770, Aug. 10, 1897, N. Vander Sleen and A. Schmeller, Alfen-Oudshoorn, Netherlands.

Employs several ozonizers arranged in line, connected by intermediate parallel cooling pipes surrounded by water. The electrode plates of each ozonizer are hung from frames above. The electric potential is regulated by a resistance column of distilled water, glycerine or dilute alcohol, interposed in the circuit of each electrode. For a potential of 60,000 volts and a distance of 30 millimeters between the electrodes the resistance should be 20 megohms.

No. 596,036, Jan. 4, 1898, F. K. Irving, Passaic, N. J.

Generates ozone by electrolyzing a solution of copper or zinc nitrate with electrodes of an alloy of lead and antimony. The electrolyte cell is provided with an inhaling tube.

No. 597,617, Jan. 18, 1898, A. Vercoy, Paris, France.

The ozonizer comprises a horizontal rectangular sheet of copper, carried by a glass table and a rectangular sheet of silvered glass, distance pieces being interposed. Glass rods may be arranged between the two plates to provide a spiral or zigzag passage, air entering at the edge and the ozone escaping through a hole at the center. The slate and copper may be above the glass, the slate then being recessed to allow a current of water to circulate over it. Or the silvered glass may be arranged over the copper and slate. The ozonizer may be cooled by the expansion of compressed air. The potential difference is 5,000 to 12,000 volts, a direct current being employed.

No. 599,455, Feb. 22, 1898, M. Otto, Paris, France.

Employs electrodes, one of which rotates. The electrodes may be flat parallel discs or concentric tubes, one of which is provided with a series of discharging points or brushes of platinum or aluminium. The outer electrode or electrodes have re-entrant portions provided with external heat-radiating wings. In the disc form the electrodes are vertical, the intermediate one being carried by a horizontal shaft. The discharge points are arranged in sectors. The air is introduced at the center through a rose and passes radially. In the form using concentric electrodes, brushes projecting from the surface of the inner cylinder are arranged in two helical series partially separated by a disc projecting from the outer electrode casing. The apparatus may be of cast or sheet iron, or of a non-conductor, such as wood or porcelain covered with metal. It may be protected by a coating of platinum, gold, varnish or enamel. The potential difference with an electrode distance of 3 centimeters may be 18,000 volts. The apparatus may be used for the production of hydrocyanic acid from a mixture of nitrogen and acetylene.

No. 607,007, July 12, 1898, E. Andreoli, London, England.

The electrodes are opposing grids bearing points, or grids facing flat electrodes. The grids consist of pieces of serrated wire, separated by washers and bolted together. The flat electrodes are boards of hardwood, covered with thin sheets of copper, tin-plate, aluminium or an electro-plated metal. They may be coated with paint or enamel. One or two sheets of glass may be interposed between each pair of electrodes, or, in the case of flat electrodes, their face may be coated with pieces of mica or a layer of powdered mica held by an adhesive. The glass plates are placed in grooves in the casing. The air in larger rooms may be ozonized by using a number of separate ozonizers without casings. Concentrated ozone may be produced by employing several ozonizers in series.

No. 614,500, Nov. 22, 1898, A. S. Ramage, Cleveland, Ohio.

The electrodes, arranged horizontally in a casing having end pipes, each consist of a sheet of corrugated or pitted tin foil, cemented between two plates of glass by shellac. The electrodes are spaced by marginal glass strips, and are arranged in a casing consisting of a wood box having an inner layer of plaster of paris faced with glass.

No. 632,301, Sept. 5, 1899, H. Abraham and L. Marmier, Paris, France.

The electrodes are circular, vertical, metallic discs, with an inert coating of paraffin, silicate, red lead or enamel. The discs are hollow and provided with pipes for circulating water through them. To avoid short circuiting, the water is supplied and discharged in separate drops, or is intermittently passed from one insulated reservoir to another by siphons. The electrodes are centrally apertured and receive a perforated ozone discharge pipe. Each electrode is hung from an insulating cross-bar above. Dielectric plates between the electrodes are supported below. The electrodes may be adjustably spaced by screws. The two armatures of the ozone generator are shunted by a discharger consisting of spaced metallic balls. A strong current of air or steam is preferably injected between the balls. An auxiliary ozonizer may be interposed in shunt between the discharger and the main ozonizer, working only during the passage of the spark in the discharger. The power of this auxiliary generator may be increased by a small auxiliary transformer.

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J. W. RICHARDS, PH. D.....*President*
E. F. ROEBER, PH. D.....*Editor*

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Electrochemical Processes as Central Station Load Equalizers

At the Cornell meeting of the American Electrochemical Society, Mr. E. A. Sperry presented a paper on the possibility of using electrochemical processes as station load equalizers. The situation is briefly this. Central station engineers recognize the importance of filling out the valleys of their load curve, and to do this they could sell the current at very low rates, since any receipts would be "velvet." On the other hand, for the successful operation of any electrochemical and electrometallurgical processes, the first requisite is cheap electric power. So far, the needs of the central station engineer and the electrochemist meet each other. But the trouble is that, at least in theory, the electrochemical engineer would have to take the electrical energy when the central station man cannot use it otherwise; and that he would have to be satisfied with how much the latter would see fit to give to him. Of course, no manager of an electrochemical works would undertake commercial work on such a basis; and the question is how much energy could central station engineers guarantee to deliver during a year. This is a question which can only be answered by experience on the basis of the daily load curves during a whole year. Some statistical data on what central station engineers are willing to do were given by Mr. Sperry's paper; and it is noteworthy to mention that, according to Mr. Sperry, when central station managers offer very low rates, they often demand the emergency privilege of cutting off the current. In the last end, of course, this, like any engineering problem, resolves itself into a question of dollars and cents. On the one hand, central station managers have to decide how low they can make the rate and still make a profit; according to Mr. Sperry, prices have been made which vary between 0.28 and 0.41 cent per kilowatt-hour ($18 to $26.50 per electric horse-power-year). On the other hand, electrochemical or electrometallurgical engineers must decide whether their processes can be operated intermittently with a profit. The discussion at Cornell seemed to indicate that this would be the case mainly with certain electric furnace processes, since electrolytic processes are mostly continuous.

The whole problem has again been discussed in an exhaustive able way by Mr. C. J. Russell, of the Philadelphia Electric Company, in a paper recently read before the Association of Edison Illuminating Companies. While it is an electric central station manager who has here the word, it is one of the very, very few central station managers of this country who has had practical experience with supply of electrical energy to electrochemical plants. It appears that the Philadelphia Electric Company has supplied energy for making bleaching liquor electrolytically, for making oxygen by discharges through

air, and for making steel in the electric furnace. Concerning the last named application, a few notes of Mr. Russell may be given here in parenthesis, since electric steel is a subject dear to electrometallurgists and the plant in question is, although small, yet the first commercial induction-furnace plant for making tool steel in this country. As our readers know, the plant is at the Diston works and the Colby furnace is used. The furnace is of about 125-kw. capacity, and "several thousand pounds of each kind of high-grade steel of different composition used for cutting tools have been made in this furnace, and samples have been submitted to chemical and mechanical tests." * * * "The high temperatures attainable in the furnace render it possible to thoroughly remove all gases, and the steel is very fluid and still in the mould. The ingots are very dense and homogeneous, and tools made from them present a grain and silvery lustre unlike that of the ordinary crucible steel. The metal also has valuable characteristic qualities for the manufacture of high-grade tools requiring uniform temper. The results obtained have been so satisfactory that it is proposed to install a 5-ton furnace calling for about 750-kw. current capacity."

* * *

Mr. Russell brings out some new points which bear forcibly on the supply question. One is that the character of load diagrams of all central stations which have cultivated the electric power field has undergone a radical change in recent years. In such cases the load condition of morning and evening peaks which represented almost the total generated load has passed. In the present load diagrams the kilowatt area of such peaks as exist bear a much smaller ratio to the area of the daylight load. Now, in view of this fact, it is questionable whether a central station manager will undertake to make a contract for several years to deliver so and so much power at such and such hours to an electrochemical works. But would a contract for a shorter time be satisfactory to the latter? The second point, which, however, is related to the former, is the conviction expressed by Mr. Russell that for purposes of load equalization, it would only be possible to add such processes to any system as should be under the absolute control of the central station interests which were to supply them with electric current. It is an interesting fact that in a careful investigation, the results of which have just been published, Mr. Georg Dettmar, the general secretary of the German Association of Electrical Engineers, comes essentially to the same result. He does not speak of electrochemical processes operated from central station mains, but discusses, on the basis of extended statistical data, the different means of making smaller and medium-sized central stations more profitable. He shows how this can be done and has been done by undertaking outside work during hours of low load so as to keep the men fully occupied, but he shows that to do this successfully, the outside work, of whatever character it may be, must be in charge of the central station manager.

* * *

The [illegible] is disappointing. It seems to [illegible] disappointing though it may be, it is better to look squarely at a situation beforehand. Then there will be no disappointments later on. Naturally there is still a large field of possible developments, since when electrochemical processes become better understood by electrical engineers, they may be adopted by the central stations themselves. Local conditions alone can determine to what extent this would be profitable. Thus in certain districts it may pay a central station to make bleaching liquor or bleaching powder and caustic; in other districts this would be unprofitable, but other electrochemical products might pay. Finally, we might draw attention to a special point in Mr. Russell's paper, though it has only indirectly to do with our subject. Mr. Russell finds fault with the usual energy unit of electrochemical engineers, the "electric horsepower-year." He claims it is the unit of optimists, since estimates based on it are based on the assumption of the utilization of the whole power capacity during all hours of a year. Mr. Russell insists on the kilowatt-hour as being the only sensible unit. But this is another story, and we may come back to it another time.

The Mining Boom.

At present there is apparent in the North American continent unprecedented and extraordinary activity in mining. With the increased diffusion of scientific knowledge, due to the progress of technical schools, technical societies and technical journals, there has been a complete reversal of former procedure. Mining ores and extracting their values at a profit is becoming more and more an exact business where the certainty of profits can be determined and the risk eliminated as far as possible in what St. Paul called "this present evil world." In addition, there is the possibility of large profits from the development of unproven yet possibly mineral-bearing ground. All this appeals to the popular imagination, and the example of good profits made in mining companies by others has caused a widespread interest in mining companies. As John Hays Hammond said to an engineer acquaintance of ours: "Mining, if you know the game, or know a man who knows the game, is the best business in the world, but if you don't, it is the most dangerous." Staid old bank presidents and conservative business men are investing heavily. As all operations are done on a cash basis, if a crash should come it will not bring any great disaster, for the effect is gradually felt now on the New York Stock Exchange in the persistent selling of stocks and bonds by the public.

* * *

As Sir Isaac Newton said: "To every action there is an equal and opposite reaction," so too, stimulated activity in one branch of investment causes decreased activity in another. One very pertinent fact bearing on this subject is the influence of metallurgical progress on mining. The successful development of any mine depends on a successful reduction plant to handle its ores at a profit. Now the improved metallurgical processes have rendered mines once valueless of great value. This is true of [illegible] the domain of the "art of making money out of

ores." The cyanide reaction was the cause of a costly and bloody war in South Africa. The improvement in blast-furnace practice in handling soft Mesaba ores allowed the formation of the titanic "Steel Trust." Other metals, such as zinc and lead, would simply extend the list of illustrations. But without a doubt the most potent of all in widespread effects was the electrolytic refining of copper. This process has been the foundation, strong and solid, of modern electrical engineering, for without its "high-conductivity" copper the dynamo, telephone and transmission line were impossible as they exist now. Thus to America, Mr. Edward Balbach and his chief assistant, Mr. F. A. Thum, who originated electrolytic copper refining in this country in 1883, the mining and engineering world owe a debt of gratitude similar to that owed Bessemer.

The Recent Increase in Interest Rates.

In an editorial in our June number we pointed out several reasons for the increase in the rate of interest. We pointed out that one of the chief contributing causes was the increase in the productivity of capital, due to the irresistible progress of applied science. It was shown how applicable this was to the mining and smelting industries. The improvement in the blast-furnace practice has given actual value to the immense Mesaba ore deposit. This has resulted in the creation of securities of the United Steel Corporation. The open-hearth steel practice has likewise created wealth of phosphatic iron mines. This same course of events has extended throughout the metallurgy of the other principal industrial metals—lead, copper and zinc, and in general the effect can be traced in the entire commercial and industrial world.

* * *

The result we prophesied has come much quicker than was anticipated, and is seen in the fact that the staid old Bank of England has raised its discount rate to 6 per cent. At the present time in New York time loans on unquestioned security are being made at as high as 7 per cent. It is thought that this condition of affairs will continue for some time. Capital differentiated—to use the language of the calculus—is interest, the rate of growth of capital. And while returns to the borrower are so large, the capitalist demands and receives from the "entrepreneur" a large measure of payment. In the nature of things there must be some time a reaction. But the accurate diffusion of business and technical knowledge through the press, the celerity with which modern business is despatched, the improvement in farming methods, due to Secretary Wilson's propaganda, the solid advance in all engineering indicate that this day of reaction is far distant, provided only that the American people keep their heads. In other words, there is a great permanent movement due to a new, far-reaching and persistent cause, which has reversed the precedents of history and has lifted the world upward one step in her evolution.

The American Business Man.

In these days of unprecedented business activity in all circles, but especially that of mining and treating ores at a profit, which is the especial field of this journal, it may be well to analyze the characteristics of the guiding force of these commercial energetics. In America the "*entrepreneur*" and capi-

[illegible] determining commercial success. The capitalist in this country is distinguished from [illegible] features, partly due to [illegible] inherited qualities derived from [illegible]

* * *

One of the chief of these is the use of [illegible] holding mental conceptions clear in the mental horizon as a whole. What our German friends call "Geist," differentiates the American business man from all others. [illegible] Carnegie, Rogers, Harriman and Rockefeller all possess imaginative faculties, which if turned to [illegible] were the spirit of times, their inordinate ambition, and the environment suited, would raise these men to the first rank as artists. Cross and sordid as they may seem to the European, they are creative artists—business artists to be sure, but still creators into actuality of the figments of their minds. This most peculiar and seemingly and irreconcilable [illegible] brought out by Lefevre in his story, "Golden Flood." Naturally, their commercial imagination is always curbed by conservative judgment, but it is still the exercise of the imaginative faculty that is the reason for their commercial success. The next characteristic that is apparent in the American business man is the fighting spirit and persistency. Descendants of men and women who crossed the ocean to fight savages, and "Cooper Indians," to rear the civilization of democracy, these men possess the resolution of the pioneer. By selective emigration Europe is more or less barren of this spirit among her business men. Another trait of the American man of affairs is broadness of view and conception as can be expected in a country with Niagara Falls, the broad prairies and the Rocky Mountains. Ready adaptability to circumstances and environment is a fourth characteristic. This is seen [illegible] spread use of the telephone, telegraph, typewriters, rapid transportation and other modern business [illegible] these conveniences is the basic cause of enormous [illegible] tion of personal fortunes in America.

* * *

We do not hesitate to criticise the American [illegible] in two important particulars, lack of devotion to [illegible] and lack of intelligent interest in the affairs of [illegible] With his intense mental concentration, concentrativeness [illegible] another name for imagination—the duties to [illegible] State do not receive proper attention. The splendid [illegible] science, education and art of many of our [illegible] this fault is noted. But in this connection [illegible] complete personal happiness there should be [illegible] all duties. In the future, with free play of public opinion, this fault will be corrected somewhat. Lack of conservatism is a national fault, and the millionaire [illegible] gory with the rest of us, for Americans [illegible] "bulls." Taken all in all, these men who have [illegible] past generation our commercial size, are pretty good [illegible] but peculiar in their persistency at the [illegible] like all of us, as Kipling said of "Tommy Atkins" [illegible] in barracks most remarkable like you."

Denver Meeting of the American Mining Congress.

(From Our Special Correspondent.)

On Tuesday, Oct. 16, the ninth annual session of the American Mining Congress was called to order at the Broadway Theater, Denver, Col., by the president, Judge J. H. Richards, of Boise, Ida. Among the members and delegates present were many representative men, mostly from the Western States, among them Governors Pardee, of California; McDonald, of Colorado; Brooks, of Wyoming; Cutler, of Utah; Ex-Governor Prince, of New Mexico, and a number of prominent Western mine owners and operators.

After the addresses of welcome by the Governor of Colorado and the Mayor of Denver, the morning session was devoted to brief addresses from delegates representing various States. The program of the afternoon session included the reading of several papers, the first of which, by Mr. E. W. Parker, of the United States Geological Survey, dealt with the loss of life in mining, and especially coal mining. Mr. Parker called attention to the fact that the number of lives lost in American coal mines is considerably greater than in Europe, and he pleaded for a more careful safe-guarding of the lives of the miners as well as for a rigid enforcement of the inspection laws.

Mr. Parker was followed by Dr. J. A. Holmes, of the United States Government coal testing plant at St. Louis, who spoke upon what European Governments are doing for mining. He described his observations in the principal mining districts of Europe, and referred particularly to the investigations undertaken by several European Governments in testing explosives and safety lamps; he announced the intention of the United States Government to begin similar experiments at its coal testing plant at St. Louis.

The next paper, by Mr. Thomas A. Rickard, the editor of the *Mining and Scientific Press,* was entitled "The Geological Distribution of Gold." The author showed that gold deposits are not confined to any particular formation, but that they are found in rocks of all geological ages.

In the following paper, "Mining and Mineral Resources of Utah," Mr. J. Dern, a well-known mine operator from Utah, drew an interesting picture of the resources of that State and the remarkable progress it has made during recent years.

The session on Tuesday evening was occupied by the annual address of the president and a paper by Mr. W. D. Brunton, of Denver, who introduced the draft of a proposed mine drainage district law which had been drawn up by a committee appointed for that purpose. The bill provides for unwatering all the mines of a mining district by a common drainage plant, each owner being proportionately taxed for the cost. The subject is an extremely important one for many districts.

The session on Wednesday, Oct. 17, was opened with a discussion of the mine drainage law, the final result being that the various speakers were invited to put their views in the form of amendments, to be presented to and considered by the committee responsible for the original draft, the committee being instructed to report to the Congress at the evening session.

Another very important subject was then introduced by Governor Pardee, of California, namely, the prevention of fraudulent mine promotion. Dr. Pardee drew attention to the injury to legitimate mining worked by these wildest schemes, and urged the adoption of the draft of a bill to be recommended for passage before the Legislatures of the various States of the Union, making the publishing of wilfully false or exaggerated statements a criminal offense. This paper was also followed by an earnest discussion, during which it developed that the convention was thoroughly in sympathy with the idea of suppressing mining frauds, but, that in the opinion of various speakers, some more fundamental legislation was needed, such as to establish State departments to pass upon the status of newly promoted enterprises. Several resolutions looking toward that object were introduced.

The evening session was devoted to the discussion of the draft of the proposed mine drainage law, which was finally adopted with some minor modifications.

The session on Thursday, Oct. 18, was expected to be the most important on the program, as it was to be devoted to a discussion of the grievances of the ore producers against the so-called smelter trust—the American Smelting & Refining Co. The subject was introduced by State Senator E. De La Vergne, of Colorado, who drew attention to the intimate relation existing between mining and smelting, and complained about the absence of competition, the enforcement of arbitrary and unjust regulations, with especial reference to the Cripple Creek district, and the failure of the smelters in general to pay full value for the metals contained in the ore.

He was followed by Mr. F. Guiterman, the general manager in Colorado for the American Smelting & Refining Co., who endeavored to show that that company had competition in Colorado, that it had stimulated ore production in the various districts, and that its policy was justified from a business standpoint.

In the subsequent discussion various points were raised by the speakers, such as the question of the market quotations on which the settlement prices were based, the question of paying for small quantities of metals in the ore, etc.

The meeting then adjourned till the afternoon, when the discussion was to be continued. Owing to Mr. Guiterman's unavoidable absence, however, the discussion was closed and the afternoon devoted to the reading of several papers of general character, one of the most interesting being by Prof. Trapphagen, of the Colorado School of Mines, who dealt with the subject of smelter fumes and their effect on live stock and vegetation, with special reference to the suit pending against the Anaconda Co., of Butte. He showed that the effects were not as serious as they are usually regarded.

The session on Friday was devoted to the discussion and adoption of a number of resolutions and the election of officers, Judge Richards being re-elected president. Joplin, Mo., was chosen as the next meeting place. On Friday night about 150 of the members and delegates went to the Cripple Creek district as the guests of the Colorado Mine Owners' Association, where they spent Saturday in looking over the mines.

On the whole, the sessions were characterized by much earnestness of purpose. There was very little outing indulged in, and the social features were almost entirely absent. The participants realized that they were there to work and to discuss matters of vital interest to them and the whole mining world. The meeting can, therefore, be called a successful one, and has materially added to the standing and reputation of the American Mining Congress.

The Iron and Steel Market.

The great buying movement in iron and steel inaugurated about four months ago has had a wonderful course. If the orders and contracts prove good, and there is no reason to expect otherwise, the total of all finished steel sold is equal to full production until some time towards the end of the second quarter of next year. Of course, there is unsold tonnage for earlier delivery, balanced by sold tonnage for later delivery. The distribution is not entirely uniform among the different finished products, but it is more nearly uniform than would be expected, considering the varied uses to which steel products are put, and this encourages the idea that the reason for this heavy buying is psychychological rather than physical. The era of "steady prices" and the scarcity of the fundamental raw material, pig iron—seem to have encouraged all consumers to buy very far ahead. There is physical ground for railroads buying steel cars far ahead, since the obvious needs of their traffic require the provision; there is no such ground in tin plate, yet the mills are booked through the first quarter

and well into the second quarter. Never before in the history of the tin plate trade has there been such buying for forward delivery in September and October. Two-thirds of the tin plate consumption depends upon the canning crops. The late canning season was not especially good, and nothing can be known of next season. Tin plate production is about the heaviest yet seen, and deliveries are fairly good.

Granting the pschychological basis of much of the buying of finished steel products, the present acute scarcity of pig iron is readily explained. From time immemorial the alignment of the American iron and steel trade has been that there is an excess of steel-making capacity over pig iron capacity, and of finishing capacity over steel-making capacity. This, because there is less loss in operating a steel works to less than absolutely maximum capacity than with a blast furnace, and the same of finishing capacity as compared with steel-making capacity, while in addition it could not be expected that demand for the various finished steel products, from rails on through shapes, plates, merchant bars, wire and sheet products should be uniformly in proportion to the capacities installed. There are constant cross currents. Demand for rails may be relatively heavy and for sheet and wire products relatively light, and so on. Here has been a condition where demand has called for the operation of all finishing departments to their full capacity. Necessarily with these conditions a shortage in pig iron has resulted; had there been enough pig iron a secondary shortage would have been found in steel-making capacity.

Added to this there was a decline in pig iron production from March to the early part of September, from purely physical causes, but the ultimate buyers do not analyze what can be ascertained as to pig iron production as they should, and appear to have interpreted conditions as they worked out as indicating an increase in the ultimate consumptive demand rather than a decline in the production of the fundamental raw material. In March the United States was making pig iron at the rate of approximately 25,750,000 tons per year. Production declined steadily, until Sept. 1 it was at the rate of about 23,000,000 tons a year, through the humidity of the atmosphere and the necessity of relining an unusually large proportion of the furnaces. The rate of production is increasing rapidly, and barring unfavorable weather conditions leading, through railroad blockades and otherwise, to poor working, the rate could reach 26,000,000 tons before the end of this year, and 27,000,000 tons early next year. A number of new blast furnaces have been completed since last March, and others are to be completed shortly. Such results would undoubtedly change the whole aspect of affairs.

During the past month the best posted men in the trade have reached the conclusion, not always openly expressed, of course, that present conditions foreshadow the end of this remarkable movement, that next year will see declining prices and declining demand. The principal uncertainty is whether the full operations will last only to the middle of next year, or will be prolonged well through the second half.

Pig Iron.

Nearly all furnaces are behind on their contracts, and some consumers are buying prompt lots at fancy prices. Steel-making pig is fully sold through the first quarter, and very little is available for the second. Prices are nominal at about $21, valley furnace, for Bessemer or basic. The shortage of foundry pig is especially acute in the Pittsburg district, and recourse is had to the South. Pittsburg consumers who ordinarily use only Northern iron have bought Southern iron freely. For instance, one consumer took over 1,500 tons of Alabama iron, for first quarter delivery, at $17.00 and $17.25, Birmingham, or $21.60 and $21.85, delivered Pittsburg, equal to $20.75 and $21.00 f. o. b. valley furnace. Most users of Southern iron are content, or even pleased, with the higher phosphorus contents of Southern foundry irons, but forced consumers like the car manufacturers are [illegible] the Southern irons, which are lower [illegible] a spread of 50 or 75 cents is sometimes found between Southern irons, which are usually on a parity. Virginia furnaces are quoting $20.25, furnace, for [illegible] first half, and will not sell for first quarter only. Valley foundry can be had at between $19.00 and $20.00, furnace, for second quarter, but earlier delivery cannot be quoted accurately.

Billets and Sheet Bars.

The large steel mills, being busy in their own finishing departments, and having some crude steel contracts on which they are behind, are not sellers. They are continually turning down large inquiries. A very large tonnage of billets could be sold at present nominal prices of $28 to $29 for Bessemer and $29 to $30 for open-hearth, but the inquirers could not afford to pay a much higher price, as prices on finished materials have been held down. Steel mills are very hard pressed to make outputs, and there is more laxity in analyses. It has long been the case that Bessemer pig, outside the phosphorus limit, has been blown, resulting in steel [illegible] phosphorus, for uses where the defection would not be noticed, and of late there is ground for believing that considerable tonnages of relatively low phosphorus basic iron have been used in the converter. Throughout the industry the rules against phosphorus are being relaxed. Thus, in "low phosphorus pig," used for special acid open-hearth work, the commercial definition has been raised from .035 to .040. Some iron as low as .025 can still be made, but more iron had to be let into the class. The Republic Iron & Steel Co. has made some important sheet bar contracts for first half delivery, chiefly with tin mills, on the basis of about $30.00 [illegible] Pittsburg. The company naturally finds it much more profitable to turn its surplus steel into sheet bars at such prices than into billets or rails at $28, in its convertible mill, and it is practically out of the rail market.

Finished Material.

The National Tube Co., a subsidiary of the United States Steel Corporation, on Oct. 13 advanced merchant pipe two points, or about $4.00 a net ton, and casing about $2.00 a net ton, establishing the market on the basis of 79 and 5 off list on merchant pipe ¾ to 6 inches inclusive, in large lots. While the steel corporation has been uniformly opposed to any price advances at this time, in order to keep the brakes on the market, the independent pipe mills claim this advance was forced by their refusal to book ahead at the low prices, throwing such a tonnage of business to the leading interest that it had to yield.

Other finished material prices are unchanged, except that a concession of 5 cents a hundred on wire products has been withdrawn, the official price remaining unaltered. Prices are, therefore, as last quoted:

Beams and channels, 3 to 15 inches, inclusive, $1.70; over 15 inches, $1.80; tees, $1.75.

Plates, tank quality, ¼ inch and heavier, 100 inches wide and less, $1.60.

Steel bars, $1.50, base; iron bars, $1.60, base.

Sheets, 28 gauge, box annealed, one pass cold rolled, $2.55; galvanized, $3.55; blue annealed, No. 10 gauge, $2.75.

Plain wire, $1.70; wire nails, $1.85, both base.

Tin plates, 100-pound cokes, $3.75 per box.

Above prices are for carload and larger lots, f. o. b. Pittsburg, on sheets and tin plates a concession of [illegible] cents is given, wire concessions are claimed to be [illegible]

Coke.

There has been a spectacular buying movement in furnace coke for next year. Early in the month the regular buyers of Connellsville furnace coke were [illegible] at $2.75 to $2.85 on their contracts for next year. One lot of

Photograph Taken at Dinner Tendered to Sir William Perkin.

Frick Coke Co., which handles all the steel corporation coke, and has heretofore been able to meet all requirements except in extraordinary conditions, suddenly appeared as a buyer and made large contracts on the advancing market, from $2.85 on up to $3.10 for standard Connellsville furnace coke, taking altogether between 400,000 and 500,000 tons of Connellsville, about two-thirds for first half and the balance for second half, besides buying liberally of West Virginia coke. Other consumers are turning more and more to coké outside the old Connellsville region. The coking coal resources of that region are well known; at the present rate of exhaustion they will last about twenty-five years. The rapid growth in production appears to be checked, at any rate the number of ovens has been stationary for a year or more. An acute scarcity of coke cars is on all hands expected this winter.

The Coal-Tar Industry Jubilee.

The fiftieth anniversary of the invention of the dyestuff mauve by Sir William Henry Perkin has been duly celebrated as an event of international importance. England had its celebration during July, and many scientists and chemical manufacturers from other countries gathered at that time in London to pay their respects to Sir William.

New York Celebration.

America followed in October with its own celebration, which culminated on Oct. 6 in a banquet in New York City, which was tendered to Sir William at Delmonico's, and was attended by some 400 American chemical manufacturers and scientists.

Dr. C. F. Chandler was a genial toastmaster. In opening the after-dinner speeches he proposed a toast to the President of the United States, the King of England and the Emperor of Germany. It has been said that in the coal-tar industry these three countries form a triple alliance, Great Britain being the seat of the starting of the industry, Germany the chief manufacturer of dyestuffs, and America the country of coal-tar distillation and a large user of colors.

In his speech, Dr. Chandler referred to the scheme of establishing a chemical library in New York, to be known as the Perkin Library. This is intended to cover the complete field of theoretical and applied chemistry, with duplicate sets of all books, one for consultation at the library, the other for circulation among American chemists. The scheme is quite pretentious, since the library will be under the supervision of two trained chemists, and is intended to be maintained not simply as a library, but also somewhat as an information bureau for American chemists on all questions of a chemical character.

The president of the Board of Aldermen of New York City, Mr. P. F. McGowan, as the representative of the Mayor, Mr. G. P. McClellan, welcomed Sir William in a humoristic speech to the city of New York.

Dr. Hugo Schweitzer then presented what was undoubtedly the most elaborate speech of the evening. In a 15-minute address he cleverly condensed in concise form the vast material, which, as he said himself, could have been covered more easily in an address of 15 hours. After speaking of the way in which Sir William, as an 18-year-old boy, discovered mauve, he emphasized the enormous difficulties which he had to overcome later on in manufacturing the dyestuff on a commercial scale and in inducing dyers and printers to adopt it. The latter were extremely conservative, and would use nothing but natural colors with the formulas inherited from their grandfathers. It has been almost forgotten that Perkin was the first to introduce the method of dyeing silk in a soap bath, which is commonly employed to-day for all artificial dyestuffs, and that he and Pullar first made use of the mordanting of cotton with the insoluble, inorganic compounds of tannin.

After these initial commercial difficulties had been overcome by Sir William, it was comparatively easy to introduce other coal-tar colors. But Perkin also paved the way for the discovery for the later coal-tar colors by creating commercially aniline and benzol, which up to his time had only been laboratory curiosities. Of the three methods available for obtaining aniline he selected as the most promising the reduction of nitrobenzol, made by nitration of coal-tar benzol, and the production of aniline from this source led shortly after the discovery of mauve, to the discovery of magenta, which opened up a new and immense field for this industry.

To-day about 2,000 individual dyestuffs are known, giving the whole range of the colors of the rainbow, and complying with every demand of taste, fashion and stability. They surpass in beauty and brilliancy the colors supplied by nature, and contrary to the impression prevailing among the public, the shades obtained with some of them are faster to the influence of time, light and chemicals than the fastest which nature produces. The greatest triumph of this branch of the industry was the artificial production of alizarine and indigo. In the technical production of alizarine, Sir William has played a prominent part. Coal-tar colors, however, are not only used for the dyeing of textile fibers, like wool, silk, cotton, linen, jute, ramie, etc., but for a host of other materials.

"The lady's hair is gray, or of a hue not fashionable at the time—coal-tar colors will assist her in appearing youthful and gay. In eating the luscious frankfurter your soul rejoices to see the sanguineous liquid oozing from the meat—alas, coal-tar colors have done it, and friend Wiley can prove it. The housewife selects a bright green broom, on account of its anticipated good wearing quality, but finds to her sorrow that coal-tar colors furnished the freshness. The product of the hen is replaced by yellow coal-tar colors in custard powders, and butter is colored yellow when the dyestuff laboratory of the cow is on a strike. Leather, paper, bones, ivory, feathers, straw, grasses are all colored, and one of the most interesting applications is the dyeing of whole pieces of even the bulkiest furniture by dipping them in large tanks containing the dyestuffs, which transforms the wood into walnut, mahogany at your command, as carried out in our big factories in Grand Rapids and elsewhere."

In the further course of his address Dr. Schweitzer referred to the medicinal properties of the coal-tar colors, and gave an amusing account of the way in which the beneficial effects of some coal-tar remedies were discovered. He then spoke of saccharine as a substitute for sugar, and then dealt with the industry of artificial perfumes. He finally referred to the prominent part which coal-tar colors and preparations now play in the reproductive arts, such as printing, writing, photography, etc. Curiously, that problem which was the incentive of Perkin's research, and which started the whole coal-tar industry—the artificial production of quinine—remains unsolved up to the present day.

In conclusion the speaker referred to the enormous influence which the development of the coal-tar industry has had on other chemical industries. "The coal-tar industry gave us our modern chemical institutes, the wonderful equipments of which were first utilized in the laboratories of the factories, and, above all, it gave us the intimate coöperation of technics and science, which is in fact at the root of all this magnificent success."

Dr. W. H. Nichols, president of the General Chemical Co., then presented to Sir William Perkin the first impression of a gold medal to be known as the Perkin medal, and to be given hereafter annually to the American chemist who has most distinguished himself by his service to applied chemistry.

Mr. Adolf Kuttroff, who has been the pioneer of the coal-tar industry of the United States, presented to Sir William a silver tea service of eight pieces, as a token of gratitude from his American friends. (Sir William later on said that the present was not only beautiful but also a very useful and a very suitable gift for a total abstainer.)

which ceased in 1873. After this date Sir William occupied himself purely with scientific researches.

The banquet was undoubtedly one of the most successful and most representative ones ever held by scientists and engineers in this country, and not enough credit can be given to the committee of fifteen who made all the arrangements. Dr. C. F. Chandler was the chairman, Mr. A. Kuttroff the treasurer, and Dr. Schweitzer the indefatigable and ever resourceful secretary.

On the evening of Oct. 9 a reception was given to Sir William at the Chemists' Club.

BOSTON CELEBRATION.

The chemists of the New England States tendered a dinner to Sir William on Oct. 11, at the Algonquin Club, in Boston. The attendance was about 170. The celebration was under the direction of Mr. F. E. Atteaux, while Prof. H. P. Talbot acted as toastmaster.

Among the speakers were Governor Curtis Guild, Jr., United States Senator Henry Cabot Lodge, Dr. Henry S. Pritchett and Mr. W. Whitman. Dr. W. H. Walker, of the Massachusetts Institute, presented to Sir William a silver punch bowl. The reply of Sir William was in general the same as at the New York dinner.

The British Columbia Zinc Report.

The Department of the Interior of Canada exercises a paternalistic care of the mining and smelting industries. Not only have iron and steel companies been liberally subsidized by the Dominion at their outset, but realizing the throttling of the iron industry by lack of adequate coke supplies, they made a most careful investigation of the possibilities of electric iron smelting and steel refining. This was published in the well-known comprehensive report of the commission with Dr. EUGENE HAANEL, Superintendent of Mines, as head, which has had such a stimulating effect on industrial developments.

It has been long known that there were extensive bodies of zinc-silver-lead sulphides in the "Slocan District" and elsewhere in British Columbia. In past ages ore has been shipped to Belgium and to the Kansas zinc plant at a freight rate of about $10 per short ton.

To secure the proper technical knowledge bearing on the zinc industry, the department engaged a commission of engineers from the United States. The chief of these was Mr. W. R. INGALLS, a distinguished author on the metallurgy of zinc, well known because of his two comprehensive treatises on the subject. Under Mr. Ingalls were Mr. Philip Argall, of Denver, and Mr. A. C. Garde, of Denver, both of whom have the reputation of being careful engineers. Mr. Henry E. Wood, of Denver, conducted the tests at his ore-testing plant.

The result of their extended and exhaustive investigations is a book of nearly 400 pages. The subject has been studied from the commercial, metallurgical and mining standpoints. In the question of the valuation of ore deposits and ore prices, the commission has recourse to the formula used by the Belgian zinc plants, which this journal recommended in an editorial in our March number. This method is being somewhat followed in the United States. Spelter and Joplin statistics are given in full, so that the volume forms a ready supplement to previous volumes of Mr. Ingalls.

Especially to be recommended is the section devoted to ore dressing. The magnetic, electrostatic, "flotation," wet processes are elucidated. Numerous tests were made on the ores of the several camps at Denver and the grade raised in cases to over 50 per cent. zinc. The various separators are described and even the patent situation investigated. In fact, there is no work extant in the English language which better describes the up-to-date methods of concentrating complex zinc-lead sulphides than this.

Mr. Ingalls gives considerable space to the consideration of the question of handling zinc ores or zinc concentrates in the electric furnace. His conclusions are that there is absolutely no chance for electric smelting unless power is generated at a figure of $12 per horse-power-year or less in an hydro-electric plant. He also points out vaguely certain metallurgical difficulties in the way of the practical accomplishment. We discussed fully the subject of electric zinc smelting in an editorial in our June issue of 1905.

Our present views are somewhat at variance with those of Mr. Ingalls. While we believe that it is a very difficult problem, yet we consider that intelligently directed hard work will solve the problem in the next ten years, and that the electric furnace will sooner or later modify the metallurgy of zinc and possibly revolutionize it.

Mr. Ingalls is not an electrical engineer, for, had he been so, he never would have overlooked on page 335 the use of the phrase "with a current of 120 volts." Electric current is measured in amperes at a pressure of so many volts, and accordingly we can assume that he is not as familiar with electric furnace design and practice as are those who are electrical engineers.

Concerning the efficiency of the electric furnace, Mr. Ingalls declares that "50 per cent is probably as high as should be reckoned at the present time." This is about right for the so-called induction furnace. With all due respect to the excellence of this special type of furnace for certain work, we do not think that any electrochemical engineer would recommend it for zinc distillation. It is a fact that other types of electrical furnaces show efficiencies of 80 to 90 per cent. In view of this fact, we may say that there is as great danger in making ultra-conservative positive statements as in making random guesses. We believe that time will prove the truth of our assertions.

Taken all in all, we can praise the careful and painstaking work of the commission as well as the discerning paternalism of the Canadian Government in providing proper funds for so complete and accurate study of the best way to develop the mineral resources of the country. The United States Government would do well to emulate the example of Canada. Perhaps the proposed Bureau of Mines at Washington will effect this happy result.

We consider that the British Columbia zinc-silver-lead camps have a promising future, and with adequate concentration and smelting facilities will be large producers of zinc and lead for years to come. To this end the work of the commission has contributed much.

CORRESPONDENCE.

American Electrochemical Society.

To the Editor of Electrochemical and Metallurgical Industry:

SIR.—Please permit me, as a New York member of the American Electrochemical Society, to point out a few facts which should be generally known to understand the undeniable lack of success of the recent New York meeting of the Society.

The members which constituted the local New York committee did not hear of their appointments until three weeks before the meeting. In the short time which remained they did all they could, and the program they arranged was undoubtedly attractive. But the arrangements were, naturally, finished so late that no proper announcement could be made in advance. Otherwise the excursion to the splendid Maurer works should have attracted a great many more members.

No printed "definite" program of the papers to be presented at the meeting was available until the Saturday immediately preceding the Monday on which the meeting began.

The best arranged meeting will fail if not properly announced and advertised. Engineering societies should learn to properly value publicity.

A NEW YORK MEMBER

NEW YORK MEETING OF THE AMERICAN ELECTROCHEMICAL SOCIETY

The Autumn meeting of the American Electrochemical Society (its tenth general meeting) was held in New York City on Monday and Tuesday, Oct. 8 and 9. These dates had been selected so as to enable visiting members to attend the dinner given to Sir William Perkin on Oct. 6 and the reception on Oct. 9. We report at another place of this issue on these functions.

The meetings of the American Electrochemical Society were held in a lecture hall in Havemeyer Building, Columbia University.

Monday Session.

The first session was called to order by President Carl Hering, on Monday, Oct. 8, at 10 A. M.

Dr. C. F. Chandler, the distinguished professor of chemistry at Columbia, made a most cordial speech of welcome. While the American Electrochemical Society is the youngest chemical society, he thought it would prove one of the most important. Electrochemical methods have already worked revolutions in many chemical processes. It is a really great pleasure now to visit one of the great electrochemical works and to compare the new method of working with the old chemical one concerning simplicity, ease and convenience for the workman. For instance, to manufacture caustic soda, how easy and simple is the electrolytic method compared with the old reverberatory furnace process. Dr. Chandler then pointed out the complete revolution brought about by electrochemistry in the manufacture of aluminium. He referred to some details in the electrolytic process, which indicate in an interesting manner the simplicity of the operation and the better protection of the workingman. The molten bath is, of course, very hot. Mr. Hall found that to protect the workingman against the heat it was sufficient to put a layer of pulverized coke on top of the bath. The alumina is then thrown on it and dried by the heat. During operation the bath, which is a mixture of alumina in cryolite, becomes impoverished in alumina, so that at intervals more alumina has to be added to the bath. To find out when this becomes necessary the simple expedient is used of putting a 20-volt lamp between anode and cathode (in shunt to the bath). As long as the bath contains sufficient alumina no current passes through the lamp which remains black, since only about 7 volts are required for electrolysis. When, however, the alumina is being consumed, the voltage rises (since the electrolysis of cryolite would require 25 volts) and the lamp lights up. This is a sign to the workman to add new alumina. In almost all electrochemical plants we see the wonderful effects of simplifying metallurgical operations by electrochemical methods, and of saving the wear and tear on human beings who work the process.

Dr. Chandler finally referred to the fact that whenever a new society is founded the older ones feel somewhat sore about it. It was so with the Perkin jubilee. When the English committee which arranged the big celebration in England, heard of the intention of American chemists to hold their own celebration in New York, they felt sore. Later on New York felt sore when it was reported that Boston would also have its own celebration. Finally everything turned out to the greatest mutual satisfaction, and there was room and glory enough for all. It is the same with the formation of new societies—the more societies the better for the cause of science.

Mr. Carl Hering, as president, expressed the thanks of the Society to Dr. Chandler for his speech of welcome, and to Columbia University for the courteous hospitality extended to the Society.

Concentration Cells.

The first paper, by Dr. Henry S. Carhart, discussed the formula for the Helmholtz concentration cell. In the absence of the author the paper was read by Dr. Bancroft. The author referred to his former statement that concentration cells are devices for converting the heat of [illegible] into electrical energy. The well-known Helmholtz equation for the e. m. f. for any reversible voltaic cell becomes of specially simple form if the energy of the chemical reaction [illegible] Dr. Carhart maintains that in simple concentration cells—in which two electrodes of the same metal are immersed in two different concentration of a solution of a salt of the metal—the condition is approximately fulfilled that the heat of reaction and of the dilution is negligible. In this case the Helmholtz equation assumes the form that the e. m. f. equals the absolute temperature, multiplied by the temperature coefficient of the e. m. f.

Dr. Carhart showed that the same equation may be derived from Nernst's formula for the e. m. f. of a concentration cell as function of the two concentrations. The results were confirmed by experiments with two concentration cells, one containing silver electrodes in silver nitrate.

The paper was briefly discussed by Messrs. Hering and Bancroft.

Visible Migration of Particles.

President Carl Hering presented a paper with this title, in which he first called attention to some peculiar facts observed in electrolysis. In the electrolytic deposits of iron the deposited iron sometimes contains carbon from the cast iron or steel anode. In the electrolytic deposits of alloys the deposit on the cathode sometimes contains some oxide. The deposit of silver in the silver voltameter sometimes exceeds that required by Faraday's law. When a bath is turbid with suspended particles, the deposits on the cathode sometimes become rough, due to the foreign materials becoming enmeshed by the molecules of the metal which is being deposited.

These facts suggested to the author the possibility of a molar migration of suspended particles under the action of the electric current, essentially different from ionic electrolytic migration.

During the past winter Mr. Hering and Dr. E. F. Northrup made some microscopic researches in connection with the very interesting, curious and still unexplained Brownian movements, those persistent and untiring movements of minute, lifeless bodies which are perhaps still the closest approach to an apparent perpetual motion yet observed. These small but molar particles (which for convenience of reference were nicknamed "brownies") were subjected to the influence of an electric current in the surrounding liquid, or to charged electrodes in insulating liquids, and it was then noticed that they wandered across the field from one electrode to the other.

Their motion was very decided; it was along the lines joining the electrodes, that is, from one electrode to the other, showing that they were attracted by one or the other electrode. Their direction of motion depended upon circumstances; most of them traveled with the current, while some traveled in the opposite direction, namely, toward the anode, but each particle had its characteristic direction of motion, relatively to the current, which never changed, at least so far as the experiments showed.

In some mixtures of particles of two different materials, those of one kind traveled with the current and those of the other against it; when the current was stopped or reversed their direction of motion was stopped or reversed with great promptness, like an aperiodic galvanometer; they acted as though they had no inertia; that is, as though the moving forces were immensely greater than those due to their inertia. They obeyed the current as promptly as a body of well trained soldiers obey their officer.

Their speed increased with the current or the difference potential between the electrodes, and seemed to be approxi-

mately constant for constant electrical conditions; it was under certain conditions so great that the particles raced across the field in a few seconds (actual speed 1 foot in 4 hours); the smaller particles, as a rule, traveled faster than the larger ones. In some cases an extremely small difference of potential suffice to move them.

These motions were so positively related to the electrical conditions that it suggested the idea to Dr. Northrup of making it the basis of a sensitive electrical measuring instrument for special purposes.

A peculiarity of these particles while thus in motion, was that they repelled each other during their migration; they rarely if ever quite touched each other; when two particles traveling in opposite directions approached each other, they deviated their paths as they passed each other, like two wagons meeting on a narrow road. They acted as though each were surrounded by an invisible sphere of some repellant force like a very elastic jelly. When electrodes were used in non-electrolytes the particles traveled up to their respective electrodes, and crowded together around them like floes of ice in a running river, but apparently not quite touching them.

The motion takes place in electrolytes as well as in insulating liquids; it is not limited to solid particles but takes place also with liquid particles suspended in a non-mixible liquid, like the little spheres of liquid fat in milk; milk, under certain conditions, even shows the rather rare phenomenon of two kinds of particles traveling in opposite directions in the same liquid. Almost all the particles which were examined traveled in the same direction as the current, namely, from the anode to cathode.

The author suggested that since some colloids have been known to travel bodily when a current is passed through them, and since many of the colloids show Brownian movements, the phenomenon described in his paper might be of the same nature.

In the extended discussion of the paper which followed, Prof. Bancroft questioned whether the different facts mentioned in the introduction of Mr. Hering's paper would be correctly explained as a result of colloidal movement under the action of electric current. In respect to the interesting observations of Mr. Hering, he thought that what had been watched under the microscope was in reality the opposite to electric endosmose. An electric current always causes a relative motion of a diaphragm and a liquid. Whichever is movable moves. When we have a solid stationary diaphragm in a liquid the liquid moves, and we have the phenomena of electric endosmose as first studied by Quincke. On the other hand suspended particles in the liquid represent a movable diaphragm, and will then move under the action of the current in a direction opposite to that of the movement of the liquid.

Mr. C. E. Acker briefly called attention to a recent German patent of J. A. Nussbaum (see our October issue, page 416 and page 379).

Mr. Hering, in reply to Prof. Bancroft, did not see that any other satisfactory explanation of the occurrence of carbon in electric iron deposits had been given than that given on the basis of his paper. With respect to the patent of Nussbaum, which he called quite ingenious, he explained that its quintessence is to add to the electrolyte a very small quantity of such a colloid which will travel with the current towards the cathode. Under these conditions (Hering's explanation is somewhat difficult from and somewhat more plausible than that of Nussbaum) the colloidal particles will travel with the current and at such places where there is a tendency of forming a tree, resulting in a high current density, the colloidal particles will accumulate and will form a protective diaphragm. The lines of current will thereby be shifted to other places at which the current density was less before, and so on. In this way the floating colloidal diaphragm will tend to produce a uniform metallic deposit.

Mr. F. A. Sperry called attention to the fact that Mr. Betts has successfully used the same method for years in his lead-refining process on a large scale, and asked what was the difference between Betts and Nussbaum. Prof. Bancroft answered that Betts uses glue or gelatine, because he has found it effective for getting a smooth deposit of lead. Nussbaum would use glue because it travels toward the cathode with the current, and would therefore be effective. Nussbaum's patent is an outcome of investigations why the addition of some colloids is beneficial while others have no effect. It was found that those which are beneficial travel with the current, the others in the other direction.

Laboratory Appliances for Electric Fusion and Other Work.

Mr. S. S. Sadtler then presented a paper on some very simple laboratory devices for doing electric fusion work. The author has used apparatus ordinarily found in a chemical laboratory for doing quite a range of electrochemical work, and in a number of cases, when he had exhausted his laboratory facilities, larger furnaces, etc., were installed on terra firma.

The current used for the different experiments referred to was from the city lighting service, the voltage being 220 between the outers of the three-wire system. The voltage available between an outer and the neutral was, therefore, 110. Both 220 or 110 volts could thus be obtained with very little trouble. A very simple bank of lamps was used to vary the current. Six stationary sockets were fixed to a board and wired in multiple, with binding posts at one end for connection to the work. The current could thus be varied by putting in more lamps, and varying the capacity of the same from 1/5 amp. to 5 amps., the 1/5 amp. being approximately obtained with the use of one 220-volt, 16-candle-power lamp, while the 5 amps were obtained by the use of six 110-volt, 32-candle-power lamps, which gave nearer to 5½ amps. In this way all the current desired was quickly obtainable and without danger of short circuiting.

For carrying out a quick experiment to see whether a metal is deposited, or to get a rough idea of the state of fluidity of a proposed bath, a Chaddock gas burner is found serviceable, upon the top of which a small Hessian crucible or one of iron or nickel can be rested, in which the molten electrolyte can be contained, and the anode, a rod of graphite, or whatever is desired, can be dipped into it from a clamp which makes electrical contact, and for a cathode, a very simple and effective device is made as follows: A wire is bent into a spiral at one end, which fits in the bottom of the crucible and more or less covered by the separated metal, and the surface of the wire contained in this spiral is made proportional to the desired current density. The stem of the spiral comes up the inside of the crucible, and is cemented to it by a plastic mass, which, in some cases, could be fire-clay or carborundum powder and silicate of soda, which works satisfactorily for many fusions of a neutral or acid character.

When a larger experiment is to be performed a Buffalo dental furnace, with a large gas blast lamp or gasoline lamp for purposes of heating, was found very satisfactory by the author. In the top of this the crucible can be inserted, of proper size, with the same spiral arrangement for cathode. In some cases, where an iron cathode is wanted of considerable surface, a large iron crucible or iron body of an iron still of about a quart capacity may be used in the furnace.

In some experiments carried out by the author, in which certain gases were passed over a solid at definite temperatures, a tube furnace was employed, which was very effective for the purpose.

In looking up the proper size of platinum wire for electrical heating, it was almost impossible to decide upon the size and amount of wire that would be required, and it was not until a trial was made that a fair idea could be gotten. The author gave the following specifications for this furnace:

The tube consisted of Royal Berlin porcelain, of about

½-inch inside diameter, and 2 feet long. Around this, allowing about 8 inches free at each end, was wrapped platinum wire of No. 28 gage. It took 10 feet of wire to cover the body of the tube, allowing about ⅛ to 3/16-inch space between the folds of the wire. As the platinum wire was wrapped tight and clamped at the ends with heavy copper wire, the free ends of which were used as electrical terminals, it was not found necessary to put in any insulating material to keep the wire apart. Asbestos cord could have been used for this purpose.

A small section of 3-inch galvanized piping, such as that used for rain spouts, was used for the outside shell, and between this and the tube a mixture of magnesia and asbestos was packed, sheet asbestos being put on the ends and fastened thereto with silicate of soda wired.

A variety of rheostats may be used for varying the current. The author used one of small discs of carbon, the resistance of which was varied by adjusting the pressure by means of a screw. Temperatures varying from 600° or 700° F. to 2,000° were used by the author, and undoubtedly considerably more could have been obtained. The pyrometer used, however, only registered to 2,000.

The control of temperature was gotten by the use of an ammeter in circuit, and a pyrometer, the fire end of which was encased in a small porcelain tube, which ran in the larger one and held at one end firmly in place by means of a rubber stopper. The use of an ammeter is quite essential to the successful control of an experiment, as the pyrometer, especially one encased in a porcelain tube, does not register a change of temperature quickly enough, and too high or too low temperatures would be the result where more time would be desirable.

Copper Cathodes in Nitric Acid.

A paper by J. W. Turrentine on this subject was read in the absence of the author by Prof. Bancroft.

Attention is called to the fact that nitric acid in dilute solutions undergoes reduction quantitatively at the cathode. This method has been recommended for the quantitative determination of nitrates. This reduction is equivalent, of course, to an action by nascent hydrogen, liberated at the cathode, on the nitric acid, according to the equation:

$$7H_2 + 2HNO_3 = 2NH_3 + 6H_2O$$

Nitric acid in acting on copper, on the other hand, is reduced not to NH_3, but to nitric oxide, as indicated by a free evolution of brown fumes. In other words, copper reduces HNO_3 to NO and not to NH_3. It reduces nitro-benzene to aniline. Why does it not reduce HNO_3 completely? Why does the reduction stop at nitrites?

These facts appear to contradict Bancroft's general law of the parallelism between electrochemical and purely chemical action and a series of experiments were made to explain this discrepancy. The first clue was found when experiments showed that HNO_3 could not be reduced quantitatively to NH_3 in the presence of Cu ions, and that the reduction stops at the nitrite stage.

The chief results of the author's experiments are that, when copper dissolves in nitric acid the anode reaction is the formation of copper nitrate and not of nitrite. When copper dissolves in nitric acid the cathodic reaction is the reduction of the nitric acid. The chief reaction product is NO when there is copper as ion in the solution, and NH_3 when there is practically no copper as ion in the solution. When a solution of copper nitrate and nitric acid is electrolyzed, with a copper cathode, nitric oxide is formed at the cathode. When a solution of nitric acid is electrolyzed, with a copper cathode, ammonia is formed at the cathode. The apparent discrepancy between the chemical and electrochemical action of copper on nitric acid is, therefore, due entirely to a difference in the conditions under which the two sets of reactions have been performed. The paper was discussed by Dr. Thatcher and Prof. Bancroft.

Aluminium Magnesium Cell.

A paper by Messrs. G. H. Cole and H. T. Barnes, on an aluminium-magnesium cell, was read in the absence of the authors in abstract by Prof. Barnes.

The authors describe a series of experiments with a cell in which a magnesium and an aluminium electrode were used with potassium alum as electrolyte This acted as a galvanic cell with the current passing in the cell from aluminium to magnesium. The most interesting result is that when hydrogen peroxide was added as a depolarizer, or oxygen bubbled through the cell, an e. m. f. of 2 volts could be fairly well maintained.

Since aluminium and magnesium are so close together, the high e. m. f. is remarkable, but no explanation of it was offered by the authors.

In the discussion which followed Prof. Loeb suggested that the magnesium might decompose the water, and the cell might be after all a gas cell.

Melting Currents of Cryolite-Alumina Mixture.

A paper by Mr. Francis R. Pyne was then read in the absence of the author by the secretary, Mr. S. S. Sadtler. The melting points of different mixtures of cryolite and alumina, containing between 0 and 20 per cent of alumina, were determined by plotting the cooling curves with the aid of thermo-couples. The chief results are given in the following table:

Cryolite %....	100	97	96	95	94	93	92	90	85	80
Alumina %...	0	3	4	5	6	7	8	10	15	20
Melting point..	1000	974	960	915	960	982	992	980	994	1015° C.

The melting point is a minimum for a mixture containing 5 per cent of alumina. The drop just behind 8 per cent seems to indicate the formation of a compound.

Double Decomposition of Zinc Sulphate and Sodium Chloride.

In a paper by Mr. P. B. Sadtler and Dr. W. H. Walker, an account is given of an investigation, the object of which was to find a cheap and easy method of making zinc chloride from sulphate. This is accomplished by using saturated solutions of common salt and zinc sulphate and chilling them to a certain point, which results in the crystallization of glauber salt, leaving zinc chloride. Under proper conditions an efficiency of almost 100 per cent may be obtained. In the explanation of the theory of the process, the phase rule was found to be of much usefulness, but this part of the paper was only briefly read in abstract.

Distribution Law.

A paper by Dr. H. E. Patten, of the Bureau of Soils, Washington, entitled "Some Factors Affecting the Distribution Law," was read in brief abstract by Mr. Sadtler. It discussed essentially the quantity of dye absorbed by quartz powder. For details the reader must be referred to the full paper, which will be published in the Transactions.

Use of Ultra-Violet Light in Laboratory and in Practice.

In the afternoon, Dr Charles Baskerville delivered a very interesting lecture at the College of the City of New York on the use of ultra violet light in the laboratory and in practice One very marked property of ultra violet light is its power to transform O_2 into O_3. This takes place best at low temperatures It has been thought that the ultra violet light from the sun forms O_3 in the upper atmosphere, which then settles down and oxidizes the impurities nearer the earth

An electroscope is almost instantly discharged under the influence of ultra-violet light, but the action is quite slow if a quartz screen is interposed.

The most important application of ultra violet light up to the present time has been in the testing of minerals Certain minerals are inert, others effloresce, some phosphoresce, and still others both effloresce and phosphoresce

Kunzite was discovered by aid of ultra violet light A large

number of spodumane samples were tested, and all found to be inert except one. Upon closer examination this one was found to contain a hitherto unknown mineral which was named Kunzite, in honor of the discoverer, Dr. George F. Kunz.

All the minerals from Mono Lake are phosphorescent after treatment with ultra-violet light. Some of the same minerals from other localities are not phosphorescent.

The genuineness of diamonds may be proven by their phosphorescence after being subjected to ultra-violet light.

The most important practical application at the present time is for testing the purity of tailings or precipitates, and for sorting minerals. A crushed mineral may readily be separated into an efflorescent and a non-efflorescent portion by sorting while acted upon by ultra-violet light.

The same principle is made use of by the New Jersey Zinc Co. for testing willemite concentrates and tailings. The willemite is efflorescent while the gangue is inert. If the tailings contain no efflorescent specks it shows that the concentrating tables are working well. The eye can also roughly judge the degree of concentration in the concentrates.

Tuesday Meeting.

Pyrometers.

The main part of the Tuesday meeting was devoted to an exhibition and discussion of various types of pyrometers. A brief introductory talk on this subject was made by Dr. E. F. Roeber.

First, concerning the definition of temperature. Our senses tell us whether a substance or a body is hot or cold, but they do not give us a temperature scale. All measurements of the temperature of the body are essentially based on using a body of comparison, which is called a thermometer or pyrometer. We bring this in contact with the body, the temperature of which is to be measured, and we wait until thermal equilibrium is established. As thermometric substance we use a body which has some property which greatly changes when the body is cooled or heated. This variable property we employ for the temperature scale. Thus in a gas thermometer the absolute temperature is directly given by the volume.

It is important that thermal equilibrium is established between the substance under test, and the thermometer before a reading is taken. This is especially of importance for recording instruments for use in cases where the temperature changes very quickly. A sensitive thermo electric pyrometer, for instance, will follow immediately all changes of temperature in air during a thunderstorm. It will show a sudden drop of temperature, for instance, while a mercury thermometer shows at the same time a less sudden drop, simply because it is not able to get immediately into equilibrium with the surrounding air.

The use of a thermometer for measuring temperatures makes us dependent on the substance of which the thermometer consists, mercury, gas, etc. We may choose two fixed temperature points. Then different types of thermometers, when properly calibrated, will give correctly the temperature at these two fixed points, but there will be variations at other points, though they may be small. Now, there is really no reason why one substance should be better than another substance, if we had not the fact that the laws which govern what we call perfect gases are so simple and uniform that we conclude that these regularities are based on a simple and uniform constitution of gases, and the temperature uniformly indicated by them we define as *the* temperature. This is the reason why the gas thermometer temperature is the standard.

But, really even those gases which are almost "perfect" show slight differences, and if we want to take those into consideration then we are in the air with our definition of temperature, and we cannot get on solid ground except we get free altogether from an arbitrary thermometric substance. We need a temperature scale which is absolutely independent of any special substance.

This can be accomplished in no other way but by using the second principle of thermodynamics, in its most general form, where it does not depend on any special substance. This leads to the thermodynamic temperature scale of Lord Kelvin. How it is possible to realize experimentally this thermodynamical temperature scale—in other words how it is possible to apply the corrections to the indications of our gas thermometers—is shown in detail in Planck's well-known book on thermodynamics. While we have theoretically a possibility of establishing an exact absolute temperature scale, it has not yet been established.

While this shows how far we are still backwards in an exact definition of temperature, enormous progress has been made in recent years in practice in the invention and application of thermometers and pyrometers. We have now decidedly what we may call a pyrometer boom, and while the progress has been rapid, there is no doubt that it will be even more rapid in future, since the absolute necessity of controlling temperature in all kinds of industrial processes finds due recognition only just now. For all practical purposes the gas-thermometer scale is perfect, and by comparison with its scale we can calibrate any other pyrometer. This is all that we need in practice. For practical purposes we need some fixed points, and we have them in the fusing points of pure metals. But it is evident that there should be an international agreement concerning the figures for these fixed points. The author referred to some cases in which the figures of the Bureau of Standards in Washington and by the German Reichsanstalt agree with each other, while different values are used in France. This must, of course, lead to confusion.

We know that in many industrial, chemical and metallurgical processes the maintenance of the correct temperature is absolutely essential. Formerly the manufacturers depended on the trained eye of the workman. The disadvantages are evident. All the knowledge is stored in the workingman. When he leaves, the knowledge how to regulate the temperature leaves with him. On the other hand, if we use a pyrometer, we need to establish once for all the correct series of temperatures required for economical working and make a record, and we can then duplicate this record all the time. The pyrometer makes us independent of the personal equation of the furnaceman.

Further, recording pyrometers give us a method of control of the workman. We see from the record how efficient the different men are.

Moreover, if properly put up to the workingman he will look at a pyrometer, not as a spy, but as a help. It gives him some guide to rely upon. Mr. Whipple, of the Cambridge Scientific Instrument Co., has proposed to establish a bonus system. A certain bonus should be paid to the workingman if he keeps all the time within certain limits of temperature.

This is not only so with pyrometers. The same experience has been made with CO_2 recorders; for instance, in the works of the New Jersey Zinc Co., or in the power plant of the New York Subway. In both cases the use of these recorders has greatly enhanced the efficiency.

Bristol Thermo-Electric Pyrometer.

Mr. F. F. Schuetz then presented a paper on the thermo electric pyrometer of Prof. William H. Bristol, which has already been described in detail in our March issue (our Vol. IV., p. 115). The paper was fully illustrated by demonstrations of the instrument, various types of which were exhibited. A new type shown on this occasion is Bristol's new recording pyrometer, illustrated in Figs. 1 and 2.

It is well known that the actuating current for the indicating arm derived from a thermo-electric couple is so small as to shut out all direct inking or any continuous direct contact of recording arm with chart. Prof. Bristol employs a specially prepared chart which is rotated by a suitable clock movement, and an electrical measuring instrument controlling a light recording arm provided with a marking point. The record chart

is supported only over a portion thereof, the remaining portion over which the recording arm acts being unsupported and perfectly free to vibrate.

Vibration is periodically communicated to the unsupported portion of the chart by a tapping apparatus controlled by the clock movement. The marking point is normally free of this chart, and comes in contact with it only during the period of

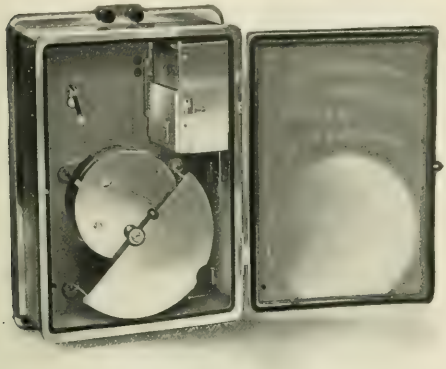

FIG. 1.—RECORDING PYROMETER

vibration. Because of the delicacy of the recording arm no attempt is made to ink with it, but through the vibration of the chart vibration is communicated to it so that the impact is gradually taken up by the recording elements and the pointed end of this recording arm caused to touch the chart.

The chart itself is of the circular disc type, 8 inches diameter, such as used in the usual recording voltmeters, ammeters, pressure gauges, etc., and is semi-transparently smoked over

FIG. 2.—RECORDING PYROMETER.

its active surface so as to only partially obscure the graduations. The surface of the chart is so sensitive that it may be marked by a hair, and as the pointed end of the recording arm touches it a small amount of the soot is removed, leaving a small white dot visible. The frequency of vibration is such that a continuous line is marked upon the chart by the recording arm. There is, therefore, but little strain placed on the arm, and practically no friction between arm and chart to affect the sensitiveness of the movement of said arm, the contact being only momentary. After the [illegible] the chart is removed and fixed by dipping into a fixative solution. This permanently fixes the record so that the chart may be filed away.

The second part of Mr. Schuetz's paper dealt with the application of pyrometers in commercial work. The author pointed out that with the availability of a comparatively inexpensive and practical pyrometer, capable of standing rough usage, pyrometers may now be used for such work for which they were not employed before. Experience has shown that the workmen take a positive interest in the pyrometer, and

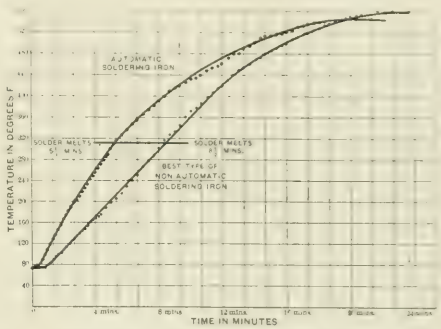
AUTOMATIC SOLDERING IRON
SOLDER MELTS 51 MINS.
SOLDER MELTS 91 MINS.
BEST TYPE OF NON AUTOMATIC SOLDERING IRON
TEMPERATURE IN DEGREES F
TIME IN MINUTES

FIG. 3.—COMPARATIVE TESTS OF ELECTRIC SOLDERING IRONS.

that whenever such instruments are introduced the efficiency of operation is improved.

For the instantaneous determination of the temperature of molten metals, Prof Bristol employs an interesting modification of the old thermo-couple, the wires at the hot end of the couple being left separated. The couple is dipped into the bath, which makes an electrical contact with the two elements, completing the circuit and immediately indicating the temperature by a suitable indicating device. When the couple has worn away to a length of about 1 foot the fire end is replaced by a new one.

This form of couple with pointed ends also affords a most convenient means for determining the temperature of metallic plates, and has been used to determine the proper temperature of the plates used in the manufacture of brazing powder and of zinc plates in the rolling of zinc sheets. This form of couple is sometimes provided with a small disc contact plate, especially when used to determine the temperature of electrically non-conductive objects.

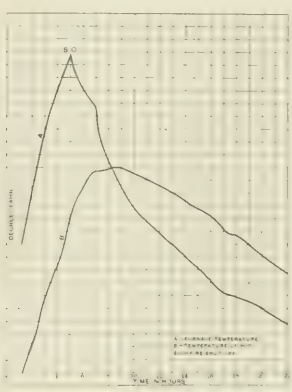

FIG. 4.—ANNEALING FURNACE RECORD.

Of the many practical applications mentioned in the paper we may mention only the application to comparative tests of electric soldering irons. The two curves in Fig. 3 represent the results obtained with an automatic iron and with a non-automatic iron, with the time plotted as abscissa and the temperature as ordinate.

Readings were taken about every 7½ seconds; and it will be seen that it required the automatic soldering iron but 5⅓ minutes to reach the soldering temperature, whereas the non-automatic soldering iron required 8½ minutes.

Another interesting application is in connection with lead-hardening baths. The use of pyrometers has here resulted not only in more uniform results, but in a longer life of the pots containing the lead. The two curves A and B in Fig. 4 show respectively the fire temperature and the temperature of the box containing the articles to be annealed. It will be noted that the temperature of the box rose for some time after the fire was shut down.

In the handling of many small parts, such as in the manufacture of watches, clocks, etc., a very convenient device has been constructed by a large clock company. The pot containing the articles is adapted to the hot end of the couple, and the couple used as a handle for inserting the pot into the furnace or into an ordinary forge fire. By revolving the pot it is heated perfectly uniformly and is removed as soon as the required temperature is reached, as shown in the indicating instrument, all guess work being thus eliminated.

Heraeus-Le Chatelier Pyrometer.

Dr. Richard Moldenke, Watchung, N. J., the genial secretary of the American Foundrymen's Association, then presented a paper on the well-known thermo-couple of Le Chatelier, as constructed by Heraeus. As is well known, the combination of pure platinum with an alloy of platinum with 10 per cent rhodium is employed in the same.

When the Le Chatelier pyrometer was first brought to the attention of the scientific public, and its excellence was realized, the German Government, through the Physikalische Reichsanstalt, undertook to determine the proper length, thickness and resistance of the couple which would give the best results, and could therefore be made standard. This was found to be a length of 58 inches, a diameter of the wire of No. 23 B. & S. gauge, and a resistance of 1.6 ohms. To get as near perfection in the preparation of the element as possible, the work of preparation was entrusted to Dr. W. C. Heraeus, the proprietor of the platinum works of that name in Hanau, Germany.

Dr. Heraeus, in working out the problem of supplying a couple which would be absolutely uniform and interchangeable at all times, provided two blocks of the necessary materials worked up to uniformity in structure and perfectly pure. Thus there is available a supply that will last far beyond a lifetime, and the first as well as the last couple is absolutely the same. This feature alone makes the pyrometer a true standard.

After describing briefly the principle of the instrument the author referred to his personal experience with the instrument in daily use in malleable cast iron foundries, where the temperature of the annealing ovens has to be taken regularly, and where a small fortune is at stake continually. Perfect results were obtained by an ordinary laborer, selected for trustworthiness, recording the temperatures observed for the benefit of the annealer in charge.

The ease with which the instrument is applied in the particular case in point is shown by the fact that where formerly the Siemens water pyrometer occupied the attention of two men in 24 hours, with the Le Chatelier pyrometer, the galvanometer located centrally, and the ovens fired up, the thermometer is introduced into the oven, and while the man walks to the galvanometer the pointer has reached the stationary point, the reading is taken, the thermo-couple moved to the next oven, and so on. A few hours in the twenty-four is all that is required, and the result so accurate easily within 25° F. Parenthetically it may be said that the temperature of the hottest pot of the annealing ovens for annealing malleable castings should not fall below 1,470° F., nor go above 1,550°, this for the best results with fairly heavy castings and on furnace iron. What can be accomplished before a good pyrometer was available can be better imagined than described.

For taking the temperature of molten metals, a clay tip arrangement is essential, since this can be dipped into the ladles directly without injury and the temperature observed within half a minute or so. The author then referred to various uses of this pyrometer, for instance, in connection with open-hearth glass works, chemical works, potteries, and especially in steel works.

Even the boiler room can come in for a share of attention. The author referred to a case in which he had to solve the smoke problem in a plant in Pittsburg, and only after applying the Le Chatelier pyrometer to every possible place in the 1,800-hp. installation was the solution found. The breeching and base of the stacks gave entirely too high temperatures for proper economy. An investigation of the grates showed that they were burned out too often. Smoke poured from the stacks of the over-worked boilers, and these had to be kept going at full pressure night and day. The stacks were raised an extra 50 feet one by one, making 150 feet in height each. The coal now danced on the grates. Plenty of oxygen got at the fuel. The grates did not burn out after this, and the boilers gave full value without a particle of smoke.

Here was a case where the best of stokers, a coal-conveying system and ash removal were installed with the highest class of water-tube boilers, and yet the overcrowding which always takes place in a live growing works brought about a lot of trouble quickly solved when the pyrometer was applied. There are others who might profit by doing this at their plants.

In conclusion, Dr. Moldenke, who was a member of the Jury of Awards at the St. Louis Exposition, remarked that while Mr. Engelhard, as the American representative of the Heraeus-Le Chatelier pyrometer, received the gold medal for his exhibition, they would have given the grand prize to Prof. Le Chatelier had he made a personal exhibit.

A Heraeus-Le Chatelier pyrometer had been placed on exhibition by Mr. Charles Engelhard at the meeting.

Resistance and Radiation Pyrometers.

Mr. R. S. Whipple, of the Cambridge Scientific Instrument Co., of Cambridge, England, who has distinguished himself in developing methods and uses of pyrometry, made a very interesting speech. He thought the little compensator in the Bristol instrument, by means of which changes in the temperature of the cold end are compensated, was quite ingenious. He agreed that it is necessary to pay attention to constancy of the temperature of the cold end. He has always recommended the use of some waste steam, which is available in every plant. With the careless methods often used at present in many plants considerable errors may arise, due to changes of temperature of the cold joint.

Mr. Whipple congratulated American manufacturers and engineers on their readiness to adopt pyrometers in their plants. When the use of a new thing is recommended to an English manufacturer, he is liable to ask whether his competitors are using it or not. The American manufacturer does not ask such a question, but simply wants to know what advantages the instrument would bring him.

Mr. Whipple then described three instruments made by the Cambridge Scientific Co. and placed on exhibition. The first was a Féry radiation pyrometer, a detailed illustrated description of which was given in our Vol. III., page 478, exhibited by the Wilson-Maeulen Co., of New York City. The second and third were electric resistance pyrometers.

Resistance pyrometers have a wide range, from the lowest temperatures to 1,200° C., or nearly 2,200° F. They can be placed in inaccessible positions and read from a considerable distance. They are extremely sensitive and respond quickly to changes of temperature. In conjunction with the Whipple temperature indicator they give direct readings of temperature requiring no correction. In conjunction with the Callendar electric recorder they furnish a continuous and visible record of temperature, extending over a day or week, and the

temperature can be read off at any time without disturbing the recorder.

Fig. 5 shows the resistance pyrometer connected to a Whipple temperature indicator. By depressing the contact key F and by turning the middle head H, the resistance of the indicator is made to equal that of the thermometer, the galvanometer needle at B showing when balance is obtained. The temperature in degrees is read off directly on the scale at A.

The indicator need not be near the thermometer, but may be at a considerable distance from it without effecting the accuracy of the readings. The instrument is so constructed that rapidly varying temperatures may be readily followed and measured. It is easily portable, and by means of the instru-

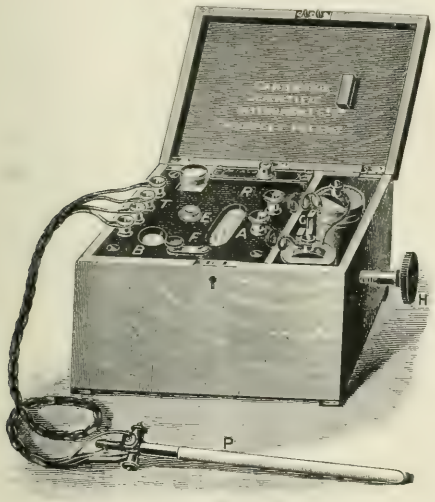

FIG. 5.—RESISTANCE THERMOMETER WITH WHIPPLE INDICATOR.

ment any workman without any electrical knowledge may find the required temperature in 2 or 3 minutes. No corrections of any kind have to be applied.

Mr. Whipple then showed several interesting lantern slides from pyrometer practice. One of these is reproduced in Fig. 6, which is an annealing-furnace record, obtained with a resistance thermometer in combination with a Callendar recorder. Such a record discloses at a glance to what extent the temperature of the furnace has deviated in either direction from its proper value. Incidentally it shows quite clearly how often the fires have been fed, each firing producing a transient heating effect, which is shown by a well marked peak on the record. The temperature is about 700° C., and the portion

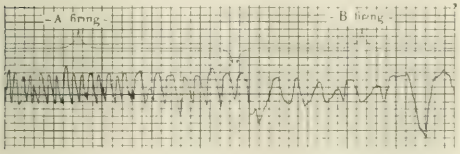

FIG. 6.—ANNEALING FURNACE.

of the record shows an extension over about 9 hours. Each space on the vertical scale corresponds to 4° C.

It is interesting to note the difference in the firing of the two men A and B. One of the two firemen was a young man, and his nervous state of mind is shown by the way how often he tried [illegible] quicker. He [illegible] little nap.

BROWN PYROMETERS BASED ON DIFFERENCE OF EXPANSION.

Mr. Richard P. Brown [illegible] Philadelphia, Pa., their exhibited and described two pyrometers based essentially on the difference of expansion of two different materials. This is, of course, the underlying principle of the ordinary mercury thermometer, the [illegible] is based on the fact that mercury expands very much more under the action of heat than the glass tube in which it is contained and the scale with which it is combined. The same principle is made use of in quite an interesting way in the two Brown instruments.

The first is a recording pyrometer, which is a modification of the Brown hot-blast pyrometer, which was originally designed for use on the hot-blast main of blast furnaces, but has now come into general and extensive use for a great variety of operations for measuring temperatures up to 1,500° F. or 800° C. The Brown recording pyrometer keeps a record of the temperature on a chart for 24 hours.

Inside a steel tube are graphite rods, having practically no expansion, and the indication of the instrument is caused by the difference in expansion of the outer seamless drawn-steel tubing and the graphite rods, 12 inches long. Above the graphite rods is steel tubing with pieces of brass or other metals, by which the instrument is accurately compensated for various insertions in the heat. No matter whether the stem is inserted 12 inches or 24 inches in the heat the instrument will indicate accurately. In the case or head of the instrument is a multiplying movement, which multiplies the slight difference in expansion, and an adjustment is furnished in the movement by which each instrument may be adjusted accurately. Also an adjustment is furnished on the outside of the case by which the pyrometer is adjusted to the temperature of the atmosphere before use.

The second pyrometer exhibited by Mr. Brown was a "quick-acting platinum pyrometer," which has a thin strip of platinum 8 inches long, hung between heavy iron frames. When the end of the stem of the instrument is inserted in the heat the thin platinum strip is directly connected to a multiplying movement, which moves the pointer. When the thin platinum strip is heated the pointer quickly moves around the dial, and when the platinum has ceased to expand the pointer stops, and as the iron frame begins to heat up the pointer will begin to move back again. The maximum temperature indicated is the temperature of the furnace or heated space. As soon as the temperature is noted the pyrometer is withdrawn and can be used again when the instrument returns to the atmospheric temperature, which will be in about 15 minutes.

As the time necessary for the platinum to heat up will be only about 10 or 15 seconds, the iron frame will not have risen about 200°, probably less, and as the platinum will only be subjected to the high temperature for so short a time in each test, the indications of the instrument will not be altered, due to any deterioration. This instrument can be used for measuring temperatures up to the melting point of platinum.

PRICE THERMO-ELECTRIC PYROMETER.

A Price thermo-electric pyrometer made by the Electric Dental Specialty Co., of Cleveland, Ohio, was then exhibited. Two couples were shown in connection with the instrument for demonstration purposes, but it was explained that out of the some 125 different industries which are now using this type of pyrometer very few have the same kind of couples, since the material of the wires and also their length and size in the couples depend upon the conditions of the several industries in which the instrument is to be used. For high temperatures platinum is required in order to get reliable results, but for low temperatures other metals are equally good.

The two wires of the couple are insulated from one another

in several different ways. Where compactness is of importance they may be run through magnesia tubes ⅛ inch outside diameter and having an 18 B. & S. hole. Another way is to secure clay tubes sufficiently refractory to stand high temperatures having holes pierced through from end to end. Sometimes the wires may be held apart by asbestos cord woven in and out; asbestos tubing is excellent where there is a great deal of vibration.

For higher temperatures which reach about 1,600° C. or more it is necessary to use a refractory porcelain or semi-clay porcelain tube, which is closed at one end, unless very sudden changes of temperature are to be recorded, when the end of the couple or junction is run through outside. It is still better if these tubes are glazed on the outside to prevent any possibility of contamination to the thermo-couple by furnace gases.

Where the heat is extreme these tubes are often again supported by large fire-clay tubes having holes straight through, allowing the end of the former tube to project through a short distance. This prevents the sagging of the porcelain tube when it is in a horizontal position, and also prevents it from breaking. When inserted in a clay tube it has been found to last three or four times as long.

When working at higher temperatures with platinum, iridium and rhodium, it has been found better to use two couples in series, in order to get greater precision or accurateness of temperature. In this case the cold end of the couple need not extend far from the heat but may be joined there by certain wires of a metal, which develop a counter-electromotive force equal to that set up by the cold junction. This has been perfected to a high state of efficiency, allowing, in

FIG. 7.—PRICE PYROMETER.

some cases, the cold end of the couple to be at temperatures as high as 500° C., thus greatly reducing the length of the couple. But at temperatures less than 850° or 900° there are several metals that may be used and can be relied upon. Brass and alloy of copper and nickel gives a very high and steady e. m. f. Iron used with the same alloy is almost as good unless it is exposed to injurious furnace gases.

The method used in the Price pyrometer for its indicating instrument is to suspend the armature on one single point, since when an armature is suspended between two points and turned over sometimes in a horizontal position, it is bound to cause a considerable amount of friction, which consequently prevents accurate reading of temperatures.

The incoming leads to the armature are crimped suspension springs extending 4 or 5 inches above the armature. Where they are attached the ends to the armature are very close together, but the ends at the top are further apart to give the proper torsion and resistance to the current. The crimp in these springs allows a lengthwise spring of nearly ¼ inch in that distance, so that it is almost impossible to break them.

When the armature is suspended in its place above the point a large proportion of its weight is carried by these suspension springs, and only enough left on the point to keep it centered around the core and in the field. This reduces its internal friction to almost nothing, making it possible to get that distance, so that it is almost impossible to break them.

The thermo-electric pyrometer can readily be arranged to give a continuous record of temperature. At every minute a tiny spark leaps from the indicating needle and passes through a continuous sheet of paper to a contact below. This paper has marked upon it the hours and parallel lines representing the temperature. It is just as easy to see the temperature on this recording instrument as on the ordinary indicating instrument, as the paper simply passes underneath the needle, leaving the ordinary scale in view just the same as usual. It has the great advantage that it has no inclination whatever to retard the needle but allows it to show the exact measure of the temperature both on the scale and the hole through the paper.

FIG. 8.—SWITCHBOARD.

These holes appear similar to the puncture of a pin. One of these recording instruments may be in a head office, and it may be a single, double or triple machine, recording the temperature of one, two or three places at once, while at the plant, for the use of the fireman, is an indicating instrument connected to the same couple or couples as that used by the recording instrument, so that the fireman always sees just the same temperatures as the recorder is continuously marking. The wire connecting the couple with the galvanometer may even be a mile in length.

For the calibration of these instruments fixed points are, of course, required. Copper melted in a graphite crucible, with its surface protected with powdered graphite, gives a very definite and sharp fixed point at 1,084° C. when freezing. (Day and Allen's value is 1083.6 in a reducing atmosphere.) During the freezing period the temperature remains quite constant, giving ample time for observations. Certain grades of gold in a very loose form or rolled exceedingly thin and bunched up in a small bunch will usually melt within 5°.

The instrument is very well adapted for immersion in small specimens of steel and other metals for the study of the cooling curves and in connection with annealing, hardening, etc.

While thermo-electric pyrometers were formerly used almost exclusively for high temperatures only, the Price pyrometer is now also made to give temperatures below as well as above the freezing point of water. When the outside junction is carefully wrapped in wool of sufficient thickness, in order to maintain the temperature uniform (wool being supposed to be the best non-conductor of heat that we have, being five times as efficient as asbestos), a very reliable instrument can be made to show the temperature from 50° below zero to the boiling point of water.

Fig. 7 shows a Price pyrometer. In this illustration the connections marked "Fig. 1" show how the couples are joined together at their extremities, their separate ends leading to the meter, while "Fig. 2," in the same illustration, represents the appearance of a couple, such as is used in tempering vats, compressed steel chambers, glass and steel annealing ovens and many such places not requiring above 850° C.

Fig. 8 shows a switchboard used in connection with the Price pyrometer in case a larger number of couples are to be

connected to the same instrument, so that by means of one indicating instrument the temperature in different furnaces may be measured. The contact points of this switchboard cover both sides of the switch, similar to a knife switch. By moving the switch to the contact point, which is connected to any special furnace, the temperature of this furnace will be indicated. As many as forty or fifty couples may be connected in this way to the same outfit. The positive wires of all the couples are connected together by one leading to the black screw at the top of the switchboard, while the negative wires of all the couples lead from the lower binding post of the instrument.

Instruments with Radiation and Resistance Pyrometers

Last, not least, some experiments were made by Mr. C. H. Wilson, of the Wilson-Maeulen Co., immediately after the meeting. As was mentioned above a type of the Féry radiation pyrometer was exhibited by this company. In this connection it is interesting to state that the Cambridge Scientific Instrument Co. has now succeeded in making this instrument recording. As far as we know this is the first and only type of radiation pyrometers which has been rendered a recording instrument. Radiation and optical pyrometers are specially useful, or rather, absolutely necessary, in all cases in which it is necessary to measure the temperature from a distance. With optical pyrometers the measurement is restricted to a wave length in the visible spectrum, while the Féry pyrometer is based on the total radiation in connection with the Stefan-Boltzmann law. For description of the Féry pyrometer reference must be made to our Vol. III., page 478.

Mr. C. H. Wilson exhibited the Féry radiation pyrometer in practical use and showed how easily it is handled. He also made a very interesting experiment on the determination of the temperature of a gas-heated muffle furnace, both with a Féry radiation pyrometer and with an electric resistance pyrometer. It was shown that for temperatures up to that for which the resistance pyrometer can be used, both instruments, although based on entirely different principles, give the same results.

Primary Cell.

A paper on the Decker primary cell was then presented by Prof. Francis B. Crocker. There is nothing new of an electrochemical nature in it. The whole novelty rests in the method of mechanical construction, but Prof. Crocker pointed out that just in this point all former primary cells were found wanting.

The cell of Mr. F. A. Decker, of Philadelphia, is of the double-fluid type, with zinc plates in dilute sulphuric acid and graphite plates in a solution of sodium bichromate and sulphuric acid. Each zinc plate and the dilute sulphuric acid surrounding it are contained in a flat porous cup. In order to form such a cup two unglazed earthenware plates with thickened edges and diagonal strengthening ribs are shaped separately in steel moulds, a special clay mixture being employed to obtain any desired porosity. These plates are made extra thick to prevent warping in burning and produce true flat surfaces and straight edges. They are then united to form a flat cup, after which it is ground down on each surface to the desired thinness. In point of fact the finished walls are very thin, so that light will show through them, which enables the degree and uniformity of thinness to be readily determined. The grinding operation involves little time or skill, being performed by half-grown boys. In this way a flat porous cup is obtained, perfectly true on all sides and with walls much thinner than could possibly be made if it were attempted to form a complete cup in the first place. The graphite plates are placed directly against the outside walls of the porous cups. The thinness of the porous wall and the nearness of the zinc and graphite plates results in a low internal resistance of the cell.

The containing vessels are made of hard rubber, and special provision is made for the supply of the electrolyte to the cells through pipes, etc. A whole battery consisting of several cells is so assembled that with all its connections and porous cups, it constitutes one substantially integral unit, with no loose joints or parts. For details of construction, the reader may be referred to the complete article of Prof. Crocker, in *Electrical World*, Oct. 13.

In tests made by Prof. Crocker, in which the cell was discharged from 1.9 to 1.3 volts, an output of 14.7 ampere-hours per pound of total weight was found. A cell weighing 17 pounds gives 150 amp-hours at an average [illegible] volts, or about 250 watt hours, which is equivalent to one-third of a horse-power-hour.

The weight of zinc, sulphuric acid and sodium bichromate required to give 1 hp-hour in the Decker cell, assuming all materials to be thrown away after being used once, cost about 35 cents. The corresponding cost for the Lalande cell is $5.30. Since the Lalande cell has numerous applications, it is thought that the Decker cell will have an even broader field of usefulness, especially for automobiles and for lighting railway trains.

The paper was briefly discussed by Mr. E. A. Sperry, Prof. Tucker and Mr. Carl Hering.

After, at motion of Mr. Morehead, thanks had been expressed to Columbia University and all others who had shown courtesies to the Society during the convention, the meeting adjourned.

Excursions and Social Functions.

Prof. S. A. Tucker, of Columbia, courteously showed to the visiting members the finely equipped laboratory of applied electrochemistry in his charge, in the basement of Havemeyer Hall. A full description of the laboratory was given in our May issue (Vol. IV, p. 175).

On Monday afternoon, after Dr. Baskerville's lecture, the members visited the Waterside station of the Edison Company, under the auspices of Dr. Elliott. The visitors saw the operation of the plant at its busiest time, namely, just when the evening lighting load was coming on.

On Monday evening a subscription dinner was served in the hall of the Liederkranz Society. Though the attendance was small—below forty—the affair was extremely enjoyable. Mr. H. B. Coho again distinguished himself as toastmaster.

A most interesting excursion was made on Tuesday afternoon to the large electrolytic copper refinery of the American Smelting & Refining Co., at Maurer, near Perth Amboy, N. J.

Many members attended the dinner and the reception given to Sir William Perkin, on the evenings of Saturday and Tuesday, respectively.

Electric Smelting of Iron Ore.

In the development of an iron industry California, like Canada, has been handicapped in the past by the lack of coke and the same trend of ideas which led the Canadian Government to a systematic and careful study of electric smelting of iron ore has induced progressive Californians to try electric smelting. Water-powers for cheap generation of electric power are abundant in California as in Canada.

This interesting work is to be carried out by the Northern California Power Company, the president of which, Mr. H. H. Noble, takes very great interest in this development, in connection with the Shasta Iron Company.

The Héroult process will be used and the work will be conducted in a general way along the lines of the Canadian Sault experiments.

Mr. R. A. Turnbull, who participated in the Sault experiments at Sault Ste. Marie, and who is now Dr. Héroult's representative in Canada, is at present in charge of the engineering side of the undertaking. M. Paterson, who has been connected with Dr. Héroult's electric smelting plants in Europe, will be in charge of the work at the spot.

The Extraction of Metallic Sodium.

By C. F. Carrier, Jr.

A brief consideration of the purpose and scope of this article must be its own excuse for being. While the literature of sodium is quite abundant, and even extended treatises have been devoted to this single topic, there is no one publication of any kind in which the bibliography has been given with any degree of completeness. It is hoped to present, in systematic form, a bibliography of the subject more complete than has previously been published, together with brief descriptions, which will make it a history of the art.

The classification used is based on the Dewey Decimal System, as elaborated by Mr. Adolph L. Vöge, of the Concilium Bibliographicum, of Zurich, Switzerland. For the sake of completeness, all processes will be mentioned, whether they have had successful technical application or not, but the relative importance will be to some extent shown by the space devoted to the description, in absence of some specific comment on that phase of the matter.

Scheme of Classification.

669.	Metallurgy.
669.733.	Sodium.
669.733.(09)	History of Sodium.
669.733.36	Sodium by Chemical Processes.
.362	Reduction of Sodium Carbonate.
.363	Reduction of Sodium Hydroxide.
.364	Reduction by Carbides.
669.733.372	Electrothermic Processes.
669.733.374	Electrolytic Fusion Processes.
.374.2	Electrolysis of Fused Salt.
.374.3	Electrolysis of Fused Carbonate.
.374.4	Electrolysis of Fused Hydroxide.
.374.5	Electrolysis of Fused Nitrate.
.374.6	Compound Electrolysis.
669.733.375	Electrolytic Solution Processes.
.375.2	Using Sodium Amalgam.
.375.3	Using Organic Solvents.
669.733.6	Preparation of Sodium Alloys.
669.733.9	Uses of Sodium.

References:

Proc. Am. El-ch. Soc., 8, 161 (1905).
This Journal, 3, 107 (1905).

History, 669.733 (09).

Electrochemistry was very essentially concerned in the discovery of metallic sodium by Sir Humphrey Davy, who subjected the fused hydroxide to electrolysis, but it was over eighty years before any successful technical application of this discovery was made. While his work was successful in this qualitative way, nothing much seems to have been accomplished in fixing conditions required, or in making clear the theory of the dissociation. Davy supposed that if the caustic soda were anhydrous, no hydrogen would be evolved at the cathode. It is also worthy of note that he made use of the principle of using the current to heat the bath as well as accomplish the electrolysis. The eighty years following were taken up with chemical processes and a few spasmodic attempts at electrochemical processes.

References:

Phil. Trans. (1808); 1 and 133
(1809); 39
(1816); 16
Ostwald's "Klassiker," No. 45, 55.
"History of the Alkali Metal Industry," Lumiere Electr., 44, 179 (1892).

Note.—The number of the volume to which a reference is made precedes the ",", the page follows the "," and the year is enclosed in the ().

Chemical Processes, 669.733.36.

It would be interesting, as well as instructive, to make a detailed study of the strictly chemical processes for the extraction of sodium, but owing to the vast quantity of literature pertaining to this branch, only a brief outline can be given, sufficient merely to show their relation to processes involving the thermic action of the electric current.

Gay-Lussac and Thenard reduced sodium hydroxide in an iron tube by treating with iron filings at a white heat, but at the suggestion of Curaudau, carbon was used in place of iron as the reducing agent. Various improvements in the details of the process were made by Brunner, Wohler, Donny, Mitscherlich and others, but the metal was first prepared on a large scale by Deville. By mixing chalk with the charge he kept it loose and open, thus facilitating the reduction. If this is not done, the carbonate or hydroxide will melt and collect in the bottom of the retorts. The sodium thus produced was used in the extraction of aluminium. Castner improved upon this by using a "carbide of iron" as the reducing agent. This carbide was prepared by heating a mixture of tar and iron filings to a high temperature in a closed vessel. The reduction can be carried out at a lower temperature than when carbon is used, and thus there is a saving of fuel, and still more important, the life of the retorts is greatly lengthened. Several other inventors were working on the chemical processes at about this same time. O. M. Thowless melted the alkali compound to be reduced, and then added it gradually to a mass of incandescent carbon. In this way he claims to avoid the formation of certain undesirable compounds in the retort. H. S. Blackmore heated a charge made up of calcium hydroxide, ferric oxide, sodium carbonate and carbon. In the process of Netto the temperature is kept lower than in the other processes, so that only the hydroxide will be reduced, the carbonate formed during the process being allowed to collect in the bottom of the retort and drawn off from time to time.

References:

Thorpe, "Dictionary of Applied Chemistry," 3, 422.
Castner, U. S. P., No. 342,897, of June 1, 1886.
Thowless, U. S. P., No. 380,775, of April 10, 1888; U. S. P., No. 380,776, of April 10, 1888.
Blackmore, U. S. P., No. 391,110, of Oct. 16, 1888.
Netto, U. S. P., No. 460,985, of Oct. 13, 1891.

There are also several chemical processes that are indirectly related to electrochemistry in that they use electrochemical products as reducing agents, or depend upon some hypothetical action of electrical energy.

The process of Hasenclever & Sons utilizes static discharges to "increase the activity of reducing material." No data are available showing the efficacy of this expedient. Ger. P. No. 65,921 of Jan. 11, 1891.

The use of calcium carbide for the reduction of melted sodium hydroxide was patented by Wolfram in 1898, and the reaction has been studied by Kügelgen. The products of the reaction are calcium oxide, sodium, hydrogen, carbon monoxide and a little calcium carbonate. Kügelgen also found that fused sodium chloride was also partially reduced by calcium carbide, but in both cases the yield is very poor.

References:

Wolfram, Ger. P. No. 101,374 of March 23, 1898 (also Eng. P.).
Kügelgen, Zeit. f. Elektr., 7, 576 (1901).

For the preparation of chemically pure sodium, Beketoff and Scherbatscheff reduce sodium aluminate with magnesium and distill off the sodium. Bulletin of the Academy, St Petersburg (1894).

A modification of the Wolfram process invented by Moeser and Eidmann has been assigned to the Chemische Fabrik Greisheim Electron. This consists in reducing the fluoride by means of calcium carbide instead of the hydroxide. It is especially recommended for potassium reduction. U. S. P. No. 710,493 of Oct. 7, 1902. Also patents in Ger., Fr., and Eng.

Another process acquired by the Chemische Fabrik Griesheim Electron is that of Specketer and Weber for reducing the alkali fluorides in a suitable still by means of aluminium. U. S. P. No. 730,979 of June 16, 1903. (Also patents in Ger., Fr., and Eng.).

With the introduction of the electrolytic processes for the preparation of aluminium, the main use of metallic sodium was taken away and the older chemical processes were abandoned, possibly never to be revived. Castner could not compete with the Hall process, and so attempted to discover a cheaper process of extracting sodium. This led to the successful electrolysis of fused sodium hydroxide, but did not reduce the cost of the metal enough to enable it to maintain its place as a reducing agent in the preparation of aluminium. Through force of necessity, new uses for sodium had to be found which were the beginnings of the modern sodium industry.

Electrothermic Processes, 669.733.372.

The great difficulty with the chemical processes was their low thermal efficiency and the tremendous wear and tear on the retorts. It would seem that the possibilities of efficient internal heating by means of electric energy would have enabled a modified chemical process to remain in the field, but the problems of condensation and furnace construction give no present evidence of solution.

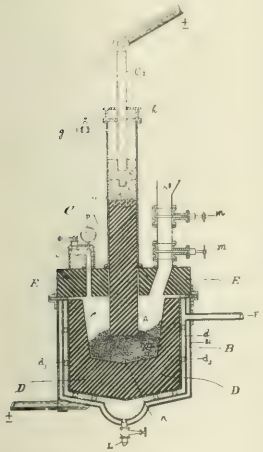

FIG. 1.—COWLES FURNACE

Probably the first suggestion of using electrical energy to supply the necessary heat for reducing sodium oxide by carbon was that of C. A. Faure in 1882. He placed the charge in a magnesia retort and passed the current through it, or through resistors in the walls of the retort, a current of nitrogen or hydrogen being used to carry the metallic vapor to the condenser. Eng. P. No. 6,058 of 1882 and No. 5,489 of 1883.

The first application for a patent in this country was that of Bradley and Crocker in 1885. They accomplished the heating by passing the current through the walls of the wrought-iron tube that formed the retort. There seems to be no self-evident cause for failure of this process, but it is possible that the tubes, being hotter than the charge, were very rapidly consumed in reducing the sodium, or the thermal efficiency may have been too low. U. S. P. No. 335,499 of Feb. 2, 1886. This Journal 3; 128 (1905).

Parker and Robinson make use of the arc as a source of heat, and in order to "distribute the heat," have the charge of sodium oxide and carbon resting upon a mass of molten metal, which acts as one pole of the arc. Eng. P. No. 11,707 of July 23, 1889.

The most extensive work on electrothermic processes has been done by A. H. Cowles. His first patent was applied for in 1895, and was followed by several others involving details of the apparatus, a progressive study of which shows some of the difficulties involved in the electrothermic distillation of sodium. The fundamental idea is to reduce sodium aluminate by carbon under the influence of electrothermic heating, condense the sodium vapor and tap off the aluminium carbide which collects in the bottom of the furnace. By adding some non-volatile metal, as iron, copper, etc., an alloy of aluminium may be obtained instead of the carbide. The special features of the furnace (Fig. 1) are the devices for making it possible to work above atmospheric pressure. The carbon electrode C is introduced into the furnace by means of an iron rod passing through a stuffing box. To prevent reducing the pressure when fresh material is added, two valves [illegible] between the hopper N' and the furnace chamber. The cover is made air tight by the ring of insulating material between it and the body of the furnace. A safety valve and pressure gauge control the pressure in the furnace chamber. The sodium vapor and the gases produced from the reaction are forced through the porous carbon walls of the chamber and the sodium is condensed in the hollow space between the outer walls, the furnace b being surrounded by water jackets to keep the metal walls cool enough to act as condensers. A slow current of neutral gas is forced into the furnace by the pipe g in order to protect the electrode as much as possible.

References:

U. S. P. No. 676,575-76-77 of June 18, 1901.

U. S. P. No. 673,761 of May 7, 1901; also Eng. patents.

I. L. Roberts supports the charge in his furnace by means of a grating of conductors that are heated by the passage of the current. The reduced material gathers in the chamber beneath the grating. U. S. P. No. 629,394 of July 25, 1899.

In a French patent, H. Becker proposes to reduce sodium manganate, chromate, tungstate, aluminates, etc., by carbon, condensing the sodium and tapping off the manganese, chromium or alloy which may be formed according to the charge. The great advantage claimed by the inventor is the simultaneous production of two valuable metals, that by the conditions of the process and their own specific properties are readily separated from one another. In his monograph, "Die Elektrometallurgie der Alkalimetalle," Becker makes no specific description of the kind of distillation furnace in which he proposes to carry out this process. He states that the process has been tried successfully on an experimental scale. On pages 109-111 of the above mentioned monograph the Cowles process is assigned to the year 1901, and the Becker process to 1899, but it is to be noted that the Cowles patents were applied for in 1895.

References:

Fr. P., No. 288,274, of April 27, 1899.

This Journal 1; 422 and 575 (1903).

To the best of the author's knowledge, none of the electrothermic processes are in commercial operation at the present time. It is probable that simply as sodium producers the high temperature and the deterioration of the apparatus prevent competition with the more simple electrolysis of fused sodium hydroxide.

Electrolytic Fusion Process, 669.733.374

In addition to the work done on the electrolysis of the more common sodium salts in the fused condition, some research work has been recorded on the more expensive compounds. Faraday investigated the sulphate, carbonate, borate and phosphate. Brester worked on the sulphate. The cyanide was found by Linnemann to be an excellent raw material if the commercial side is ignored. Extensive researches by Hittorf on fused nitrate, chlorate, etc., were incidental to his monumental work on the migration of ions. Kleiner-Fiertz took out an English patent on the electrolysis of cryolite as a source of sodium, and further work on the same line was done by Houmpe in Italy.

References:

Faraday, Ostwald's "Klassiker," No. 86

Hittorf, Pogg. Ann. d. Phys., 72: 481 (1847)

Linnemann, Jr. f. Prak. Chem. 73, 415 (1858)

Kleiner Fiertz, Eng. P., No. 8,533 of 1886

Houmpe, Electricita (1889), 307

Brester, Jahresbericht f. Ch. (1866); 84.

Electrolysis of Fused Sodium Chloride, 669.733.374.2.

The electrolysis of fused salt has so many inherent advantages and such brilliant commercial possibilities that it is not at all surprising to find that this class embraces by far the largest number of attempts to prepare metallic sodium. With an ore almost chemically pure, in almost unlimited quantities at a very low price, and which is a good conductor when melted, the problem looks alluring, but in spite of these advantages not a pound of sodium is commercially produced by direct electrolysis of fused salt.

The difficulties are numerous. The melting point of sodium chloride is above bright redness, and has been fixed by various authorities at 750° to 950° C. The most reliable figures are between 800° and 850°. It is, however, not an easy matter to prepare a bath of fused salt. The selection of materials for furnace construction is made difficult by the fact that substances which will resist the action of melted salt, such as fireclay, magnesia, etc., are readily attacked by sodium, and those which will withstand sodium are useless against melted salt. Even iron is somewhat corroded by sodium at the melting point of salt, and it is totally unfit to use as the container for the electrolyte.

The above difficulties might be overcome by cooling the walls of the container and maintaining a protective coating of the electrolyte, but the behavior of the electrolyte itself is a far more serious matter. It has been thought by some that the sodium unites with sodium chloride to form a sub-chloride, but the existence of this compound has not been conclusively proven. The Lorenz metal fog theory accounts for the depolarization quite as satisfactorily, and the high temperature of the bath is certainly a condition favorable to the formation of a metallic suspension in the electrolyte. It is also possible that chlorine is to some extent soluble in fused salt. Regardless of its exact nature, however, it is certain that some depolarizing agent is formed at one or both of the electrodes and diffuses or circulates to the other electrode, thus reducing the efficiency. This problem has been quite thoroughly investigated by A. Fischer in the Aachen Laboratory, but the author neglected to give the essential data showing the results obtained by overcoming the unfavorable conditions which he himself had noted. Zeit. f. El-ch., 7; 347-354 (1900).

As early as 1844, a patent appeared on the extraction of metals from fused salts. The inventor, Napier, used a container of conducting material lined with a refractory material on the sides, which was a non-conductor, the bottom serving as the cathode. Inasmuch as the metal was to be collected on the bottom, it is not probable that the apparatus was intended for the separation of sodium, but it is worthy of mention as being the first application, or conception, of that principle, frequently made use of in the design of furnaces for the electrolysis of fused salts. Eng. P. No. 10,362 of 1844, and No. 684 of 1845.

The first patent for a process specifically intended for the separation of sodium was that of James Watt. His apparatus consisted of a covered iron pot in which was a partition extending part way to the bottom, an electrode being located in each of the compartments thus formed. The electrolyte was a fused halogen salt of the alkali or alkaline earth metals. As cathode material carbon was used and the anode was of gold. The sodium liberated at the cathode was removed from the cathode compartment by distillation. In the light of present knowledge, this apparatus is self-evidently impracticable, and was undoubtedly purely hypothetical. Eng. P. No. 13,755 of Sept. 25, 1851.

The process of B. J. Dickson may have been worked upon an experimental scale at least, for he made use of sodium hydroxide as well as sodium chloride in his patent specifications. It is to be noted, however, that the use of sodium hydroxide as a raw material for the production of sodium had been made the subject of a patent long before the Castner process. Eng. P. No. 2,962, Aug. 13, 1892.

Jablochkoff attempted to improve on the apparatus of his predecessors by enclosing the two electrodes in tubes which might be made of clay, porcelain or metal. The products of the electrolysis were to be carried off by the tubes surrounding their respective electrode, the sodium being vaporized. An earthenware pot heated externally held the electrolyte. This apparatus did not overcome any of the difficulties which made the previous processes impossible, and introduced some complications of its own. Dingler's Poly. Jr., 251; 422 (1884).

Hoepfner attempted to overcome the disturbing influence of the chlorine by using a heavy metal as a soluble anode. The metal was placed in the bottom of the melting pot and the electrolyte above it. Upon electrolysis, the chlorine would, of course, unite with the heavy metal, but he assumed that the chloride would remain in a layer at the bottom of the pot while the alkali metal was liberated at the cathode above. The special metal cited as an example was copper. This would dispose of the chlorine very nicely, if it were only possible to get around the fundamental principle of electrodeposition and separate sodium from a bath containing copper. Ger. P. No. 30,414 of March 21, 1884.

The pioneer American worker in this field was A. J. Rogers. His apparatus was an elaboration of the same general features as are involved in the apparatus of Watt, and was no more successful. The chief distinction was that Rogers introduced the salt into the electrolytic cell in the molten state. U. S. P. No. 296,357 of April 8, 1884.

The first real steps in advance were developed in the apparatus of J. Omholt, although it must be said that, from the practical standpoint, he was not much more successful than those who had gone before. The improvements seem to be due to his having had a somewhat better understanding of the materials of construction which must be used in order to meet the special conditions involved. The most marked variation from previous furnaces is in the substitution of reverberatory heating in place of heating by conduction through the walls of the vessel containing the electrolyte.

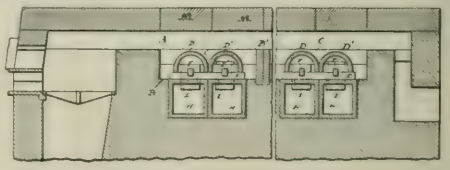

Fig. 2.—Olmholt Apparatus.

The hearth of the furnace (see Fig. 2) consists of a number of half retorts D D' arranged in pairs. These retorts resemble muffles with the flat side removed. The open side is placed downward and the molten electrolyte forms a liquid seal around the entire lower edge. The retorts are of fireproof material and are lined with a material containing carbon. Each retort contains an electrode of carbon or a "suitable metal" F F'. The liquid seal prevents the furnace gases from coming in contact with the products of the electrolysis and also the products of the electrolysis from mixing with each other. The sodium is collected in the chamber H by passing from the retort D through the pipe L. It appears improbable that this furnace was ever given a practical trial, but a number of disadvantages can be pointed out. Electrothermic heating is far more efficient than reverberatory heating that has to pass downward through the wall of a retort. The structural material problem has not been entirely solved, for there are some parts of the furnace that have got to withstand both salt and sodium. Maintenance charges would probably kill the process even if it were not a physical impossibility to separate sodium from melted salt with the electrodes in such close proximity to each other. U. S. P. No. 382,183 of May 1, 1888. Also patents in Ger., Eng., Fr. and Belg.

In 1886, F. Fischer described an apparatus that so closely resembles that of Watt (1851), although it did not have a gold anode, that he cannot be said to have profited by the experience of the intervening thirty-five years. Wagner-Fischer Jahresber (1886); 222.

Ludwig Grabau took out a series of patents between 1886 and 1890, covering improvements in both apparatus and electrolyte, which has proved one of the most nearly successful of any of the attempts to produce sodium by the direct electrolysis of sodium chloride. After trying to use bell-shaped cells made of earthenware for collecting the sodium and separating the anode and cathode products, he attributed the rapid destruction of the cathode bell more to the leakage of the current through the walls of the bell than to the corrosive action of the fused electrolyte. To prevent this, the inventor formed a coating of solid electrolyte over the surface of his "pole cell," which is impervious to the current and which is maintained by cooling the annular metal walls of the cell.

The construction of the apparatus may be readily comprehended from Fig. 3. It consists of a melting pot A closed by the cover D which supports the "pole cell" and the anodes.

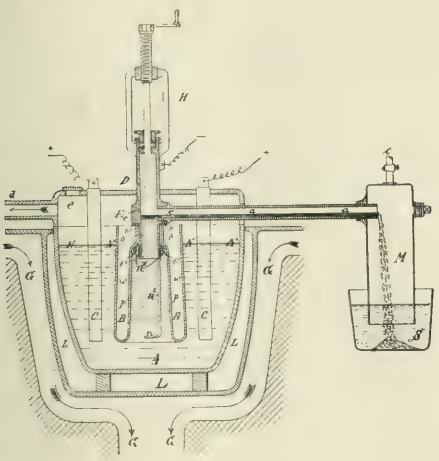

FIG. 3.—GRABAU CELL.

The pole cell consists of the double-walled bell B, the cathode n, an outlet pipe a and the boring arrangement H for removing obstructions in the bell. The electrolyte is solidified on the surface of the bell and the current is forced to go around through the electrolyte instead of through the walls of the bell.

To further facilitate the process, Grabau made use of a mixed electrolyte to reduce the melting point, but this principle is not original with him, for it had already been applied by Bunsen and Matthiessen in 1885. In this way he claims to have obtained an ampere efficiency of 95 per cent. The best results were obtained by the use of a bath of the following composition: $3(NaCl,KCl) + SrCl_2$, that is, one molecule of strontium to three molecules of the alkali chlorides mixed in molecular proportions. The sodium produced contains about 3 per cent of potassium but no traces of strontium. Barium chloride does not melt as easily as strontium chloride, and calcium chloride is so difficult to free from water that he prefers strontium chloride to the other two as the alkaline earth chloride to be used. The chief causes of the commercial failure of the process seem to be the high voltage required and the careful attendance made necessary by the small size of the units and the frequent clogging of the cells.

References:

Chem. Zeitschr. (1902)
Chem. Centralblatt (1891, II); 96.
Zeit. f. angew. Chem. (1891), 38 and (1891), 223.
U. S. P. No. 464,096 of Dec. 1, 1891.
U. S. P. No. 464,097 of Dec. 1, 1891.
U. S. P. No. 465,369 of Dec. 15, 1891.
Also patented in Ger., Eng., Fr., Italy, Aust., Switz., Belg. and Spain.

Another worker during the year 1886 was Max Sprenger. He suggested the electrolysis of fused salt in a vacuum, but no data are available concerning his results. D. R. P. 39,554 of 1886.

Several processes proposed at about this time are not worth special description, but will be briefly mentiond. Bull electrolyzed fused salt and condensed the sodium vapor. Neiwerth vaporized the salt and subjected this vapor to some kind of electrolytic action. Stoerk and Seidler do not seem to have profited by the failure of their predecessors.

References:

Bull, Eng. P. No. 10,735 of June 7, 1892.
Seidler, Oesterr.-Ung. Priv. of March 3, 1887.
Niewerth, Ger. P. No. 65,921 of Jan. 11, 1891; Eng. P. No. 23,773 of 1892.
Stoerk, Ger. P. No. 68,335 of Aug. 24, 1892; Zeit. f. angw. Chem. (1893), 356.

The next attempt was a retrogression, for the state of the art in 1888 should have enabled Hornung and Kasemeyer to design a more feasible apparatus. No better illustration can be cited which will better show how important it is for an inventor to be familiar with all the work that has previously been done on the subject. Their apparatus consists of a cylindrical containing vessel which acts as the anode and a concentrically arranged iron tube which acted as cathode. The electrolyte was replenished by adding salt through this tube. A circular concentric partition at the top of the tube, dipping a little below the surface of the electrolyte, separated the products of the electrolysis from each other. The required voltage was probably low enough to suit the most exacting commercial limits, but it hardly seems probable that any sodium was ever liberated in this apparatus by electrolysis of fused salt, for the electrodes were in such close proximity that there was nothing to hinder complete depolarization. Even with the easily melted hydroxide it is necessary to have a diaphragm, when the electrodes are near each other, as in the Castner process.

References:

Ger. P. No. 46,334 of Jan. 29, 1888
Zeit. f. angw. Chem. (1889); 216.

An apparatus devised by Rennerfelt in 1893 was a return to the Grabau type of seven years previous. This inventor did not seem willing to concede the point made by Grabau that earthenware is not impervious to the electric current when placed in a bath of fused salt, so he coated the outside of his bell shaped cathode compartment with a layer of semi-material. The inside of the bell acted as cathode, instead of having an electrode within the bell, as in the Grabau apparatus. The containing vessel acted as anode and the metal was removed from the bell by suction. The difficulties of this type of apparatus have been shown in the description of the Grabau apparatus. U. S. P. No. 495,000 of April 18, 1893.

In 1893, Borchers designed a furnace on the lines of the best theory of the time, which for a short time held a place in practice, but it could not compete with processes using fused hydroxide. This apparatus was "U" shaped, one leg being considerably larger than the other. The larger leg contained the carbon anode and was made of chamotte to withstand the action of the fused salt and the chlorine. The smaller leg was of iron, and itself formed the cathode; the

two sections were joined together by means of an ingenious arrangement of clamps and a water-cooled ring. The difficulties were the perishability of the apparatus and the high voltage required. The current density at the cathode was 50 amps. per 100 sq. cm. and the fall of potential about 10 volts per cell.

In Becker's "Elektrometallurgie der Alkalimetalle," page 37, the author gives the figures for the cost of plant and the cost of production using the Borchers apparatus. The labor charges are far too low for American conditions, and the amount allowed for depreciation is probably too large, but his figures make the cost per pound of sodium the very high figure of 26 cents. If these figures are accurate, it is very easy to see why this process was not used in practice for any extended period.

References:

Zeit. f. angw. Chem. (1893): 486.

Borchers "Elektrometallurgie," 3d ed., p. 45.

This was followed in 1896 by an apparatus devised by Hanekop, which the inventor claimed to be an improvement on the apparatus of Borchers. He made use of an earthenware melting vessel divided into several parallel compartments by hollow partitions reaching nearly to the bottom. The anodes and cathodes were in separate adjoining compartments. As there is no known fireproof earthenware that will withstand the action of both sodium and chlorine, the apparatus was predestined to failure. The use of a hollow partition between the electrodes was suggested by Grabau, so that both originality and improvements are not in evidence. Ger. P. No. 98,766 of July 4, 1896.

From the work of Rogers, in 1883, no attempt seems to have been made in the United States to electrolyze molten salt directly until 1898. Paul Danckwardt, in this year, attempted to overcome the structural material difficulty by using water jackets for the walls and the partition between the anode and cathode compartments. The bath was kept melted by an oil or gas flame playing upon its surface. This has the disadvantage that a portion of the salt would be converted to carbonate, and the carbonate thus formed would take part in the electrolysis, yielding carbon dioxide at the anode. The red-hot carbon anodes would react with the carbon dioxide and form carbon monoxide, with a correspondingly rapid corrosion of the anode. Another difficulty seems to have been that both the electrodes were located near the bottom of the bath, thus providing the best possible conditions for complete depolarization, as pointed out in the work of Fischer (l. c.). U. S. P. No. 607,506 of July 19, 1898.

Another furnace of American origin appeared in 1902, being the invention of Roepper and Scholl. This furnace was not intended specifically for the electrolysis of sodium chloride, but may be applied, with modifications, to the electrolysis of any fused salt. It consists of a reverberatory furnace on the hearth, of which is a container for the melted bath. One portion of the surface of the bath is exposed to the heat in the furnace, and the remainder is readily accessible from the outside, so that the "removable electrolytic box" can be replaced without interrupting the heating. The chief objects of the invention are to provide external heating in an economical manner and to make all parts replaceable without cooling down the furnace. U. S. P. No. 699,851 of May 13, 1902.

The effort to electrolyze sodium chloride directly seems to have ended in Europe with Borchers in 1893, little or nothing having since been done, but the last attempt in this country was that of Cowles, about 1900. The original feature of this process is the separation of the sodium at a porous carbon cathode, at such a high temperature that the sodium is vaporized. The vapor penetrates through the porous cathode and is condensed in a chamber beneath it, the cathode serving also as the bottom of the fusion chamber, as shown in Fig. 4. U. S. P. No. 679,253 of July 23, 1901.

The commercial separation of metallic sodium by the direct electrolysis of melted sodium chloride still remains an unsolved problem. It is, however, the most logical manner of reducing the cost of producing sodium, and it is probable that further work will be done along that line, presumably with ultimate success, but the wrecks by the wayside are grim testimonials of the difficulty of the problem. An indirect solution will be discussed under 669.733.374.6, compound processes.

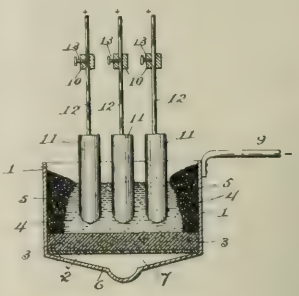

FIG. 4.—COWLES CELL.

The following references must be added to complete the bibliography of this class:

Beketow, Lumiere Electr. 30; 532 (1888).

Nuernberg Consortorium, Ger. P. No. 160,540 of July 20, 1904.

Matthiesen, Ann. d. Chem. u. Pharm. 93; 277 (1855).

Faraday, Ostwald's "Klassiker," No. 86, pp. 43, 48 and 65.

(To be concluded.)

Metallurgical Calculations.

By J. W. Richards, Ph. D.
Professor of Metallurgy in Lehigh University.

THE BESSEMER PROCESS

The outlines of this famous process are known to every educated person; the mechanical and most of the chemical details are familiar to most technologists; the exact way to run the converter is the source of income to hundreds of superintendents of works, and yet the quantitative side of the chemical and physical operations involved is mastered by very few.

To state the case briefly, melted pig iron is put into the converter, numerous air jets are blown through, the impurities of the iron—carbon, silicon, manganese and, in a special case, phosphorus—oxidize relatively faster than the iron, and the final product is usually nearly pure iron. This is recarburized to steel by spiegel-eisen. During the blow very little free oxygen escapes from the converter, and the gases produced are principally nitrogen, carbon monoxide and some carbon dioxide, while some hydrogen may come from the decomposition of the moisture of the air. The silicon, manganese, phosphorus and iron form silica, manganous oxide (MnO) principally, ferrous oxide (FeO) principally, phosphorus pentoxide, P^2O^5, which go into the slag, while a little Fe^2O^3, Mn^3O^4 and SiO^2 escape as fume.

The applications of calculations to this process are numerous and important. They include such subjects as the amount of air theoretically required per ton of iron, the dimensions and power of the blowing engines, the weight of slag produced, the balance sheet of materials, the balance sheet of heat evolved and distributed, the radiation losses, the discussion of the efficiency of the various impurities as heating agents in the process.

Air Required

Basing our calculations on the analysis of the pig iron used, and assuming it to be blown to pure iron, we must next assume the probable loss of iron itself in the blow. This varies considerably, from 1 to 10 per cent on the pig iron used in ordinary practice, but as much as 20 to 25 per cent in some carelessly run "Baby" Bessemers in steel-casting foundries. The silicon all oxidizes to SiO^2; iron mostly to FeO, and a small part, say not over one tenth, to Fe^2O^3; manganese mostly

to MnO, a small part, up to one-fourth, may form Mn^2O; phosphorus forms only P^2O^5; carbon burns mostly to CO, but from one-fifth to, at times, nearly one-half, burns to CO^2. When all the calculated oxygen has been found, the blast to contain it can be calculated, if it is assumed that no free oxygen escapes from the converter; at times, however, up to one-third of all the oxygen blown in may thus escape, but this is very exceptional, ordinarily less than one-fifth thus escapes, and often none at all.

Problem 61.

Pig iron containing 3.10 per cent carbon, 0.98 silicon, 0.40 manganese, 0.101 phosphorus and 0.06 sulphur is blown in an acid-lined converter, to metal practically free from carbon, silicon and manganese, but no sulphur or phosphorus is eliminated. To get the minimum and the maximum amounts of air which could be needed, make the following assumptions:

	Case 1.	*Case* 2.
Per cent of iron oxidized to Fe^2O^3	1	15
Proportion of iron oxidized to Fe^2O^3	none	one-tenth
Proportion of Mn oxidized to Mn^2O^3	none	one-fifth
Proportion of C oxidized to CO^2	one-fifth	one-half
Proportion of O^2 escaping in the gases	none	one-third

Requirement: (1) Find the weight of dry air needed per metric ton of pig iron blown, in each case, and its volume at 0° C. Express the results also in pounds and cubic feet per ton of 2,000 pounds.

Solution:

Case 1.

Oxygen needed per 1,000 kg. of pig iron:

C to CO^2 . . . 6.2 kg. $\times \frac{32}{12} =$ 16.53 kg.

C to CO . . . . . 24.8 " $\times \frac{16}{12} =$ 33.07 "

Si to SiO . . . . 9.8 " $\times \frac{32}{28} =$ 11.20 "

Mn to MnO . . . 4.0 " $\times \frac{16}{55} =$ 1.16 "

Fe to FeO . . . . 10.0 " $\times \frac{16}{50} =$ 2.86 "

64.82 "

N^2 accompanying = 216.07 "

Air needed . . . . . = 280.89 "
Volume at 0° C. . . . . = 217.2 m^3
Volume needed per 1000 oz. Av. . . . = 217.2 ft^3
Volume needed per 2000 lbs. Av. . . . . = 6,950 ft^3

Case 2.

Oxygen needed per 1000 kg. of pig iron:

C to CO . . . . . 15.5 kg. $\times \frac{32}{12} =$ 41.33 kg.

C to CO^2 . . . . . 15.5 " $\times \frac{16}{12} =$ 20.67 "

Si to SiO^2 . . . . . 0.8 " $\times \frac{32}{28} =$ 1.20 "

Mn to MnO . . . . . 3.2 " $\times \frac{16}{55} =$ 0.93 "

Mn to MnO . . . . . 0.5 " $\times \frac{16}{[illegible]} =$ 0.73 "

Fe to FeO . . . . . 13.5 " $\times \frac{16}{56} =$ 3.86 "

Fe to FeO . . . . . 13.5 " $\times \frac{48}{112} =$ 5.79 "

O^2 unused (one-half sum of above) . . . . . = 59.73 "

179.20 "

N^2 accompanying = 597.33 "

Air used . . . . . = 776.53 "
Volume at 0° C. . . . . = 600 m^3
Volume used per 1000 oz. Av. . . . = 600 ft^3
Volume used per 2000 lbs. Av. . . . . = 19,200 ft^3

For temperatures of the air above 0° C., a corresponding increase in the volume used would be found. Since this is net air received by the converter an allowance for loss of 10 to 25 per cent (in exceptional cases 50 per cent) would be needed to get the piston displacement of the blowing engines. The above figures are the maximum and minimum for this quality of pig iron only, blown in an acid-lined converter; other qualities of pig iron might require a little more or less, and if blown in a basic-lined converter considerably more, to oxidize the phosphorous. The detailed calculations can be made in each specific case.

Air Received.

The converse of the preceding proposition is to take an actual case, in a Bessemer converter, and to calculate how much air is being received. This will serve as a check on the blowing engines, since the volume received, divided by the piston displacement, gives the volume efficiency of the blowing plant. To make the calculation we need to know the weight and analysis of the pig iron and the analysis of the blown metal, in order to find the weights of impurities oxidized, also the average composition of the escaping gases, to find the proportion of carbon burning to CO^2 and of unused oxygen; also the composition of the slag, to get therefrom the weight of iron oxidized and the weight of slag, if practicable, but this can sometimes be calculated; also the weight and composition of the fume, if it is considerable. The temperature of the air entering the blowing cylinders, its hygrometric condition and the barometric pressure, should also be noted.

Problem 62.

At the South Chicago works of the Illinois Steel Co. [illegible] paper by Howe, in Trans. American Institute of Mining Engineers, XIX [1890-91], p. 1,127), the charge weighed 22,800 pounds, and contained 2.08 per cent carbon, 0.93 silicon, 0.48 manganese, 0.10 phosphorus, and 0.06 sulphur. After blowing 9 minutes 10 seconds the bath contained 0.04 per cent carbon, 0.02 silicon, 0.01 manganese, 0.11 phosphorus and 0.06 sulphur. The slag formed contained 63.56 per cent silica, 3.01 alumina, 21.39 FeO, 2.63 Fe^2O^3, 8.88 MnO, 0.90 CaO and 0.30 MgO, of which the MnO, CaO and MgO and part of the SiO^2 came from the lining. The gases, analyzed dry, show the average composition during the blow of

CO	[illegible]
CO^2	[illegible]
H	1.30
N^2	[illegible]

[illegible] were free from fume.

The piston displacement during the blow was 190,406 cubic [illegible] in engine room 36° C., barometer 29 per cent, barometer 756 millimeters.

Requirements: (1) The weight of oxygen acting on the bath during the blow, and the theoretical volume of air at standard conditions to which this would correspond, per 2,000 pounds of metal blown.

(2) The volume of moist air at the conditions of the engine room, received by the converter during the blow, and the volume efficiency of the blowing machine and connections.

(3) The weight of slag produced and the loss in weight of the lining by corrosion during the blow.

Solution: (1) The percentages of impurities left in the bath are so small that we can take them as equivalent to the same percentages reckoned on the original weight of the bath. If they had been larger it would be necessary to assume an approximate loss of iron during the blow, find the final weight of the bath and reckon the percentages on this revised weight.

Making this assumption, we know at once the weights of carbon, silicon and manganese oxidized, but we do not know the weight of iron lost. That follows, however, from a consideration of the slag, for the manganese and iron in the slag are derived only from the metallic bath, and the analysis of the slag practically gives us the relation between the weights of manganese and iron in it; since we know the weight of the former oxidized, the weight of iron lost can be calculated. Thus the slag contains:

MnO	8.88 per cent	=	6.88 per cent	Mn
FeO	21.39 "	=	16.64 "	Fe as FeO
Fe^2O^3	2.63 "	=	1.84 "	Fe as Fe^2O^3

The loss of manganese being 0.42 per cent = 94.5 pounds, the loss of iron is:

$$94.5 \times \frac{16.64}{6.88} = 228.6 \text{ pounds Fe as FeO}$$

$$94.5 \times \frac{1.84}{6.88} = 25.3 \text{ " Fe as } Fe^2O^3$$

The remaining item still undetermined is the weight of carbon oxidizing to CO^2 and to CO. The gas analysis shows 5.20 per cent CO^2 to 19.91 per cent CO^2, and since equal volumes of each of these gases contain equal weights of carbon, it follows that $\frac{5.20}{25.11}$ of the total carbon is present in the gas as CO^2, and the rest as CO. Since the total carbon oxidized is 22,500 × 0.0204 = 661.5 pounds, we have

$$661.5 \times \frac{5.20}{25.11} = 137.0 \text{ pounds C burning to } CO^2$$

$$524.5 \text{ " " CO}$$

Weight of oxygen absorbed by the bath:

C to CO^2	137.0 pounds	× 8/3	=	365.3 pounds
C to CO	524.5 "	× 4/3	=	699.3 "
Si to SiO^2	207.0 "	× 32/28	=	236.6 "
Mn to MnO	94.5 "	× 16/55	=	27.5 "
Fe to FeO	228.6 "	× 16/56	=	65.3 "
Fe to Fe^2O^3	25.3 "	× 48/112	=	10.8 "
			=	1404.8 "
N corresponding			=	4682.7 "
Dry air corresponding			=	6087.5 "
Volume at 0° C			=	75,483 ft³
Weight of O^2 per 2000 pounds			=	124.9 lbs. (1)
Volume of air per 2000 pounds			=	6,710 ft³ (1)

(1) This result is for comparison with data of Problem 611.

(2) The nitrogen in the gases can be obtained from its volume relation to the carbon, and from this we can calculate the total volume of blast used.

Weight of carbon in 1 cubic foot of gases:

$$0.0520 + 0.1991 = 0.54 = 0.1356 \text{ oz. Av.}$$

Volume of gases produced at standard conditions:

$$\frac{661.5 \times 16}{0.1356} = 78,057 \text{ ft}^3$$

Volume of nitrogen at standard conditions:

$$78,057 \times 0.7350 = 57,372 \text{ ft}^3$$

$$\text{Weight} = 57,372 \times 1.20 = 72,289 \text{ oz. Av.} = 4,518 \text{ lbs.}$$

The question now is, how much nitrogen is contained in each cubic foot of air in the engine room. Knowing that, we are prepared to calculate the volume of this actually received by the converter:

Barometric pressure	=	756 m. m.
Tension of moisture (44 × 0.5)	=	22 "
Tension of air present	=	734 "
Tension of nitrogen present (734 × 0.792)	=	580 "

Weight of nitrogen in 1 cubic foot:

$$1.26 \times \frac{273}{273 + 30} \times \frac{580}{760} = 0.8495 \text{ oz. Av.}$$

Volume of air actually received:

$$\frac{4.518 \times 16}{0.8495} = 85,095 \text{ ft}^3 \quad (2)$$

Volume efficiency of machinery:

$$\frac{85,095}{190,406} = 0.447 = 44.7 \text{ p. c. (2)}$$

(3) The slag contains 8.88 per cent of MnO, equal to 6.88 per cent of Mn, as already calculated. But 94.5 pounds of manganese is oxidized, therefore, the weight of slag produced is:

$$94.5 \div 0.0688 = 1374 \text{ pounds.} \quad (3)$$

Weight of SiO^2 in slag (1374 × 0.6356)	=	873 pounds
SiO^2 from the Si of bath (207.0 + 236.6)	=	444 "
SiO^2 corroded from the lining	=	429 "
CaO, Al^2O^3 and MgO (1374 × 0.0427)	=	59 "
Loss in weight of lining	=	488 " (3)

Blast Pressure

It is necessary to use sufficient blast pressure to overcome the static pressure of the metallic bath, plus that of the slag formed, also the back pressure in the converter, to give the necessary velocity to the air in the tuyeres and to overcome friction in the same. When the tuyeres are near to the surface of the bath, pressures of 1 or 2 pounds will run the small converter, but the ordinary converter with bottom tuyeres requires from 15 to 30 pounds pressure per square inch (1.054 to 2.108 kg. per square c. m.). We will consider the latter, the more frequent and the more complex case to discuss.

The metal lies 12 to 24 inches (30 to 60 c. m.) deep in the converter. Since its specific gravity melted is about 6.88 (Roberts and Wrightson), the ferro static pressure which it exerts is practically 0.25 pounds per square inch for each inch depth of metal, or 0.00688 kilos. per square centimeter for each centimeter depth.

The slag lying on the metal has a specific gravity melted of approximately half that of the metal. Its amount may vary from 5 to 10 per cent of the weight of the metal treated, in an acid-lined converter, up to from 15 to 35 per cent in a basic-lined vessel. Taking into account its lower specific gravity, its depth in the converter may be, therefore 10 to 20 per cent the depth of metal in an acid-lined vessel, and 30 to 70 per cent in a basic-lined converter; but the static pressure exerted would be only in direct proportion to the relative weights, i. e. 5 to 10 or 15 to 35 per cent of that exerted by the metal

The static pressure of the slag may, therefore, be reckoned as 0.125 pounds per square inch for each inch in depth, or 0.00344 kilos. per square centimeter for each centimeter depth, and the depth of slag as lying between the following extremes.

		Acid Lined.	*Basic Lined*
Depth of metal	Inches	12 to 24	12 to 24
	Centimeters	30 to 60	30 to 60
Depth of slag	Inches	1.2 to 4.8	3.6 to 16.8
	Centimeters	3.0 to 12.0	9.0 to 42.0

The probable depth of slag can be calculated in any particular case, when the composition of the metal to be blown is known, its approximate depth in the vessel, and the approximate composition of the slag to be formed.

The back pressure of gases in the converter itself, that is, their static pressure, will vary with the shape of the converter and the size of the free opening for their escape into the air. A measurement at the Pennsylvania Steel Co's works gave 0.275 pounds per square inch, but it is not stated just how the measurement was made. If we know approximately the volume of gas which must escape from the converter and from its temperature and the time and the size of the outlet calculate its velocity, the static pressure giving it this velocity can be calculated as

$$h = \frac{V^2}{2g}$$

in which, if V is in feet per second, $2g = 64.3$ and the resultant pressure is in feet of the hot gas; if V is in meters per second, $2g = 19.6$, and h is in meters of the hot gas. Knowing the approximate specific gravity of the hot gas (weight of 1 cubic foot in pounds or of 1 cubic meter in kilograms) the static pressure is obtainable in pounds per square foot or kilograms per square meter.

Illustration: The gases escaping from a converter are 78,057 cubic feet (standard conditions), and weigh 0.0801 pounds per cubic foot (standard conditions). They escape from the converter at an average temperature of 1,500° C., and the opening is 24 inches in diameter. What is the gaseous back pressure in the converter? Time of blow 9 minutes 10 seconds.

Volume of gas at 1,500°:

$$78{,}057 \times \frac{1500 + 273}{273} = 506{,}940 \text{ ft}^3$$

Volume per second:

$$506{,}940 \div 550 = 921.7 \text{ "}$$

Area of outlet:

$$2 \times 2 \times 0.7854 = 3.1416 \text{ ft}^2$$

Velocity (assuming 0.9 coefficient):

$$921.7 \div 0.9 \div 3.1416 = 326 \text{ ft. per second}$$

Head of hot gas giving velocity

$$h = \frac{326 \times 326}{64.3} = 1{,}653 \text{ ft.}$$

Pressure of this column per square foot:

$$1.653 \times \frac{273}{773} \times 0.0801 = 20.4 \text{ pounds}$$

Per square inch = 0.14 "

This solution omits a consideration; the velocity of the gases in the body of the converter is neglected. This is somewhat counterbalanced by the great friction of the gases against the sides of the converter, so that the one item tends to neutralize the other. If the interior were 8 feet in diameter, the velocity of the gases therein would average only some 20 feet per second, showing the above corrections to be practically negligible, since the pressure thus represented would be only 0.1 per cent of the total obtained above.

The pressure necessary to force the blast through the tuyeres is calculable on principles similar to the above; the differences are that the blast, at temperatures varying from 100° C. in the blast box to possibly 200° at its entrance into the metal, is divided up into, say, 150 streams of approximately 1 centimeter (0.4 inch) in diameter, the length of tuyere being some 50 centimeters (20 inches). The formula applied to that used for chimney draft, or rather, frictional resistance in a chimney, applies to this case.

$$h = \frac{V^2}{2g} \frac{K L}{D}$$

in which h is the head in terms of the air passing, V the velocity, 2g the gravitation constant, L the length of the tuyere, D its diameter, and K the coefficient of friction, which latter is for relatively smooth flues 0.05 (Grashof), and may be so assumed here.

Problem 63.

In the converter mentioned in Problem 62, where 22,500 pounds of metal was blown in 9 minutes 10 seconds, using as therein calculated 85,095 cubic feet of air, at 36° C., and producing 1,374 pounds of slag, assume the inside diameter of the converter as 7 feet, and that the bottom contains fourteen tuyere blocks, each containing eleven openings of 0.5 inch diameter each; blocks 24 inches long. Assume back pressure in converter 0.14 pounds per square inch, total blast pressure in equalizing reservoir 27 pounds per square inch. Temperature of air in the tuyeres 150° C.

Required: (1) The pressure needed to overcome the head of metal and slag.

(2) The pressure absorbed in friction in the tuyeres.

(3) The pressure represented by the velocity of the blast in the tuyeres.

(4) The loss of pressure from the reservoir to the blast-box.

(5) The distribution of the total pressure.

(6) The length of the blow if the blast pressure were reduced to 20 pounds.

(7) The length of the blow if the pressure were maintained at 27 pounds, but twenty-one tuyere blocks (each with eleven ½-inch holes) were used.

Solution: (1) At the start there is 22,500 pounds of melted metal, the volume of which will be

$$\frac{22{,}500}{6.88 \times 62.5} = \frac{22{,}500}{430} = 52.3 \text{ cubic feet}$$

The depth of metal, the inside diameter being 7 feet, is

$$\frac{52.3}{7 \times 7 \times 0.7854} = \frac{52.3}{38.5} = 1.350 \text{ feet}$$

$$= 16.4 \text{ inches}$$

Static pressure $= 16.4 \times 0.25 = 4.1$ lbs. per sq. in.

The slag, formed during the first half of the blow, weighs 1,374 pounds, and has a volume of

$$\frac{1374}{3.44 \times 62.5} = \frac{1374}{215} = 6.4 \text{ cubic feet}$$

The depth of slag, at its maximum, will be

$$6.4 \div 38.5 = 0.167 \text{ feet}$$

$$2.0 \text{ inches}$$

Static pressure $2.0 \times 0.125 = 0.25$ lbs. per sq. in.

The static pressure during the blow will, therefore, be 4.1 pounds to start with, increasing during the first half of the blow to 4.35 pounds, and staying practically constant at this, and, therefore, will average

$$\frac{4.1 + 4.35}{2 \times 2} + \frac{4.35}{2} = 4.29 \text{ pounds} \quad (1)$$

(2) Each of the $14 \times 11 = 154$ tuyeres receives 85,095 $\div$ 154 $\div$ 550 $=$ 1.005 cubic feet of air per second, measured at 36° C. At 150° C. this volume is

$$1.005 \times \frac{273 + 150}{273 + 30} = 1.375 \text{ cubic feet}$$

And the velocity in the tuyere:

$$1.375 \div \frac{0.5 \times 0.5 \times 0.7854}{144} = 1009 \text{ feet per second}$$

The head absorbed in friction in the 24-inch tuyeres will be

$$h = \frac{1009 \times 1009}{64.3} \times \frac{0.05 \times 2}{0.0417} = 37{,}893 \text{ feet}$$

Changing this pressure of air at 150° C. to pounds per square inch we have:

Weight of 1 cubic foot of air at 0°	=	0.0808	pounds
Weight of 1 cubic foot of air at 150°	=	0.0522	"
Weight of air column = 37,893 × 0.0522	=	1978	"
Pressure in pounds per square inch	=	**13.70**	" (2)

(3) The pressure absorbed as velocity has already been expressed in getting the friction in the tuyeres. The velocity head is simply:

$$h = \frac{V^2}{2g} = \frac{1009 \times 1009}{64.3} = 15{,}835 \text{ feet}$$

which becomes in pressure

$$15{,}835 \times 0.0522 = 826.6 \text{ pounds per square foot}$$
$$= \mathbf{5.73} \text{ pounds per square inch} \quad (3)$$

(4) The remaining part of the 27 pounds pressure used is lost between the blast reservoir and the entrance to the tuyeres. It is

$$27.00 - (13.70 + 5.73 + 4.29 + 0.14) = \mathbf{3.14} \text{ pounds} \quad (4)$$

(5) Distribution of blast pressure:

Lost between reservoir and tuyeres	=	3.14	pounds	= 11.6%
Absorbed in friction in the tuyeres	=	13.70	"	= 50.7%
Absorbed in velocity in the tuyeres	=	5.73	"	= 21.2%
Static head of liquid bath	=	4.29	"	= 15.9%
Velocity head of issuing gases	=	0.14	"	= 0.6%
		27.00	"	= 100.0%

(6) All the items of absorption of pressure are proportional to the square of the velocity of the gases, excepting the static pressure of the bath. It remains constant at 4.29 pounds. If the total pressure were reduced to 20 pounds, there would be only 20 − 4.29 = 15.71 pounds pressure to give velocity and overcome friction, instead of 27 − 4.29 = 22.71 pounds. The relative quantities of air blown through in a given time in the two instances would be practically proportional to the square roots of the two effective pressures, *i. e.*:

$$\sqrt{22.71} : \sqrt{15.71} = 1 : 0.832$$

And the times of the blows inversely as the latter:

$$550 \text{ sec.} \div 0.832 = 673 \text{ seconds}$$
$$= \mathbf{11 \text{ min. } 13 \text{ sec.}} \quad (6)$$

(7) If the tuyere area were increased 50 per cent, then the velocity of the air in the tuyeres would be decreased one-third, assuming the amount of air passing to be unchanged. This would decrease the pressure absorbed in friction, and in giving velocity in the tuyeres to (0.67)² = 0.444 of its former amount. The 19.43 pounds previously absorbed in these two items would then become 19.43 × 0.444 = 8.61 pounds, and the total pressure needed to run the converter just as fast as before would be 27 − 19.43 + 8.61 = 16.18 pounds per square inch. If, however, the pressure were maintained at 27 pounds, giving 27 − 4.29 = 22.71 pounds to overcome frictional resistances and to give velocity, then the velocity and consequent amount of air blown through by this 22.71 pounds pressure would be in proportion to the square roots of these two available pressures, *i. e.*, as

$$\sqrt{22.71} : \sqrt{16.18} = 4.29 : 4.29 = 1 : 1.38$$

The duration of the blow would be just that much shorter; *i. e.*:

$$550 \text{ sec.} \div 1.38 = 398 \text{ seconds}$$
$$= \mathbf{6 \text{ min. } 38 \text{ sec.}} \quad (7)$$

Flux and Slag.

No flux is used in the acid-lined converter, and the silica, iron oxides and manganese oxide formed in the converter unite to a silicate slag which corrodes the lining and thus takes up more silica. The slag being analyzed, its weight is obtained by considering the percentage of manganese which it contains, because the weight of manganese oxidized is known definitely from the analysis of the bath; it is usually all oxidized. Calculation of the weight of slag cannot be based upon the silica, because an unknown amount comes from the lining; nor upon the iron, because the weight of iron left in the converter is not definitely known. Having the weight of the slag, analysis tells us the total weight of silica in it, as also the amount of iron. The silica in the slag minus that formed from silicon in the pig iron, gives silica corroded from the lining.

In the basic Bessemer converter, phosphorus is nearly entirely eliminated from the metal, so that, assuming none to be volatilized, the amount going into the slag is known, and using the slag analysis the weight of slag can be calculated. In this process the lining is mainly dolomite, containing CaO and MgO, in proportions easily determined by analysis. The weight of slag being known, the amount of corrosion of the lining can be determined from the percentage of magnesia therein, which may be assumed as practically coming entirely from the lining; it cannot be told from the CaO in the slag, because nearly pure CaO is added during the blow, and some of it, a variable amount, gets blown out of the converter. For the same reason it is not possible to base a good calculation of the weight of slag on the lime alone which is added, because of the indefinite proportion of it which is blown out. The weight of slag may also be gotten from the silica or manganese oxide in it, assuming these to come almost entirely from the oxidation of silicon or manganese.

Lime must be added as flux, in the basic converter, to protect the lining and to make the slag so basic that the percentage of silica in it is below 15 per cent, phosphoric acid below 20 per cent, and lime over 50 per cent. These considerations must be balanced in each particular case.

Illustration: Pig iron blown in a basic-lined converter contained 1.22 per cent silicon, 2.18 phosphorus, 1.03 manganese and 3.21 carbon. It is blown until all of these and 2.00 per cent of iron are oxidized, and burnt lime is added to form slag during the blow. Composition of the burnt lime: MgO, 1.00 per cent; SiO^2, 2.00 per cent; CaO, 97 per cent. How much lime should be added per 10 metric tons of pig iron charged?

The slag-forming ingredients from the oxidation of the bath, and the addition of X kilos. of lime, are

SiO^2	10,000 × 0.0122 × 60/28	=	261.4	kg
P^2O^5	10,000 × 0.0218 × 142/62	=	499.3	"
MnO	10,000 × 0.0103 × 71/55	=	133.0	"
FeO	10,000 × 0.0200 × 72/56	=	257.1	"
CaO	X × 0.0700	=	0.97	X
MgO	X × 0.0100	=	0.01	X
SiO^2	X × 0.0200	=	0.02	X

Weight for slag = X + 1150.8 kg.

Corrosion of the lining will undoubtedly increase this weight, so some allowance should be made, say to increase it 5 per cent, probably an outside figure. Of this 5 per cent, half can be considered lime and half magnesia. The total weight of slag will then be 1.05 X + 1208.3, and of the ingredients principally in question:

SiO^2	= 261.4 kg	+ 0.02	X
MgO	= 28.7 "	+ 0.035	X
CaO	= 28.7 "	+ 0.995	X

To make our slag 50 per cent CaO will require the addition of enough to make

$$287 : 0.995 X = 0.50 (1.05 X + 1208.3)$$
$$X = \mathbf{1224} \text{ kg.}$$

To make a slag with at most 15 per cent of SiO^2 requires

$$204 : 0.02 X = 0.15 (1.05 X + 1208.3)$$
$$X = \mathbf{583} \text{ kg.}$$

To make a slag with at most 20 per cent of P^2O^5 requires

$$499.3 = 0.20 (1.05 X + 1208.3)$$
$$X = \mathbf{1227} \text{ kg.}$$

The larger of these three amounts would be used, with 10 per cent added to cover lime dust blown out, making 1350 kg. added, and the calculated composition of the slag:

CaO	1250 kg.	50.1 per cent
MgO	72.3 "	2.9 "
SiO^2	286 "	11.4 "
P^2O^5	469 "	20.0 "
FeO	257 "	10.3 "
MnO	133 "	5.3 "
Total	2497 "	

RECARBURIZATION.

When the bath has been blown to nearly pure iron, melted spiegeleisen is run in, to add the necessary carbon and manganese. Knowing the approximate composition and weight of the bath, and the composition of the melted spiegel, a simple arithmetical calculation would give the amount of the latter to be added, assuming no loss of carbon or manganese in the operation. But experience shows that there is some loss, and that the carbon and manganese in the finished metal are always lower than the calculated amount. An interesting field is open here for calculating the loss of manganese and carbon and the amount of oxygen which must have been in the metal to cause these losses. A tabulation of many such calculations gives the metallurgist the necessary data for assuming the average amounts of carbon and manganese lost during recarburization, under different conditions of working, such as letting the metal stand before pouring or pouring at once, turning on the blast 5 or 10 seconds to mix up the bath, etc.

Problem 64.

At the end of the blowing the converter of Problem 62 contained 21,283 pounds of metal of the composition 0.04 per cent carbon, 0.02 silicon, 0.01 manganese, 0.11 phosphorus, 0.06 sulphur, an unknown amount of oxygen (probably < 0.3 per cent) and the rest iron. There is added to it 2,500 pounds of spiegeleisen, containing 4.64 per cent carbon, 0.035 silicon, 14.90 manganese, and 0.139 phosphorus. The finished metal contained 0.45 per cent of carbon, 0.038 silicon, 1.15 manganese, 0.109 phosphorus and 0.059 sulphur. Assume no iron oxidized.

Required: (1) A balance sheet of materials before and after recarburizing.

(2) The proportions of carbon and manganese going into the finished metal.

Solution: (1)

	Blown Metal.	*Spiegel.*	*Steel.*	*Gases or Slag.*
C	8.5	116.0	106.5	18.0
Si	4.3	0.9	9.0	— 3.8
Mn	2.1	372.5	271.2	103.4
P	23.4	3.5	25.8	1.1
S	12.8	....	14.0	— 1.2
Fe	21,232	2,007	23,239	....

The differences in the sulphur, phosphorus and silicon are within the limits of error of the data, but there is no doubt as to the loss of carbon and manganese.

(2) The proportions of the [illegible] into the finished steel are

Carbon .	106.5	124	0.83	83 per cent
Manganese	271.2	374.0	0.72	72 "

The calculated percentages in the [illegible] been, and actually were

Carbon	0.53	0.45 loss	0.08 per cent
Manganese	1.38	1.15 loss	0.23 "

Concerning oxygen removed, if we assume the loss of carbon and manganese to be due to their combining with oxygen dissolved in the bath, to form CO and MnO, the percentage of oxygen thus absorbed is

By carbon	0.08	16/12	0.11 per cent
By manganese	0.23	16/55	0.12 "
			0.23

[The next instalment of these calculations will consider the thermo-chemistry of the Bessemer process.]

The Influence of Nickel and Carbon on Iron.*

BY G. B. WATERHOUSE.

The object of the following research is to help in the study of the ternary alloys of iron, carbon, and another metal or metalloid, by considering a series of steels of constant nickel with varying carbon percentages. The other elements, which are present of necessity, were kept as low and constant as possible.

The materials employed by the author in preparing the steels were puddled bar-iron, known as muck bar; wood charcoal, and an especially pure sample of electrolytically deposited nickel. By analysis it was found to contain 99.47 per cent nickel. The nickel was melted in a crucible, thoroughly mixed, and shotted to get it in a form suitable for weighing. It was decided to make a series of nine ingots, with the nickel constant at about 3.5 per cent and the carbon rising from 0.40 per cent, which was the lowest possible, using graphite crucibles, to about 1.60 per cent.

The muck bar, as may be seen in the chemical section, compares very favorably with the best Swedish iron; 89.5 pounds of it were charged into the crucible together with the calculated amount of charcoal. The furnace was a Siemens regenerative one, gas fired. When clear melted 3.5 pounds of the shotted nickel were added. Before teeming 0.5 of an ounce of aluminium was thrown in, the steel poured into a second hot crucible to ensure mixing, and then into a 4¼-inch square mould. When topped the ingots showed no sign of blowholes or piping, and the surface was good and smooth.

The fractures were so different from those of ordinary steel that they quite deceived the inspector, who usually grades the steel very accurately. After topping, the ingots were reheated, hammered down to 1½ square inch billets, and finally reheated and rolled to 1 inch diameter round bars.

An extremely interesting point was the entire absence of any difficulty due to a thick and clinging scale. This scale is mentioned by Mr. E. Windsor Richards in the discussion following the reading of Mr. Wiggin's paper. He considered it to be one of the great difficulties attending the use of nickel steel, which was then a new material, and in the case of plates it produced a rough surface. Mr. F. W. Paul, speaking on the same discussion, had experienced the same difficulty, especially with plates over ¾ inch thick. Mr. J. G. Eaton, of the United States Navy, had also noticed the same phenomenon, the rolling in of the scale producing a series of ridges on the plate surface. The difficulty has not been met with much as

* A portion of this article is reproduced from a paper by the author presented to the Iron and Steel Institute in September, 1906. The above article contains, however, considerable new matter.—Editor.

America, owing, probably, to the high manganese contents of the steels. As will be shown later, the series of steels examined in this research are very low in manganese. It is interesting to find that a small amount of aluminium has taken the place of the manganese, and that without any special precautions the steels rolled perfectly.

The plan of research was to take the bars prepared in the above manner and to examine them thoroughly, first obtaining their complete chemical analyses, and then testing them mechanically under several conditions of heat treatment. Micrographic analyses under these conditions were to be also made, and the heating and cooling curves determined. The results may be conveniently classified under the heads chemical, mechanical, microscopical and pyrometric.

Chemical.

Before the ingots were worked drillings were taken from them at a depth of 1 inch near to the top. The carbon was estimated, and the results which were obtained at the works at Syracuse are given in the first column of Table I. The other results are those found by the author in the metallurgical laboratory, Columbia University, on drillings taken from three to five places in the rolled bars, and well mixed. They are arranged in order of ascending carbon. Through an error at the mill, O and P were mislettered, so that they do not come in alphabetical order.

It is gratifying to notice the low sulphur. In the case of a coke-fired furnace it would not have been lower than 0.025 to 0.030 per cent, even with the best coke, because the steels were in the furnace for an average of 4½ hours.

The carbon in the upper members of the series existed in two forms, combined and graphitic. The percentages are given in Table II.

For the definitions of the treatment the steel has undergone reference may be made to the mechanical section.

Table I

Mark of Steel.	Muck Bar	K.	L.	M.	N.	P.	O.	Q.	R.	S.	T.
Carbon in ingot	.....	0.40	0.40	0.61	0.82	0.93	1.00	1.23	1.46	1.62	1.83
Total carbon in bar	0.12	0.41	0.51	0.63	0.70	0.92	0.97	1.24	1.54	1.64	1.82
Nickel	.....	3.70	3.70	3.76	3.81	3.70	3.75	3.81	3.82	3.82	3.70
Silicon	0.055	0.102	0.103	0.133	0.129	0.126	0.095	0.117	0.144	0.151	0.133
Manganese	trace	0.086	0.054	0.05	0.051	0.047	0.058	0.057	0.054	0.047	0.051
Sulphur	0.008	0.013	0.014	0.014	0.014	0.014	0.014	0.015	0.015	0.014	0.016
Phosphorus	0.007	0.007	0.008	0.008	0.008	0.008	0.008	0.008	0.007	0.009	0.009
Aluminium	.....	0.01	0.01	0.009	0.01	0.009	0.011	0.009	0.01	0.01	0.01

Table II.

No.	Rolled. Combined Carbon.	Rolled. Graphitic Carbon.	Submitted to Treatment A. Combined Carbon.	Submitted to Treatment A. Graphitic Carbon.	Submitted to Treatment B. Combined Carbon.	Submitted to Treatment B. Graphitic Carbon.
Q	1.24	....	1.24	....	1.24	....
R	1.48	0.04	1.21	0.31	0.50	1.02
S	1.13	0.51	0.93	0.71	0.21	1.43
T	1.02	0.80	0.91	0.91	....	....

The following results were obtained by Dr. Mathews:

Table III.

No.	Billets. Combined Carbon.	Billets. Graphitic Carbon.	Rolled. Combined Carbon.	Rolled. Graphitic Carbon.	Submitted to Treatment B. Combined Carbon.	Submitted to Treatment B. Graphitic Carbon.
Q	1.24	....	1.24	....	1.24	....
R	1.61	.47	1.44	.04	.43	1.05
S	1.09	.57	1.15	.31	.16	1.50
T	1.09	.83	1.03	.80	.19	1.73

The graphitic carbon of these steels corresponds to the temper carbon of Prof. Ledebur, now known as temper graph ite; it is insoluble in boiling hydrochloric or nitric acid. The microscope shows it to occur in an amorphous condition. The results of Table III. show that it exhibits a tendency to recombine with the iron when heated under oxidizing conditions, so that it meets Prof. Ledebur's three requirements for temper carbon.

Two pieces of "T" were taken in the rolled condition, and after submitting to treatment "A," deeply notched at intervals of 1 inch, and the Metcalf experiment performed. There were seven segments, the first was brought to a welding heat, and the last was at a low hardening heat. After quenching in cold, salt water, they were fractured and the temper graphite determined in the various segments. There was a trace in the first segment of the rolled bar, and a stronger trace in the bar previously annealed. This increased progressively, but at the sixth segment amounted to only .23 per cent and .30 per cent, respectively. The fracture at the last segment was just off color in the rolled bar, and decidedly dark in the other sample. The experiment shows that the temper graphite readily passes into solution by heating above Ar. 1.

Mechanical.

Definitions: Treatment A.—Pieces of each bar, 12 inches long, were placed in a gas-fired muffle, and raised to 1,000° C. They occupied 1 hour 25 minutes in reaching this temperature. For 25 minutes they were maintained at this heat, then removed and cooled in air. The object of this procedure was to recrystallize the steels, and to remove any distortion or strain produced during the rolling. The treatment corresponds to the "normalizing" used by Prof. Arnold in his researches, so that the results given in this paper are comparable with his.

Treatment B.—The bars were packed in a wrought-iron tube, placed in an annealing furnace at 500° C., and brought to a temperature of 870° C. to 880° C. in 5½ hours. They were kept at this temperature for 3 hours, then the tube was withdrawn and cooled in a pit with many other filled pipes, and the whole covered with ashes. In 24 hours the pipes were emptied, the steel being at about 150° C.

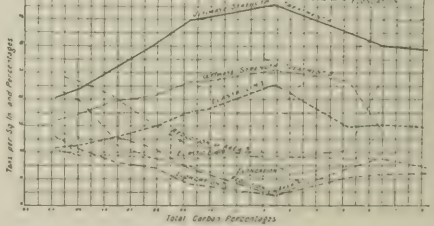

Fig. 1.—Tensile Tests.

For tensile testing the bars were turned down to a diameter of 0.564 inches, giving an area of 0.25 square inch. Through the kindness of Prof. I. A. Woodson, of the department of

Table IV.

No.	Ni.	C. C.	Gr. C.	Elastic Limit per Square Inch. Tons.	Elastic Limit per Square Inch. Lbs.	Ultimate Strength per Square Inch. Tons.	Ultimate Strength per Square Inch. Lbs.	Elong. Per Cent. in 2 Inches.	Reduction of Area. Per Cent.	Fracture.
KA*	3.70	0.41	....	21.68	48,503	45.28	90,246	26.	44.75	Light grey granular; silky edges; half cup and cone.
KB*	3.70	0.41	....	20.74	46,400	31.94	71,550	28.	49.4	Light grey granular; silky edges; half cup and cone.
LA	3.70	0.51	....	22.52	50,400	44.24	99,109	28.	50.	Light grey granular; silky edges; half cup and cone.
LB	3.70	0.51	....	18.66	41,812	34.75	77,861	25.5	46.78	Light grey granular; silky edges; half cup and cone.
MA	3.76	0.63	....	25.71	56,041	51.52	115,421	16.5	26.27	Finely crystaline; serrated edges.
MB	3.70	0.63	....	18.83	42,200	39.0	87,360	23.5	41.45	Grey granular; inclined to crystaline; serrated edges.
NA	3.81	0.70	....	29.84	66,850	62.33	138,194	14.0	21.27	Finely crystaline, with serrated edges.
NB	3.81	0.70	....	17.70	39,875	41.21	92,313	17.5	30.11	Grey granular; crystalline near the edge.
PA	3.70	0.02	....	33.94	75,000	69.42	155,522	8.0	14.08	Crystalline; radial furrows; serrated edges.
PB	3.70	0.02	....	17.71	39,688	46.45	104,068	15.0	22.61	Fine crystalline; serrated edges.
OA	3.75	0.07	....	35.76	80,120	70.01	156,827	7.5	8.39	Fine crystalline; even surface.
OB	3.75	0.07	....	17.87	41,035	46.46	103,187	13.0	22.43	Crystalline, with granular center.
QA	3.81	1.24	....	45.44	102,040	75.31	168,697	3.5	3.26	Fine crystalline; very hard fracture.
QB	3.81	1.24	....	18.38	41,186	50.74	113,673	11.0	17.21	Rather coarsely crystalline; little glistening flakes.
RA	3.82	1.21	0.31	29.25	65,530	65.01	145,642	10.5	13.9	Crystalline, but grey in color; serrated edges.
RB	3.82	0.50	1.02	17.34	38,847	46.84	104,934	14.5	20.66	Crystaline, but grey in color; serrated edges.
SA	3.92	0.93	0.71	30.18	67,605	60.59	135,734	11.5	17.86	Very dark grey granular; uneven surface.
SB	3.92	0.21	1.43	17.81	39,910	29.34	65,732	27.5	46.80	Very dark grey fracture, almost cup and cone.
TA	3.79	0.01	0.91	29.66	66,440	58.76	131,612	12.0	13.92	Very dark grey granular; edges slightly silky, and deeply serrated.

*"A" after the number of a steel signifies subjected to treatment A, while "B" signifies subjected to treatment B.

Number.[1]	Ni.	C. C.	Gr. C.	Elastic Limit.	Ultimate Strength.	E. Per Cent. in 2 Inches.	Reduction of Area. Per Cent.
T Rolled	3.79	1.03	.80	86,500	143,000	8.4	15.2
T B	3.79	.10	1.73	40,000	58,500	33.0	52.22

mechanical engineering, Columbia University, the testing was carried out in his laboratory. The results are given in Table IV., and also diagrammatically in Fig. 1.

The hardness, by Brinell's test, was decreased from 271 to 131.

He also subjected the steels to the compression test, using

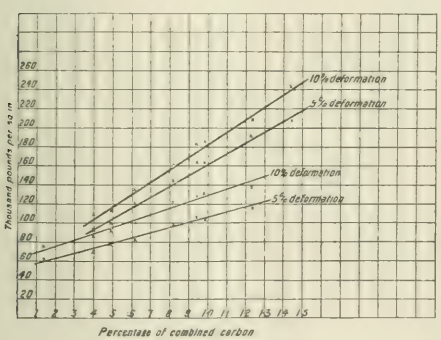

Fig. 2.—Compression Tests.

test pieces 1 inch in diameter by 2 inches high. With the maximum pressure of the machine, 200,000 pounds, the pieces failed by bending, with the exception of the R rolled. All the pieces stood at least 20 per cent compression before bending, and in Fig. 2 is shown the load necessary to produce 5 per cent and 10 per cent reduction in length with the rolled steels, and those given treatment "B." The result is practically a straight line.

(*To be concluded.*)

Obituary.—Dr. T. C. Beilstein, professor of chemistry in the Institute of Technology of St. Petersburg, died on Oct. 19, at the age of 68 years. He is best known through his work in organic chemistry, especially on the aromatic series.

[1] Results obtained by Dr. Mathews on "T."

Notes on Electrochemistry and Metallurgy in Great Britain.

(*From Our Special Correspondent.*)

The British Association Meeting.

Sir John Wolfe Barry's paper on "Standardization in British Engineering Practice" had its distinct metallurgical interest, as the engineering standards committee had been first concerned with rolled sections. The committee have laid down 16 sizes of equal angles, 30 of unequal angles, 20 of bulb angles, 6 of bulb tees, 7 of bulb plates, 8 of Z bars, 25 of channels, 36 of beams, and 20 of tees. Besides the above there are, advancing by 5 pounds at a time, nine sizes of bull-headed rails from 60 pounds to 100 pounds per yard, seventeen sizes of flat-bottomed rails from 20 to 100 pounds per yard, and five sizes of tramway rails 60 pounds to 110 pounds per yard, with their corresponding sections having a wider groove for use on curves.

As to whether standardization would bring with it disadvantages neutralizing its manifest advantages, it was pointed out that, should standarization be pushed too far it might in some instances stultify design or retard progress of invention; but provided that it is clearly understood that special circumstances must exist which warrant special designs, and as long as standardization is confined to broad principles, and the standards laid down are sufficiently numerous, the advantages must immeasurably outweigh the disadvantages. Exceptional cases may, no doubt, exist where the use of some particular form of angle or channel is justified, and where the expense entailed in cutting special rolls and delay in manufacture by the consequent changing of rolls is warranted; but in the vast majority of cases an engineer can equally well use standard sizes without in any way cramping or hampering his design.

In the case of steel standard sizes are desirable to trade, as they avoid the necessity of cutting new rolls that can be rolled to stock, thus obviating any retardation in process of manufacture or delay in delivery. As a practical illustration the testimony of a large steel maker in Scotland may be cited, who admits that since the introduction of standard sizes his firm has been able to knock no less than half of pairs of rolls, and by no means the least advantage gained is that in his works the process of manufacture is now no longer so com-

stantly interrupted as it used to be, due to the frequent changing of the rolls to produce in smaller quantities the many special sizes asked for, without any corresponding advantage to the consumer.

The next paper was by Mr. W. Rosenham, on "The Deformation and Fracture in Iron and Steel," in which the author described his method of studying metallic fractures by means of transverse sections, the accuracy of such sections being secured by first embedding the specimen in electro-deposited copper and then cutting through the compound mass. Sections of various types of fractures were shown, and the general conclusion arrived at that in mild steel, fractures occurring after considerable plastic deformation pass indifferently through both ferrite and pearlite areas, while fractures due to shock, or otherwise occurring without much distortion of the metal, pass almost exclusively through ferrite areas. In conclusion, the author showed how his method could be used to study the mode of action of cutting tools.

A paper by Mr. J. E. Stead on "Segregation in Steel Ingots and its Effect in Modifying the Mechanical Properties of Steel," was then read, in which it was pointed out that the part of an ingot which cooled first was the purest, and that segregation consisted in the transport of impure matter mostly towards the upper central axis of the ingot, and also in the central cavities. The chief elements segregating out were carbon, phosphorus and sulphur in the relative ration of 1 : 1.5 : 2, sulphur segregating most and carbon least. Gases and blow holes assisted segregation. For limiting segregation fluid compression of the ingots in the mould was the safest remedy, but part of the segregation could be reduced by the judicious use of aluminium or silicon.

The vagaries exhibted by the nickel heating coil of an electrically-heated porcelain-tube furnace afforded to Dr. H. C. H. Carpenter the subject matter of a paper on "Structural Changes in Nickel Wire at High Temperatures," and described the fundamental changes in the mechanical properties of such wires. The wire contained 98.60 per cent nickel, 1.22 per cent iron, 0.16 per cent manganese, and a trace of cobalt. Some dissolved gas or gases were also present. The diameter of the wire was 1 10th of an inch. The ultimate tensile stress was 35.2 tons per square inch, with a percentage elongation of 34.4 on 3/8 inch, and a percentage reduction of area of about 70. The resistivity at 0° C. was 9.2 michroms-cm. In actual use the wire carries 20 amps. at 50 volts pressure, and a temperature of 1,200° to 1,300° C. can be obtained in the tube.

With care the life of such a furnace is usually three to even more months. But sooner or later it breaks down. The wire is then usually found to be so brittle that it can be snapped between the fingers. Occasionally it is still tough, but has become perfectly fibrous. These changes of mechanical properties are accompanied by structural changes which have been studied with the microscope. They are the result of the combined influence of heat and electricity, and are not produced by either of these agencies singly. It appears that the changes are due mainly to two effects, viz.: recrystallization and the penetration of gases, which are themselves the result of heat and electricity on the metal. The frequent association of brittleness with gross crystallization has long been known. But the evolution of dissolved or combined gas or gases from nickel and their mode of penetration through an eventual exit from it by means of cracks between the gross crystals are, it is thought, described here for the first time.

The last metallurgical paper to which I must refer was that by Mr. W. Taylor, on "A Magnetic Indicator of Temperature for Hardening Steel," in which the author described experiments with a magnetic induction balance used for ascertaining the critical temperature in the heating of steel for hardening. Also a simple magnetic attachment to a muffle furnace, used for giving an audible signal when the steel heated in the muffle has reached its critical temperature. He took advantage of the change of magnetic permeability at the critical point, when the steel attains this temperature it becomes non-magnetic, and the sound in the telephone ceases abruptly; the steel is then quenched and hardened. The hardening in all cases investigated had been excellent, and not a single piece had been lost by cracking in the hardening. This is in fact a form of pyrometer with one—a predetermined—point of indication, but requiring continuity of attention lest the operator finds too late that the steel has lost its permeability, by which time the steel may have become too hot for quenching.

Before passing on to the work of other societies I must refer to a highly controversial paper read before another section by Mr. Swinburne, on a physical subject, "Radiation from Gas Mantles," in which their relation to radiation of selective emission, luminescence, catalytic action, resonance and unstable oxidation were discussed. On the theory of resonance, Mr. Swinburne said: "Particles of gas are vibrating and changing their velocity fast enough to produce light, but somehow do not produce it. Particles of solid, however, get into vibration synchronously with the gas particles, and thus radiate energy of the same grade as the heat of the flame. But ceria is supposed to be specially timed to vibrate with frequencies corresponding to visible radiation; so it radiates more lights than other solids. This really amounts to ceria doing the work of Maxwell's demon, except that it is working on waves instead of particles."

The concluding words were characteristic of the author: "The explanation that there is nothing anomalous about the mantle, and that it gives light just because it is very hot, has a Jordan simplicity about it which makes it unpopular compared with Abana and Pharpar of luminescence and catalytic action. But all the same, the very simple explanations are often wrong."

Metallurgical Work at the National Physical Laboratory

Among the papers put aside for subsequent use during a summer overcrowded with copy, was that of the report of the National Physical Laboratory for 1905, a very lengthy and detailed document. I can only refer to some of the points of chief electrochemical and metallurgical interest.

In the Thermometric Division, Dr. Harker extended his work on high temperature furnaces; his application of the cascade principle of heating to these furnaces has proved very fertile. The small furnaces first constructed were described in a paper read early in the year before the Royal Society. (Our Vol. III., p. 273.) These led to a redetermination of the temperature of the melting-point platinum, which was found to be 1,710° C. ± 5° C. Some experiments by Profs. Holborn and Henning, at the Reichsanstalt, published in 1905, give the figure 1,710°. Thus it seems clear that the value 1,780°, usually accepted, is 70° too high. During the year Dr. Harker also published a research on the specific heat of iron up to 1,100° C. (our Vol. III., p. 436), while another investigation, which is now practically ready for publication, is a comparison of the Kew mercury thermometer scale and the hydrogen scale, between 0° C. and 100° C.

In the Metallurgical Division, the nickel-steel research of Dr. Carpenter, Mr. Longmuir and Mr. Hadfield, was concluded during the early part of the year. It was presented to the alloys research committee in April, and read before the Institution of Mechanical Engineers at the end of the year. The committee of the laboratory were indebted to the institution for a continuation of their grant of £200, which has been employed in accordance with the wishes of the Alloys Research Committee in an investigation into the properties of the aluminium copper alloys. Good progress has been made with this, but this has only been possible on account of the help of the Broughton Copper Co. and the British Aluminium Co., who have provided all the raw materials and given every facility for the forging and rolling of the alloys. These have been made by Dr. Carpenter and Mr. Edwards at the laboratory.

In addition to the research work in connection with the

aluminium-copper alloys, to which reference has already been made, a program was outlined in connection with "An International Steel Research." This reads as follows: "Steels of carbon percentages about 1.4, 0.9 and 0.6 carbon will be examined. Two series will be tested. In the preparation of the one, aluminium has been used; in the other it has not. Polished specimens will be heated for various periods at 1,000°-1,200° under conditions which prevent oxidation and afterwards quenched in a suitable bath. The specimens will then be microscopically examined with no further treatment other than possibly etching. To characterize the austenitic structure two tests will be employed—those of chemical attack and polish. The conditions will be studied under which the austenitic structure is destroyed by heating at definite temperatures and during a definite time period. Subsequently it is hoped to examine the conditions of production of the troostosorbitic structural types, and for this purpose two more steels will be added to the series with 0.2 per cent and 1.8 per cent carbon."

The Institution of Mining and Metallurgy.

The gold medal of the institution has been awarded to Prof. Bauerman "in recognition of his services in the advancement of metallurgical science." The honor is one which is well deserved, and the recipient's many friends are cordial in their felicitations.

Market Prices During September.

The chief feature during the month was the phenomenal position of copper. Rising from the abnormal price of £84.15 per ton, the price of £90 was reached on Sept. 24, and [illegible] by the end of the month. Tin has been somewhat irregular, closing at £190, the lowest price of the month being £182 on Sept. 17. Lead has risen steadily from £17.10 to £19. Zinc is quoted at £32.15. Cleveland pig iron has been firm throughout, the highest price being £2.14.9 per ton, and the lowest £2.14.3. This does not mean stagnation, but an adjustment of supply to the present heavy demand, which is somewhat unusual. As for platinum, to pass from one of the commonest to one of the rarest metals, the price of one hundred shillings was paid at the end of the month for scrap platinum. Copper sulphate has, of course, risen in sympathy with the high copper prices to £26. Shellac is fetching £11 per cwt. No changes call for special comment in regard to the alkali trade or to the coal-tar derivations.

London, Oct. 6.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

Copper.

Pyrite Smelting Without Coke.—The third installment of this lengthy and interesting article by Mr. L. T. Wright in the *Mining and Scientific Press*, Sept. 29, is devoted to thermochemical considerations. The author discusses in particular the advantages accruing from the use of hot blast, and gives some interesting comparisons, based upon results obtained in practice, averaging several monthly runs in which the sulphur contents of the charge was much the same for a set. The relations between sulphur and coke, using cold blast, is expressed in the following table:

	S in Fe S %	*S in $Fe S_2$* %	*Total Sulphur* %	*Coke* %	*Cu. in Matte* %	*Furnace Months.*
A	13.5	2.2	15.7	8.92	34.8	4
B	11.7	6.8	18.5	8.55	29.4	6
C	12.8	7.1	19.9	8.43	27.9	4
D	7.5	17.5	25.0	7.88	22.8	4

All the sulphur in the unroasted ore is calculated as FeS_2, and all that of the roasted ore, slag, flue dust, etc., as FeS. The results are calculated to unit weights of sulphur and coke per 100 unit weights of the actual average charge. It appears from the table that the replacement of roasted pyrite by the raw pyrite is attended by an increase in the total sulphur, a diminution of coke, and a lower grade of matte, 70 per cent of the total sulphur being oxidized to SO_2. Comparing the extreme numbers of the table it is found that 1 unit weight of sulphur oxidized replaced 0.159 part of coke. Theoretically from the heats of combustion, if it had been oxidized by the reaction $FeS + 3O = 113{,}000$ Cal., it should have replaced 0.53 part coke, containing 83 per cent of carbon. If it had been oxidized by the reaction $FeS + 3Fe_2O_3 = 85.2$ Cal., it should have replaced 0.4 part of coke. On comparing the results A and C in the table a difference of one unit weight of sulphur oxidized is seen to have replaced 0.166 part of coke instead of 0.53 or 0.4 part respectively. On comparing A and B it appears that one unit of sulphur replaced 0.188 part coke. A similar comparison between the sulphur and coke in the case of warm blast is given in the following table:

	S in Fe S	*S in $Fe S_2$*	*Total S*	*Coke*	*Matte % Cu.*	*Blast*	*Rate of Smelt'g Tons per Sq. Ft. per 24 Hrs.*
E	5.51	15.10	20.07	4.0	31.0	28.4	8.58
F	7.20	12.60	19.80	4.5	31.6	28.6	8.41
G	7.15	11.30	18.45	4.8	31.4	28.4	8.47

These results have been obtained with warm blast in three furnaces similar in design and operation, each result being the average of six months' work with the same furnace. The three furnaces are stated to have been working under the same conditions, and to have exhibited much greater regularity in respect to coke consumption day by day than was shown by the same furnaces when they were working previously under cold blast. As in the case of cold blast, the coke consumption diminishes with addition of sulphur to the charge, special attention being given to reducing the coke consumption to a minimum. The average rate of smelting shows that the furnaces ran with great regularity and very fast. The percentage of sulphur oxidized to SO_2 is calculated as 71 per cent. A difference of one unit weight of sulphur oxidized, comparing the extreme results, is equal to a difference of 0.508 part of coke. This result shows quite a contrast to those obtained with cold blast, and seems to the author to illustrate again the value of the warm blast, which insures more active and regular smelting of the charge by increasing the temperature in the smelting zone. The author also makes a calculation to ascertain the vanishing point of carbonaceous fuel on a pyrite charge in order to find out how much additional sulphur should replace the 4 per cent coke used when 20.67 units of sulphur existed in 100 units of charge and 14.67 units of sulphur were oxidized, or how much sulphur must be oxidized per 100 units of charge to enable the coke to be entirely eliminated. From the extremes in the table given above he calculates that an additional 7.86 units of sulphur oxidized would, if the same relation between sulphur and coke continued, be sufficient to replace all the coke. Thus other things being equal, 14.67 + 7.87 = 22.54 of sulphur oxidized per 100 units of charge should enable the charge to be self-smelting. Thus the author [illegible]

be possible, and often the furnaces ran with just that amount of oxidation. A little excess, say 25 units of sulphur per 100 of charge, is, of course, better. The author devotes the latter part of the paper to the discussion of a heat-balance sheet for pyrite smelting per unit charge, and concludes by remarking that the use of warm blast is a logical sequence of a theoretical examination of the heat and the other reactions involved, and that these in their turn explain our practical experience that, without coke, and with cold blast, pyrite is not self-smelting.

A Hot-Blast Stove Efficiency Test.—The matter of using cold or hot blast in the operations of pyrite smelting is one of considerable interest, and opinions pro and con have been frequently expressed. Recently the consensus of opinion seems to tend towards the adoption of the hot blast. In the *Mining and Scientific Press,* Sept. 15, 1906, Mr. Herbert Haas gives an account of an efficiency test made in a U-type of stove of his own construction, at the blast furnace plant in the Afterthought Smelter, at Ingot, Shasta County, Cal. During the months of March to July, 1905, the working conditions of the stove were determined by careful records of the average volume of air heated per 24 hours, the average air pressures at the blowers, inlet and outlet of hot-blast stove and tuyeres, the average temperature of the inflowing and outflowing air, the average temperature and composition of the combustion gases at the stove-flue outlet and the average heat value and quantity of the wood burnt. He finds the following heat balance for the stove: Heat furnished by 8.5 cords wood: 2,800 pounds × 60 per cent = 1,680 pounds; 1,680 pounds × 0.4536 kg. × 8.5 cords = 6,477 kg.; 6,477 kg. × 3,800 kg. Cal. = 24,614,120 kg. Cal. Heat used, air: 7,500 cub. ft. × 0.028317 cub. met. = 212.4 cub. met. min.; 212.4 cub. met. min. × 0.2375 (284° — 19° C.) × 1.440 min. = 12,247,904 kg. Cal. Water in wood: 2,800 pounds × 40 per cent × 8.5 cords = 9,520 pounds; 9,520 pounds × 0.4536 = 4,318.3 kg. H_2O. Regnault's formula: W = heat units (kg. Cal.) required for raising 1 kg. water at t° C. to saturated steam at T° C, 606.5 — 305 (T° — t° C.) = 606.5 — 305 (200 — 19 C.) × 4,318.3 = 661.7 × 4,318.3 = 2857.419 kg. Cal. Combustion gases at 200° C. computed from analysis of gases and wood substances (cellulose), which contained 44.4 per cent carbon, 49.4 per cent oxygen, 6.2 per cent hydrogen, 24,614 kg. nitrogen × 9.2438 × 200° C. = 1,200,178 kg. Cal.; 3,451 k g.carbon dioxide × 0.2169 × 200° C. = 149,704 kg. Cal.; 3,200 kg. water × 0.475 × 200° C. = 304,000 kg. Cal. Thus nitrogen 1,200,178, carbon dioxide 149,704 and water 304,000, a total of 1,653,882 kg. Cal. The excess of air supplied to the fire-box carried away 618,500 kg. Cal. As tabulated in the following table:

Air	19,247,904 kg. Cal.	= 78.27	of A
Water	2,857,419 "	11.6	"
Combustion gases	1,653,882 "	6.7	"
Excess of air	618,500 "	2.5	"
Radiation, etc., to balance	236,415 "	1.0	"
24,614,120 kg. Cal.	= A = 24,614,120 "	100.0	"

These results show that 78.27 per cent of the heat value of the wood was transmitted to the blast furnace in the form of heated air. In the opinion of the author the excess of water in the wood is responsible for the low figure, as in his opinion with coal or oil fuel the absolute efficiency would probably exceed 85 per cent. The 19,247,904 kg. Cal. would equal 2426 kg. coke with a calorific value of 8,000 kg. Cal., or 2.65 tons at 2000 pounds. The price of coke delivered at the smelter was $21.00 per ton or, for 2.65 tons, $55.65. The operation of the hot-blast stove cost $8.00 per 24 hours, being $17.65 in favor of hot blast.

Ore Conveying at the Blast Furnaces at the Cananea Copper Co. In the course of an illustrated article on the Cananea Mining Camp in the *Engineering and Mining Journal,* Oct. 6, 1906, Mr. Dwight E. Woodbury gives a description of a new application of the conveying belt system for bedding the ores and charging them into the blast furnace. The plant consists essentially of two systems of rubber belt conveyors and trippers. One belt system is used for taking the ore from the hopper-bottom bins, into which it is received from the railway dump cars, and passing it through the sampling mill. There the coarser lumps are crushed and an accurate sample is taken. The conveyor then stacks the crushed and sampled ore on long ore beds of triangular cross-section, which are built up by a tripper, which moves back and forth on an overhead belt over the center line of each ore pile. In its shuttle-like passage back and forth it drops ore always on the crest of the pile. Successive additions of ore are thus disposed in concentric layers, thus giving a uniform composition for all cross-sections from one end of the pile to the other. Three such stacking conveyors are provided, each for a pile of 8,000 tons, a total bedding capacity of 24,000 tons. The second system of belts has been installed for conveying the smelting mixture from the reclaiming machine, as it attacks the pile, to the feeding devices at the blast furnaces. A conveyor belt passes along a trench which runs on one side of each ore pile. The reclaiming machine is a great girder-like car, which spans ore floor and trench, and, as it moves longitudinally towards the ore pile to be reclaimed, it cuts a slice clear across the bottom of the pile by means of a scraper-conveyor. It then delivers the material in a continuous stream into a hopper which feeds the belt-conveyor in the trench. A rake-like tickler keeps the end face of the pile sloping towards the machine, so as to present a thin edge for the plow blades of the scraper conveyor, and also prevents undercutting and caving. The carriage is actuated by electric motors, the speed of cutting and advance being readily regulated by the operator, who is guided by signals automatically set at a semaphore bin near the furnaces. The semaphore bin is placed between two of the belts in the series leading to the furnaces, and is supported on springs which allow a few inches of vertical movement, depending on the weight of ore contained in the bin, and thus actuating the semaphore signal arm up or down, according as the weight of the ore decreases or increases. The furnaces will be equipped with a mechanical feeding device, consisting of a long, narrow box on each side, of a size to hold a furnace charge. This can be set in a vertical position to be filled from the conveyor belt and then swung down to a horizontal position opposite each furnace door, whence the charges are dropped into the furnace. The old-style furnaces will be served by dropping the charges in front of the feed doors from overhead measuring boxes filled by the conveyors. The charges are then to be fed into the furnaces by hand. The ore-feeding system is thus based on using measured and not weighed charges. As, however, the measuring boxes can be checked against the known weight of the ore bed, it is expected to keep well within allowable limits. It is stated that it has been found possible during the last year or two to reduce materially the amount of barren flux introduced into the furnaces, so that the concentrates fed in now run about 9.5 per cent copper, 22.8 per cent silica and alumina, 28.5 per cent iron, 31.8 per cent sulphur, and a small amount of silver. Raw ores running above 5 per cent are smelted directly. The great bulk of the ore smelted is concentrates.

The Kiddie Hot-Blast System for Copper Smelting Furnaces. The advantage of hot blast for copper smelting furnaces seems to be pretty well established, but the cost of obtaining it in hot-blast stoves has militated against its introduction. A system in use at the Tyee Copper Co. Smelting Works, at Ladysmith, Vancouver Island, B. C., invented by Mr. T. Kiddie, and described by E. Jacobs in *Engineering and Mining Journal,* Sept. 29, depends upon utilizing the waste heat from the blast furnaces, by running a steel pipe, 6 inches in diameter, the entire length of the dust chamber, and return.

The pipe is designed so as to give the greatest heating surface and at the same time keep its cross-section down to the lowest practical limit. Its shape is such that the distance through which the heat must radiate to its center is reduced to a minimum, that is, approximately 4 inches in a circular pipe of equal area. As far as the cost of installation is concerned the author claims that it is approximately 50 per cent less than with the ordinary hot-blast U-pipe stove, and that it can be applied to any modern copper-blast furnace without disturbing the brick work of the furnace above the feed floor. On account of the comparatively low temperature, namely, 400 to 600° F. of the gases passing through the dust chamber, and the long travel and small cross-section of area of the heating pipe, the wear and tear is reduced to the lowest limit. In the arrangement of the pipe the lowest part of the double horizontal flue leading from the dust chamber to and over the furnace is kept sufficiently high above the charge in the furnace to prevent any burning of the steel plates. The outer jacket, which is in sections, can be readily taken apart if necessary, and all the joints are equally accessible. The author gives the following results obtained by three months' hot blast practice, in order to show the saving of coke, and the superior oxidizing power of the installation under consideration as compared to cold blast under similar furnace conditions: Running with hot blast—The charge consisted of 37 per cent roasted ore, containing 6.32 per cent sulphur and 4.08 per cent copper; 63 per cent raw ore, containing 16 per cent sulphur and 4.08 per cent copper; coke used, calculated on the ore smelter, 9.30 per cent; coke used, calculated on the mixture smelted, 7.70 per cent. The matte produced contained 42 per cent copper, sulphur to copper, 3.04 sulphur to 1 copper. Running with cold blast—The charge consisted of 63.62 per cent roasted ore, containing 6.32 per cent sulphur and 4.08 per cent copper; and 36.38 per cent raw ore, containing 16 per cent sulphur and 4.08 per cent copper; coke used, calculated on the ore smelted, 13.93 per cent; coke used, calculated on the mixture smelted, 12.97 per cent; matte produced contained 42 per cent copper, sulphur to copper, 2.42 sulphur to 1 copper. Thus it is seen that, as compared with cold blast practice, the hot blast made a saving of 33 per cent coke on ore smelted and 41 per cent on charge, the tonnage smelted being practically the same. In regard to oxidation, the respective percentages of roasted and raw sulphide ore used in the furnace charges were reversed, the hot blast producing equally good results with a two-thirds proportion of raw ore as the cold blast with a one-third, the product being in both cases a 42 per cent matte. The saving in ore roasting cost therefore represented about 50 per cent.

LEAD.

Lime Roasting of Galena.—Within the last few years the roasting of galena by the old methods is gradually being abandoned in this country and abroad, and other processes, chief of which and most widely used at the present time, is the Huntington-Heberlein process, are taking their place. A paper by Mr. W. R. Ingalls, presented before the London meeting of the American Institute of Mining Engineers, and published in the *Bi-Monthly Bulletin* of the Institute, September, gives a concise review of the various processes practiced, but deals more extensively with the Huntington-Heberlein process and the economies to be effected by it. In the practical operation of the process the ore, limestone and silica are crushed to pass a 4-mesh screen and are then charged into a circular roasting furnace with revolving hearth. The diameter of the latter is 26 feet, and it roasts about 80,000 pounds of charge per 24 hours. The consumption of coal for reducing the sulphur of an ore containing 20 to 22 per cent down to 10 to 11 per cent is about 22.5 per cent of the weight of the charge. Two furnaces can be managed by one man per 8-hour shift. On the basis of the tonnage assumed above the cost of roasting is approximately estimated as follows: Labor, three men at $2.50 = $7.50; coal, 18 tons at $2.00 = $36.00; power, $3.35; repairs, $3.35; a total of $50.20 for 80 tons, or 62.5 cents per ton. From the roaster the ore is charged into the pyritic converters of cast iron, 9 feet in diameter at the top and about 4 feet deep. A perforated grate prevents the charge from falling to the bottom and stopping up the air openings. Two roasting furnaces and six converters are stated to be rated nominally as a 90-ton plant. The rating, however, is considerably in excess of the actual capacity, as most of the ores contain ores. The operation of blowing usually requires from 12 to 16 hours, and at the end the charge is dumped by tipping the pot. The ordinary charge for the standard converter is stated to be about 8 tons of ore, weighing from 100 pounds per cubic foot. The finished charge contains from 2 to 3 per cent of sulphur. Each converter requires 400 cubic feet of air per minute. The cost of converting is approximately estimated as follows: Three foremen at $3.20 = $9.60; 9 men at $2.50 = $22.50; power, 21 hp., $6.30; further supplies, repairs and renewals, $5.00, a total of $43.40 = 60 cents per ton of charge. As the cost of converting is reduced direct as the time required for the operation, the above estimate is based upon unfavorable conditions concerning the time required for working a charge. The total cost of treatment, from the initial state to the delivery of the desulphurized ore to the blast furnace is estimated as follows, per 2,000 pounds of charge: Crushing 1 ton at 10 cents = $0.10; mixing 1 ton at 10 cents = $0.10, roasting 1 ton at 63 cents = $0.63, delivering 1.1 ton to the converter at 12 cents = $0.13; converting 1.1 ton at 60 = $0.66; breaking 0.9 tons at 60 cents = $0.54; a great total of $2.50. The increase in the capacity of the blast furnace is attributed by the author to three things, namely: 1. Delivering to the furnace a charge containing a reduced percentage of fine ore, the speed of the furnace is increased, that is, more tons of ore can be smelted per square foot of hearth area; 2, there is less roasted matte to go into the charge; 3, under some conditions the percentage of lead in the charge can be increased, thus reducing the quantity of gangue that must be fluxed. The last part of the paper is devoted to notes on the operation of the Carmichael-Bradford and the Savelsberg processes.

ZINC.

Composite Retorts for the Manufacture of Zinc. It is a well-known fact that the retorts in the manufacture of zinc are a source of considerable trouble. A good deal has been accomplished of late years by the introduction of presses for manufacturing them, but for refractory ores high in iron and lead, of other ore of easily fusible character, not even the retorts manufactured by hydraulic means, are altogether satisfactory. Retorts may, of course, be made from a more refractory material than is ordinarily used, namely, graphite, chromite, carborundum, bauxite, etc., but they are very costly. In order to do away with part of the expense it has been proposed to line the retorts with materials such as dolomite or magnesite, which is made to adhere to the surface proper of the retort by the interposition of silicate of soda. Another process for the manufacture of so-called composite retorts has been elaborated by A. L. Queneau, and is described by him in the *Engineering and Mining Journal* of Oct. 13, 1906. The requirements for retorts in the zinc manufacture are summarized by him as follows: 1. High refractoriness to well stand temperatures that often reach 1,600° C. 2. Chemical composition adapted to resist the corrosive action of bases such as FeO, MnO, CaO, MgO, etc., and of metallic sulphides and compounds, such as ferrous sulphide, fluorspar, garnet, etc. 3. Good heat conductivity, since the charge is heated only by transmission through the vessel's walls. 4. High tensile and compressive strength, together with the ability to resist the abrasive and impact blows of the charging and leaning tools. 5. Dense walls, to be little penetrated by the [illegible] 6. High elasticity in order to endure the repeated contractions and expansions due to the varying temperatures. 7. Low [illegible] The composite retort constructed by the author is obtained as

one operation by means of the Dor hydraulic press. It has a main body portion of the usual fire-clay and sand mixture, and an outer surface of predetermined and suitable thickness, made of a mixture of fire-clay and of a basic or neutral material, which latter material takes the place of the customary sand or chamotte, either wholly or partly, according to the requirements of a particular case. In the operation of the Dor press, owing to its automatic action, particles of clay placed in a like position in the various wads of material operated upon, cover always the same path while the retort is being made, and always go to form the very same portion of the finished articles. Therefore, if the relative positions of the particles in the wad and the resulting retort are ascertained experimentally, it will become possible to substitute in the proper zone of the wad the clay and sand mixture by a suitable refractory mixture to form the desired protective coating in any portion of the vessels. In starting the manufacture of composite retorts an initial batch of fire-clay, sand and the selected refractory material is prepared, the batch being made sufficiently for the first day's work. The refractory mixture for the following days is obtained by using the press wastage as well as that from the hammering machine. To each barrel-load of this material an appropriate weight of fresh refractory material is added, and the whole is then thoroughly mixed and pugged. The author states that the same number of composite vessels is made as when the press is used for the manufacture of ordinary retorts, the extra cost over the ordinary retorts being only due to the difference of price between the fire-clay and the selected refractory material. This extra cost is stated to vary between 30 cents for a Ceylon graphite mixture and 15 cents for a carborundum-sand mixture, the thickness of the protective layer being usually ⅜ inch. According to the author, in a test lasting several months, made at the Palmerton (Pa.) plant of the New Jersey Zinc Co., it was found that with a protective layer formed of a mixture of Ceylon graphite with fire-clay, an increase of life of the retorts of 33 per cent was obtained in treating Willemite ores of the following composition: 46 per cent zinc, 3 per cent FeO, 8 per cent MnO, 1.8 per cent MgO, 3.5 per cent CaO, 24 per cent SiO_2.

Magnetic Separation at Galena, Ill.—In a communication to the *Engineering and Mining Journal*, Sept. 15, 1906, the author describes the process of separation of low-grade zinc ore adopted at the plant of the Joplin Separating Co., at Galena, Ill. The ore, after being crushed and screened, goes to an ¼-inch impact screen, the undersize of which passes to two Wilfley tables, on each of which Galena, pyrites and blende are separated, and a clean Galena product is made, but only a rough separation of pyrites from the blende can be effected. The oversize from the impact screen goes to a seven-compartment jig having 30 x 36-inch screens of the following sizes, beginning at the lead end: No. 1, 5-mesh; No. 2, 8-mesh; No. 3, 4-mesh; No. 4, 5-mesh; No. 5, 6-mesh; No. 6, 7-mesh. The tailings flow direct into the creek, while a clean lead concentrate is made on the first two cells, and a mixed pyrites-blende concentrates on the other five cells. The concentrates are drained and roasted in a roasting cylinder, 32.5 feet long and 5 feet in diameter, with a capacity of 40 tons in 24 hours. The ore is roasted for 2½ to 3 hours, the roasting being conducted in such a manner as to burn off one atom of the sulphur of the pyrites. The cooled ore then goes to a belt elevator, which delivers it to two trommels, each installed over a bin holding 30 tons. The product from either trommel can go to either bin. One of the trommels is fitted with a ¼-inch perforated screen and also a ¼-inch screen, while the other trommel has only an ⅛-inch screen. The oversize from the screens is crushed and returned. The product from the ¼ and ⅛-inch screens is kept separate, and each size is fed to a Cleveland Knowles apparatus. The middlings product coming from the second magnets goes to 10 x 12-inch recrushing rolls, and then to a screen having 3 mm holes, the oversize going back to the rolls again. The undersize goes to a third magnetic separator, the middlings from which are not treated any further. Cutler-Hammer resistance boxes are used to regulate the amperage of the magnets. The voltage is kept at 90; the current used on the first or roughing magnets is 2 amps., and that on the finishing magnets is 6.5 amps. The average of the concentrates made last year are stated to have been for the coarse ¼ inch size, 60.49 per cent zinc and 2.11 per cent iron; for the fine ⅛-inch size, 57.07 zinc and 2.19 per cent iron. At the relative proportions of one part of the coarse to two parts of the fine material, the average was 58.21 per cent zinc and 2.16 per cent iron.

Gold.

Cyanide Practice with the Moore Filter.—In the *Mining and Scientific Press* of Sept. 8, 1906, Mr. R. Gilman Brown concludes his lengthy article on the cyanide practice with the Moore filter at the Standard Mine, Bodie, Cal., the first part of which was abstracted in the October issue of Electrochemical and Metallurgical Industry. The present part of the paper deals exclusively with the operations in the Moore filter plant. The essential difference between the Moore and the Cassel-Butters process is that Moore transfers the filters with their load of slime from the pulp vats to a wash-water tank, and then to the discharge hopper by means of a crane, whereas, in the Cassel-Butters process the pulp is removed from the filters by pumps of large volume, wash-water is substituted, and then discharged, the operation being conducted in the same vat. In the plant under consideration the filters are made of canvas of medium weight, 5 feet wide and 16 feet long, the filters being so constructed that percolation of the solution is possible. A frame of steel I-beams supports forty-nine of such filters hung 4 inches from center to center. This arrangement constitutes a so-called basket, which is approximately 16 feet square. On top of each basket rests a suction pump, driven by a 10-hp. d. c. motor. The filter space afforded by this arrangement is quite large, inasmuch as each unit comprises 7,840 square feet. The filtering vats are round and flat-bottomed, with a diameter of 24 feet and a depth of 7 feet. They are provided with four-arm agitators, which run close to the bottom at 7 r. p. m. The process is carried out as follows: A basket is lowered into the vat, which is full of pulp until the tops of the slats are submerged. The suction pump is then started, the first solution, which comes through muddy, being returned to the vat. Under ordinary circumstances the solution soon clears, but if there should be a leak in any of the filter plates, this can be quickly detected and the damaged filter cut out. The suction is continued with intermittent agitation until a sufficient thickness of deposit is obtained—in the present case an average of ¾ inch. The basket is then raised, the suction pump being still kept running, transferred by a crane to the wash-water tank, and lowered into the latter. The washing is continued with frequent titration of the filtrate until the solution has reached a predetermined minimum of strength in cyanide. About 0.7 tons of wash-water is stated to be needed per ton of dry slime. The cake in this condition carries about 40 per cent of moisture. The basket is then raised again and run over the discharge hopper, the suction being continued until the excess of moisture is removed. An air pressure of 35 pounds is then turned on in successive blasts of a few seconds each. This operation causes the cake of slime to drop off, the discharge averaging to 85 per cent of completeness. Every alternate day the filters are cleaned by substituting water under 20 pounds pressure for the air. The time consumed in this cycle of operation depends, of course, essentially of the thickness of the deposit, the average thickness of the latter for the last year in the plant under consideration being about 0.74 inch, and the maximum for one month 1.14 inches. The time of the accretion period is about 3½ hours, washing and discharge consuming about the same time. Three complete cycles can thus be made in 24 hours with time for emergency matters. The author claims that this time is considerably longer than has been

experienced elsewhere, and that this fact must be attributed to the large percentage of clay in the materials. On the important question of the wear of the filters the author remarks that an extreme life of ten months has been noted in some cases, while six months can be taken as a fair average life. However, if they are badly made, or carelessly handled, they require constant attention, and half a cycle per day can easily be lost in this way.

Tube Milling in Korea.—An interesting experimental plant for carrying out experiments in tube milling with a cyanide solution, established at the Oriental Con. Mining Co., at Chittabalbie, Korea, is described in the communication by Mr. A E. Drucker in the *Mining and Scientific Press,* Sept. 22, 1906. The plant consists of one tube mill, 2½ x 12½ feet, two mechanical agitators with plow shoes, 8 x 6 feet in diameter, three filter and settling boxes, 4 x 5 x 5 feet, forty-eight separate-compartment zinc boxes, two sumps and one stock solution vat. The method is continuous, the concentrate being put into the hopper with cyanide solution, nearly all being ground to slime in passing through the tube mill. A good percentage of the gold is claimed to be extracted. At the end of the mill is located a spitzkasten 3 inches wide by 1½ foot long, by 1½ foot deep, which is supplied with clear water from the bottom. The bottom discharge of the spitzkasten consists of coarse concentrates and clear water. What passes over the spitzkasten is the cyanide solution and slime. The coarse concentrate which may escape from the tube mill is caught in the settling boxes and again reground, while the overflow contains very little value, as it assays 6 cents per ton. The slime concentrates and the cyanide solution pass on to an agitator, which has been set in motion at 3 r. p. m., and the muller lowered. The agitation of the product is continued for about 15 hours, the muller being then raised 2 feet, the agitator stopped, and the concentrate allowed to settle. The clear cyanide solution is decanted into filter boxes, and from there passes through the zinc boxes. The strong precipitated cyanide solution from the sump is then pumped up to the agitator as a wash to partly remove the gold remaining within the settled concentrates. The muller is then allowed to work down on the charge and agitated for a few minutes, this being followed by settling and decantation. This operation is followed by two successive weak cyanide washes to remove the cyanide and gold, and finally a water wash is given which is run through a row of zinc boxes to waste. The charge of about 5 tons of clean concentrates is then discharged. The total treatment is said to consume 24 hours, both agitation and decantation being done in the agitators. While one agitator is being filled the other is decanting and getting ready to be discharged. Strong cyanide solution of 0.43 per cent is used in the tube mill and agitators; the weak solution which is employed for the washes has a strength of 0.1 per cent. No cyanide is added to the weak solution. About 2 pounds lime per ton is added with the concentrate as it is fed into the mill. The author states that the extraction obtained at present with the tube mill and agitators on clean concentrate is 93 per cent, and hopes to better these results by gradually making improvements.

Ore Treatment at the Smuggler Union Mill, Telluride, Col.—A description of the milling practice at the above plant is given in the *Mining Reporter,* Sept. 29, 1906. The Smuggler mill contains sixty 1,050-pound stamps, dropping 6 inches at a rate of from 100 to 108 drops per minute, crushing through 14-mesh No. 20 steel wire screens of 18 x 52-inch size. The duty of each stamp is given as nearly 5 tons per 24 hours. The stamps are fed by simplified adjustable challenge feeders. The amalgamating plates are 8 x 4 feet. From the plates the pulp passes to stationary No. 24 brass wire screens of 20-mesh, the undersize dropping on bumping tables, and the oversize going to 5-foot Huntington mills, which are fitted with 20-mesh wire screens. The thirty-two 5-foot Triumph vanners set on a floor below are fed with battery pulp without classification. The vanner tailings are taken by a wheel-bucket elevator to the top of the mill building, and are delivered by a launder to the cyanide plant. This wheel bucket elevator has an [illegible] diameter of 28 feet with a rim of 12 inches, and is constructed entirely of iron. It is fitted with overlapping pockets or buckets, 5 x 12 x 18 inches. As the wheel is rotated in the pit below, which contains the vanner tailings, the buckets are filled and, on assuming a horizontal position at the top, dumped directly into wooden receiving boxes, which empty into the cyanide plant delivery launders. It is stated that the operation of the wheel requires about 5 hp., and that during a continuous operation of twenty months, the repair expenses have been less than $2.00. The tailings material from the concentrating plant goes to large settling tanks in the cyanide plant, the overflowing slimes being pumped back to the can[illegible] while the slimes are treated by percolation. The canvas plant consists of ninety-eight tables, 10 x 12 feet, the overflow being retabled over four Wilfley slime tables, and the heavy material as washed down being still further concentrated over vanners and Wilfley tables. The canvas slime plant produces from 10 to 12 tons monthly of high grade concentrates that carry about 17 per cent moisture. None of the concentrates are dried before shipment. The cyanide plant is equipped with sixteen wooden tanks of 40 feet diameter, 8 feet depth. The solution tanks are 12 feet deep and 40 feet in diameter. The strength of the cyanide solution used is variable, on account of the diversity of the leaner ores treated, the average, however, being given as 8 pounds cyanide to the ton of ore for the strong solution and 2.5 to 3.5 pounds for the weak solution. The diverse character of the ores also causes a variance of the leaching period of from twelve to sixteen days.

Ore Treatment at the Combination Mine, Goldfield, Nev.—A lengthy description of the method adopted for treating the ore of the Combination Mines Co., as well as customs ores from the district, by Mr. F. R. Bosqui, in the *Mining and Scientific Press,* Oct. 13, contains a number of interesting data. The mill is equipped for treating oxidized and sulphide ore. The ore is crushed by a 10 x 16 Sturtevant roll-jaw crusher, after having been mixed with the required amount of lime, varying from 5 to 10 pounds per ton. The crusher breaks the ore to about ½-inch size. The ore is then elevated to the mill bins, there being four of these bins for the four units of five stamps each. Ten more stamps have recently been added for treating sulphide ore. The following account refers to a complete five-stamp system. Inside amalgamation is carried on by means of a curved plate screwed to the chuck block, the ore being crushed through a 12-mesh wire screen. A spash-plate is used outside the screen, from which plate the pulp falls to a lip-plate about 12 inches wide. The front edge of this plate is slightly bent down, giving the pulp a gentle drop to the apron-plate. There are three plates to each mortar, arranged in steps, giving an amalgamating surface 53 inches wide and about 12½ feet long. The whole tray, by means of wheels and track, can be shifted during the clean-up of the battery. At the bottom of each tray there is a small cone hydraulic classifier, which separates the coarse mill pulp into two products, namely: 1, fine sand and slime, which passes to the outer discharge lip of the Bryan mill, and thence direct to the concentrators; 2, coarse sand, which passes to the Bryan mill for regrinding. The ore is extremely hard and tough, and is therefore crushed with 1,350-pound stamps, falling 100 times per minute with a 6-inch drop. The stamp duty is only 3½ tons, using a 12-mesh screen. One of the 5-foot Bryan mills, running at full speed, takes all the coarse sand from ten stamps and crushes it through a No. 9 slotted screen, equivalent to 40-mesh. The final product of the Bryan mill is passed over two 6-foot vanners and two 6-foot Triumph tables. The pulp is then raised by two Frenier sand pumps to two sets of cone classifiers. The top cone is not a classifier, but is so adjusted that it sends a fairly uniform flow of pulp to the two smaller cones. The first rough classification is made in the small hydraulic cones, from which a stream of sand flows

direct to the sand vats. The overflow from these small cones, consisting of slime and fine sand, flows to the larger lower cone, where a closer classification is made, the stream from the bottom of this cone also flowing to the leaching vats. The overflow from the large lower cone passes to the slime settlers. The separation was not ideal though the recovery from the slime (over 95 per cent) has been very good, but a contemplated re-arrangement of the three cones is expected to improve the extraction from slime. From the cones the sand flows to four settling vats on each side of the mill. When the sand-settler is filled the surplus moisture is removed by a Gould vacuum pump, and the charge shoveled to the treatment vat below, where it is given an eight days' treatment with a 0.1 per cent and a 0.2 per cent cyanide solution. The slime is delivered to the center of a conical-bottom settler, provided with a rim overflow, two settlers being arranged at each side of the mill. Each settler is alternately allowed to fill and overflow for 12 hours, and allowed 12 hours for settling. The surplus water is carried off by a pipe decanter, the slime being left with about 50 per cent of moisture. Sufficient strong cyanide solution is added to the charge of slime to make a solution of from 0.15 to 0.2 per cent cyanide, and to give the pulp a consistency of three parts solution to one of slime. By means of a centrifugal pump the pulp is then transferred to an agitator with a steep cone bottom, and the same pump then agitates the slime by taking it from the bottom of the vat and throwing it back at the top, a supplementary agitation being given by means of a mechanical stirrer, revolving slowly. The agitation lasts from 12 to 15 hours, and the pulp is then discharged into the slime reservoir, which is a vat of large capacity, provided with a mechanical stirrer. It is thence drawn as required for filtration in the Butters-Cassel filters. In the plant under consideration there are twenty-eight frames, 5 x 10 feet, set 4½ inches apart in a box 10 feet square, with a pointed bottom inclined at an angle of 50°. The slime pulp is introduced at the point of the box, vacuum of 22 inches of mercury being maintained for about 20 minutes, during which time a cake, ¾ to 1 inch thick, is deposited on each side of the filter frame. The surplus pulp is then withdrawn to the slime reservoir, and a wash, consisting of a weak solution of cyanide, introduced. When the cakes are thoroughly washed the weak solution is withdrawn into its respective vat, and water is introduced until the frames are completely immersed. The object of this final water is to assist in removing the cakes. More water is introduced into the interior of the frames under a low head, and this water causes the cakes to drop off clean into the pointed bottom of the filter box. From there they are finally removed by sluicing. The whole requires about 3½ hours, and about 9 tons of dry slime are treated at each charge; the plant has, therefore, a capacity of 63 tons. One man on each shift is required for the operation. The author states that the filter plant has required no repairs since it was first operated in February of this year. The cost of filtering exclusive of power has been reduced from $0.96 per ton of slime in January, when filter pressing was used, to $0.26 in May, when the above described system was in operation. In the treatment of the sulphide ore a 4 x 12 Abbé tube mill of the trunnion type is used.

RECENT METALLURGICAL PATENTS.

Copper.

Converter.—A novel converter construction for use with copper matte has been patented by Herbert Haas (832,665, Oct. 9), the object being to secure a uniform consumption of the lining in forming the slag, and a reduction in the labor and power required. The converter is shown in Fig. 1. It is built in two parts, an upper part 1 and a lower part 2, which are secured together by means of the iron bands 5 and 6, which have flanges 3 and 4 secured together by bolts 7. The compressed air is introduced through the hollow shaft 14, and then passes through the transverse air conduit 17, which is joined to the short pipe 18 conducting to the wind-box. The converter is rotated by means of the pinion 22. The wind-box is built in two parts, the casing 30 having an annular concaved flange 31, adapted to be connected with the bottom of the converter, and a flange 32 for supporting the bottom of the wind-box, and, secondly, the bottom 33 of the wind-box, which is a plate for supporting at suitable angles the tuyere direction pipes 34. The object of these pipes is to give a suitable direction to the apertures which are formed in the converter lining over the wind box, in order to produce the best result. The construction of the wind-box with direction pipes arranged tangentially in a ring about a central axis is a special feature of the converter, and the effect of this construction is that the blast imparts to the molten matte a rotary motion. The reduced copper cannot collect in the bottom of the converter so long as the blast is blowing, for the fluid mass is forced by the tangentially rising air currents to travel around the entire converter lining, thereby evenly corroding the lining. The direction of the tuyeres is such that the issuing air develops a centrifugal action upon the matte, and the reduced copper being of greater specific gravity than the matte is thrown by the centrifugal force toward the converter walls as far as possible from the orifice of the tuyeres or conduits, and thus avoids the chilling action of the cold blast, but at the same time centrifugal force also acts upon the iron, which is being constantly reduced from the sulphide, and which having a greater specific gravity is thrown against the converter walls, and being nascently oxidized as the sulphur is burned off combines with the silica in the lining to form a slag.

Lining Converters.—When the basic or silicious lining of a converter has been stamped in, it is necessary to dry the lining carefully. An improved and shortened process of drying has been patented by H. L. Charles (832,895, Oct. 9), of Butte, Mont. The tuyeres are first closed and molten slag, low grade molten copper or molten iron matte from converters or reverberatory furnaces is filled in. Converter slag is preferred, as it has a higher temperature than all other known slags. A few pieces of wood or coal are then put on top of the slag or matte, and a light fire kindled to keep it from chilling. In about an hour a shell of slag or matte forms all around inside of the lining. The first slag or matte is then poured off, and a fresh, hot layer of slag or matte is poured into the converter. This can remain indefinitely, as the heat from the first slag or matte penetrates the lining so far that the remaining moisture will be totally driven out in 3 hours, and the lining remains hot for 2 hours. A complete and uniform hardening of the lining is thus secured in a short time. In addition the slag shell formed on the inside is easier smelted than the cold, hard silica lining, making the converter run faster and raise heat quicker than is possible in the old way. The resulting slag from first corrosion will run higher in iron than with the old process.

Smelting Copper Matte.—W. Kemp (832,738, Oct. 9) patents the following process for smelting copper matte so as to produce black or metallic copper directly therefrom in a single continuous operation. The copper matte is charged into the furnace in a hard state, mixed with silica flux, and a burning flame of oil vapor, steam and air is continuously forced from burners into and through the mass of the matte at the bottom, the power of the flame being very great (approximating a temperature of 1,200° C.) due to the large excess of air in it

The action of the flame is to successively reduce the lowermost zones of the matte to a molten condition, the mass of the matte above being at the same time subjected to a gradual roasting action upwardly. As each lowermost zone is reduced to the proper state of fusion, the copper and slag are drawn off and the next succeeding and highly roasted zone takes its place. The chemical action is said to be that the copper-iron

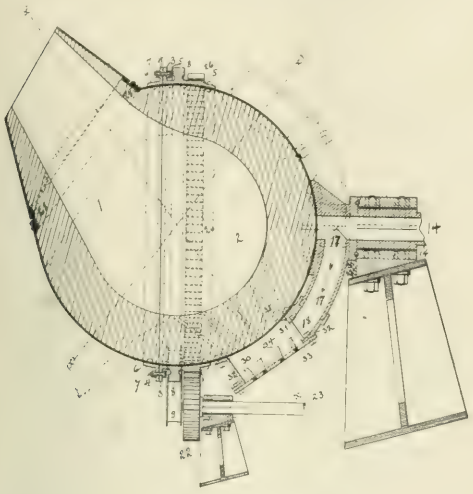

Fig. 1.—Converter.

sulphide reacts with the excess of oxygen and forms metallic copper, ferrous oxide and sulphurous acid gas. The ferrous oxide then combines with the silica to form slag. The copper remains in molten metallic condition.

Welding Copper.—J. Spittall (832,755, Oct. 9) welds pieces of copper and other metals by first heating them to a cherry red. The parts of the metal which are to be made contiguous in the weld are then covered lightly with a composition consisting of 2 ounces of boracic acid and 1 ounce of phosphate of soda, thoroughly mingled by stirring. The two pieces to be welded together are then placed together, the composition not necessarily lying between them, and the welding is completed by hammering or pressure, as is done in the ordinary welding of iron.

Nickel.

Orford Process.—We have repeatedly referred in these columns to the method of tops and bottom smelting of nickel-copper ores, used so extensively by the Orford Copper Co. The present practice is to smelt the matte containing sulphides of nickel, copper and iron in a cupola furnace, together with a charge of coke and sulphate of sodium, in the form of crude niter cake. The carbon of the coke reduces the sulphate of sodium to sulphide of sodium, and as this material when fused has a solvent action upon the copper and iron sulphides contained in the matte it will dissolve them to a considerable extent, so that when the molten contents of the furnace are tapped into settling pots the materials will stratify. Sulphide of nickel, together with some sulphide of sodium, will settle to the bottom, and the sulphide of sodium, with the copper and iron sulphides in solution, will float on the top. The mass when solidified is divided into tops and bottoms, the tops containing sulphides of copper, iron and sodium, and the bottoms containing sulphide of nickel, together with such of the sulphides of copper and iron as have not been taken up in solution in the sulphide of sodium. The bottoms are again charged into the smelting furnace with sulphate of sodium, and are resmelted a sufficient number of times to purify the material to such an extent as to make the nickel matte [illegible] copper, generally reducing the copper to less than 1 per cent. After having thus been treated the matte is [illegible] leached for the purpose of removing the soluble portions of the sodium compounds, and the product is dried and partially calcined, and is then mixed with sodium chloride and sodium sulphate and again calcined. The sodium chloride and sodium sulphate react to produce chlorine gas, which converts the copper contained in the mass partially to the form of copper chloride, and the remainder of the copper to copper sulphate, the nickel being converted largely to the form of nickel oxide and partially to the form of nickel sulphate. The calcined product is then leached with water for the purpose of removing the soluble salts of the three metals—namely, the copper chloride and copper sulphate, sodium sulphate and nickel sulphate. The residuum is then rewashed with dilute sulphuric acid, which removes the copper as a sulphate, but does not attack the nickel oxide. It has been the practice to precipitate the metals from these solutions by the addition of sodium carbonate or sodium sulphide and then to remove the precipitate by passing the liquor through filter presses, in which operation the sodium compound is lost.

R. R. Maffett, of the Orford Copper Co. (833,722, Oct. 16), has invented a modification of the last part of the process as described above. He avoids the precipitation and filtering. The liquor which results from the washing of the nickel-copper compounds with water and sulphuric acid, and which contains in solution sulphate of sodium and sulphates of copper and nickel, is withdrawn and introduced into an evaporating tank containing steam coils. There it is heated and concentrated to the point of crystallization, the sodium-copper-nickel-sulphate being recovered in the form of crystals. These crystals are taken from the vat, and can be introduced directly into the smelting furnace as a part of the charge of matte, coke and sodium sulphate in the same manner in which niter cake has heretofore been introduced, and it has the advantage not only of supplying the sulphate of sodium requisite for the smelting operation, but also of saving the contained copper and nickel, which pass with the molten contents of the furnace into the settling pots and are there recovered.

Zinc.

Distillation Process with Internal Heating. The [illegible] low thermal efficiency of the zinc distillation process with a large number of retorts heated from the outside, is well known and well understood. It is just for this reason that attempts have been made to introduce the electric furnace in the zinc industry by using internal heating by means of the current and by using large-size furnaces. A process patented by H. Mehner (833,472, Oct. 16) also endeavors to use internal heating in larger furnaces, but without employing electric heating. His process consists essentially in bringing in contact with each other the zinc ore, the reducing agent, and an "incandescent fluid heat carrier of such a nature as not to absorb or otherwise do harm to the metallic vapors generated." The heat carrier may consist of a molten silicate (including slag, scoria, cinders and the like) heated above its melting point up to incandescence. The operation is indicated in Fig. 2: *h* is the furnace for heating the silicate, which is a shaft furnace filled with coal, coke and the like. The zinc ore is added to the molten silicate, and is supplied to the top of the shaft furnace through the centrally located tube *i*. The mixture of zinc vapors and carbon monoxide resulting from the reduction escapes from *n* into a condenser. The zinc vapors are obtained in as dense a state as in the usual distillation process, being only attenuated by such a quantity of [illegible] oxide corresponds to $ZnO + C = CO + Zn$. On the other hand the great losses of zinc, which in the usual distillation process are caused by the permeable condition of the walls of the retorts, may be avoided by lining the furnace with bricks of dense carbon, which do not absorb the vapors to allow the vapors to penetrate them. The silicate escaping from the lower

part of the furnace through *f* is collected, heated anew to a bright cherry-red, at least, mixed with a fresh quantity of zinc ore and then used over again. In case the ore contains beside zinc other metals—such as lead, silver, copper, etc.—or is a so-called composite ore, then those metals will collect on the bottom of the furnace below the silicate, and the mixture may be tapped off through a tap hole at suitable intervals. The molten silicate may be replaced by a molten metal heated above its melting point up to incandescence. The author prefers to use iron, since it is by itself a reducing agent for zinc ores, especially zinc blende. In case of sulphur-containing zinc ores, the iron, by its contact with the ore, drives out the zinc in the state of vapor and combines with the sulphur in the ore. The molten iron after being removed from the lower portion of the furnace is reheated by blowing a current of air through, whereby the sulphur in the iron is burned to sulphur dioxide. By the same process pure zinc sulphide may be produced instead of metallic zinc by producing sulphur vapor within the furnace, which combines with the zinc vapor and forms zinc sulphide, which is collected as an impalpable powder. To obtain pure zinc white, in a finely flocculent condition, oxygen or air, etc., is admitted to the escaping zinc vapors so as to burn them.

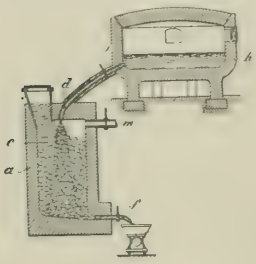

Fig. 2.—Zinc Distillation.

Gold and Silver.

Treatment of Refractory Ores.—J. W. H. James (833,394, Oct. 16) patents the following treatment of refractory gold and silver ores, the object being to eliminate from the ore the contained relatively baser metals. In a cylindrical retort, heated from the outside by means of producer gas, the ore is roasted in a loose state without access of air, and simultaneously subjected to the action of water gas. The result of this treatment is that the ore becomes exceedingly friable so that it may be easily broken by hand and passed between crushing rolls. The gold particles are said to assume thereby a globular form, so that the loss in the form of float-gold is reduced to a minimum. The sulphur, arsenic, tellurium, antimony, bismuth, zinc, lead, iron, copper pyrites, iron oxides, and the other baser metals or materials with which the gold may be chemically or mechanically combined, are simultaneously eliminated from the ore. They are either carried off by the water gas as volatile compounds, or they are reduced to such a condition that the gold can be obtained by amalgamation or otherwise. The absorbed volatile compounds may then be readily separated by precipitation in suitable wet and dry chambers. The water gas is preferably employed in the condition of about equal volumes of associated but uncombined hydrogen and carbon monoxide.

Precipitation from Cyanide Solutions.—In ordinary cyanide practice the solutions are almost invariably alkaline in reaction. W. J. Sharwood (832,880, Oct. 9) first makes the solution neutral by adding either commercial sulphuric acid or acid sodium sulphate. If finely divided zinc is then used for precipitation, it is claimed that the amount of zinc dissolved during precipitation is greatly reduced, and that the precipitation of gold, silver and copper is more perfect.

Ozone in Cyanide Process.—In cyanidation, as practiced at present, the oxidation is generally effected by aerating in large, flat tanks, the pulp being allowed to percolate. This requires constant agitation for quite a long period, so that the pulp will be exposed over and over again to the action of the air. E. J. Crane (827,630, July 31) recommends the use of ozone. He forces the solution of pulp, with cyanide added, to the upper portion of a tank, from whence it passes through an ozone-containing chamber, the solution being constantly taken off from and returned to the tank, and in its passage entrains an additional supply of ozone, which by the action of the circulating pump is thoroughly beaten into the pulp. In treating refractory ores previously ground to pass through, say, one-eighth or one-quarter-mesh, pulverization may be effected by the impact of the discharge of the pump being directed against an arrester located above the ozone-containing chamber. Nothing is said concerning the cost.

Iron.

Treatment of Fine Ores.—A. J. Mason (833,406, Oct. 16) patents the treatment of fine iron ores in a furnace, illustrated in Fig. 3. The fuel is pulverized and introduced into the reducing and melting chamber 2 from the hopper 1 with the aid of a strong air blast. The finely-divided ore and flux are supplied through the hoppers 7 and 8, respectively, and drop in a finely-intermingled stream downwards through the stack 9. In the melting chamber 2 reduction takes place to metallic iron which flows in a molten stream through the discharge spout 15 into a pot underneath; 17 is a bed of coke, which constitutes a lining for a portion of the bottom and side of the melting chamber 2, against which the combustible gases and fuel are directed to complete the reduction and melting of the ore by furnishing an incandescent bed on and through which the partially reduced and melted ore drops down. This bed of incandescent coke will remain unconsumed as long as the pulverized fuel itself furnishes all the carbon required for reduction. The hot gases are used for regenerative heating. The essential differences between this process and blast furnace practice are, first, that all materials are introduced in finely-divided condition and in shallow broad streams, and, second, that the carbon for reduction is not introduced together with the charge at the top but is supplied directly to the reducing zone.

Briquetting.—H. Schulte-Steinberg (833,005, Oct. 9) uses for briquetting ores a binding material made from blast furnace slag. The comminuted slag, preferably white furnace slag, is exposed in a revolving cylinder to the quick and thorough action of high-pressure steam. The binder thus obtained is intimately mixed with the refuse or rotten ore in

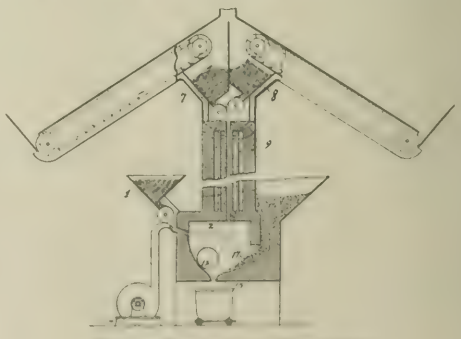

Fig. 3.—Treatment of Fine Ores.

the proportion of 1 to 10, and then the mixture is applied to any kind of briquetting machine.

Iron from Copper Slags.—R. Baggaley (832,948, Oct. 9) endeavors to save the iron contained in the ferruginous slag of copper-smelting furnaces, and points to the fact that pig iron and steel are generally expensive and scarce wherever copper smelters are operated. The ferruginous slag discharged from copper-smelting furnaces is composed of silica combined with iron and other bases. This fact, of course, demands the use of fluxing agents. The silica is separated from the con-

tained iron by means of limestone as flux. The limestone combines with the silica to form a slag, and thereafter in the operation of bessemerizing the various impurities are eliminated in succession in the order of their oxidation. The last vestiges of sulphur in their elimination will demand the sacrifice of a considerable percentage of the iron, which is thought to be permissible in this process. In some cases also it will demand the almost complete elimination of the carbon, and in such cases the carbon must be restored to the bath by means of adding spiegeleisen or ferro-manganese, etc.

Purifying and Reheating Blast Furnace Gases.—D. Lamond and D. D. Lamond (833,467, Oct. 16) have patented the apparatus shown in Fig. 4 ,for utilizing the heat given up from the impure blast furnace gas, for reheating it after it is purified and before it passes to the furnaces, stoves, etc., for use as fuel. There are three large stove-like settling chambers, 2, 3, 4, connected in series by horizontal pipes 5, and having the usual bell and hopper-bottom construction 6 for the accumulation and discharge of impurities which separate by gravity from the gas. Each chamber is enclosed by an outer wall 7, thus forming a flue 8. The upper ends of the flues 8 surrounding chambers 3 and 4, are connected by elbow 10, while the lower ends of the flues surrounding chambers 2 and 3 are connected by duct 11. The upper portion of chamber 2 is connected by pipe 12 with the dust catcher 13 on the blast furnace. Chamber 4 is connected by duct 14 with the lower portion of the gas washer 15, the upper end of which is connected by the vertical pipe 16 with the lower end of flue 8 surrounding chamber 4. The blast furnace gas passes directly from the dust catcher 13 into the upper end of chamber 2, and then successively through pipes 5 into chambers 3 and 4. In passing through these relatively wide chambers the gas expands, and the heat is absorbed by the chamber walls, thereby heating the outside flues 8 to the temperature of the gas as it leaves the blast furnace. The passage of the gas through these chambers is comparatively slow, so that it can give off its heat. The gas then passes into washer 15, and in its upward passage it encounters downward sprays of water discharged from transverse perforated pipe 17, whereby all the dust and impurities are removed. The pure and cool gas passes now from the washer downward through pipe 16, and into the lower end of flue 8 surrounding chamber 4. It passes

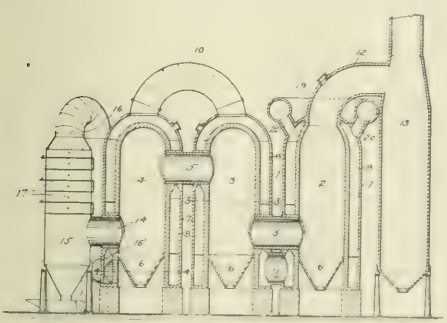

Fig. 4.—Treatment of Blast Furnace Gases.

then upwards through 8 and through elbow 10 into the corresponding space surrounding chamber 3, and then through duct 11 into the space surrounding chamber 2. The pure gas is thus reheated, and its temperature when it enters the bustle pipe 19 is about the same as that of the impure gas coming from the dust catcher. The bustle pipe 19 has a series of connections 20.

Briquetting Friable Ores.—For the briquetting of natural friable ores and ore waste or dross, C. Reinke uses as binding agent so-called "Permian" limestone or other limestone containing a high percentage of [illegible] a small quantity of magnesia and good Portland cement. About four parts of limestone are mixed with one of cement. The ore is intimately mixed with the mixture serving as binding agent, and the mass is then formed under great pressure into briquets. A special advantage is the low percentage of water in such a prepared briquet, analysis showing the following composition: Iron, 56.78 per cent; residue, 3.60 per cent; lime, 7.01 per cent.; [illegible] per cent. The briquets thus produced stand red heat perfectly, and remain solid until removed from the furnace.

Self-Hardening Tool Steel.—J. Churchward (832,773, Oct. 9) patented [illegible] of the following composition: 84 to 90 per cent steel, containing about 0.2 to 0.6 per cent of carbon; 4 to 6 per cent nickel; 0.5 to 1.5 per cent tungsten; 2.5 to 3 per cent chromium; 0.5 to 1 per cent manganese, and 0.25 to 0.5 per cent vanadium.

Mercury

Distillation Process.—In the ordinary method of heating ores by a current of fuel gas, the latter (*i. e.*, the total calories supplied to the furnace) is introduced at one end of the ore receptacle and passes through the ore in the direction opposite to the movement of the ore itself. The ore on its travel through the furnace is thereby gradually heated, while the temperature of the fuel gas current is gradually reduced. A radical departure from this practice is proposed in two patents of W. B. Dennis (833,679, process patent, and 833,680, apparatus patent, both of Oct. 16). In the Dennis process the furnace gases of the heat current flow through the ore-containing receptacle from the cooler portions to the hotter portions in the same direction as the ore is moved itself. Since the heat supplied from the gas to the ore in a given time depends on the difference of the temperatures of both substances, and since the daily capacity of the furnace (which is one of the chief factors in the cost of ore treatment) depends on the time required by supplying the heat to the ore, it would, of course, be impossible to supply the whole fuel gas at once at the cold end of the furnace, and let it pass together with the ore in the same direction through the furnace. For, in this case the thermal efficiency would be high only in the beginning, on account of the high temperature difference between the fuel gas and the ore, but the further the progress in the furnace the lower would be the temperature difference, and the less heat would be supplied from the gas to the ore in the unit of time. The principal feature of Dennis' furnace construction consists in dividing the whole furnace into a number of zones, and in adjusting and regulating the temperature slope between the ore and heat current in each zone according to the special conditions. The fuel gas made in the gas producer is not supplied in bulk at once at the cold end of the furnace, but is gradually supplied to each zone, together with enough air, according to the conditions prevailing in that zone.

The general scheme is indicated on Fig. 5. The different heating zones, eight in the present case, are 14, 15, 16, 17, 18, 19, 20, 21; the primary combustion chambers for the fuel gas are 32, 33, 34, 35 on one side and 40, 41, 42, 43 on the other side, the secondary combustion chambers are 44, 45, 46, 47 on one side and 36, 37, 38 on the other side, 39 is the final superheating chamber. The fuel gas, mixed with air, enters the primary combustion chamber 32, the amount of gas and air admitted thereto being regulated by valves. The mixture of gas and air passes across the furnace through the zone 14 to the secondary combustion chamber 44 on the other side of the furnace, and descends through the checkered flue roof into the primary combustion chamber 40, into which an additional supply of fuel gas and air is admitted, which additional supply, mixed with the gases coming through the zone 14, travels through the zone 15, the result being that the temperature in the zone 15 is higher than it is in the zone 14, thus carrying out the primary principle of the invention, which is

that the fuel gas or heat current should pass from the cooler to the hotter portions of the furnace. This operation continues, the air and gas current flowing back and forth in opposite directions through the respective heating zones and being reinforced in each of the primary combustion chambers by an additional supply of fuel gas and air until it reaches the final heating zone 21, in which the temperature is the highest, and in cases of ore which is to be sublimed preferably above the boiling point of the metals contained in the ore to be recovered. Thence the current of air and gas passes into the superheating chamber 30, which incidentally is a dust-collecting chamber, and is connected with the dust collector proper. To insure the superheating of the final combustion chamber 30, provision is made for an additional supply of air and fuel gas thereto. This current of mixed air and gas is passed through the first cooling zone 22, the temperature in which is much less than in chamber 21.

The succession of the temperatures in the different zones is indicated by the following example: Supposing that the temperature of the gas current in the zone 14 is 200, and the temperature of the ore in said zone is 100, the temperature of the gas is twice that of the ore, which causes a very rapid absorption of heat by the ore. Similarly in the zone 15 the temperature of the gas is 600 and of the ore therein 300; in the zone 16 the temperatures are 1,000 and 500 respectively; in zone 17 the temperatures are 1,400 and 700 respectively; in zone 18 they are 1,800 and 900 respectively; in zone 19 they are 2,200 and 1,100 respectively; in zone 20 they are 2,600 and 1,300 respectively, and in zone 21, the final heating zone, they are 3,000 and 1,500 respectively, while in the first of the cooling zones 22 the temperature is 1,700, all these numbers being in degrees on the Fahrenheit scale. These figures are given to illustrate the regulation of the temperature slope at a uniform ratio throughout the series, and are based on the assumption that it is desired to secure an ultimate induced temperature of, say, 1,700° in the ore before discharging from the last roasting zone, and that the temperature of the ore when delivered from the lowest drying chamber 13 into the first roasting chamber 14 is 100°, or that 1,600° of heat must be inducted into the ore.

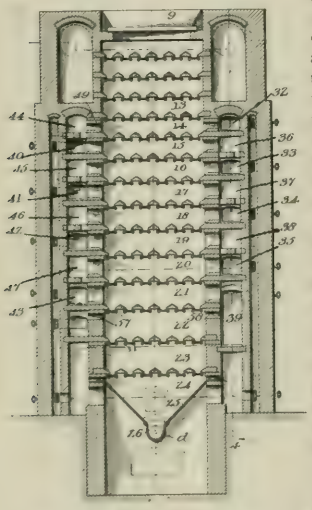

FIG. 5.—MERCURY FURNACE

The operation of the furnace is as follows. Assuming each zone of the ore tower to contain a charge of ore, and the ore in the last treating zone 21 to be fully treated, the grate bars of the cooling zone 23 are opened, dumping the ore into the spent ore pit 24. These grate bars are then immediately closed, and the bars of zone 22 opened and the ore dumped into the now vacant zone 23. This operation is repeated from zone to zone until the hopper 9 has been emptied and is ready for a fresh charge. Each series of bars is operated by a single lever and these levers being located on the charging floor the entire dumping operation requires but a few minutes. The spent ore being discharged by a spiral conveyor operated by power, the furnace operator is not required to leave the charging floor during the entire dumping operation. In treating mercury sulphide ores the best results have been obtained by the inventor in practice by using a charge covering the ore floor from 6 to 9 inches deep, and a perfect roast has been secured in 4 hours. In a furnace with eight roasting floors or zones, as shown in the drawing, one charge would be dumped every 30 minutes. The dumping serves to thoroughly mix the ore better than could be done by mechanical appliances.

A distinctive feature of the process is the method of generating heat from fuel, the feature of which is that the fuel is oxidized gradually or by degrees in progressive zones of combustion by admitting to each zone a volume of air measured in proportion to the temperature of the fuel in such zone, and of causing the gases and vapors of each zone to pass into and through the next succeeding zone in the line of progression of the combustion process, the travel of the gases or products of combustion being always from a zone of lesser to one of more combustion, or from a cooler to a hotter zone. In the zone of incipient oxidation where the temperature is comparatively low, only water-vapor is liberated, and only sufficient air is supplied to produce this result. In the next zone of the series the temperature being somewhat higher than the preceding one, the lighter hydrocarbons are released, and sufficient additional air only is supplied to support this degree of combustion. In like manner from zone to zone the process of combustion is advanced gradually until complete oxidation has finally been accomplished in the last combustion zone. The gaseous products are then delivered into the reducing chamber, containing incandescent carbon through which the gases pass. No oxygen or air is admitted to this zone other than that already contained in the fuel gases. The well-known reaction which takes place here is the reduction of carbon dioxide and water to carbonic oxide and hydrogen.

The process is primarily intended for treating mercury ores but is stated to be of much more general application, "as it may be used in connection with heating furnaces, or furnaces for desulphurizing copper matte, treating cement materials, or even in driers."

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

Manufacture of Silicides, Ferrosilicon, Etc.—F. J. Tone, 833,427, Oct. 16. Application filed May 18, 1905.

The object is the production of various silicides, such as silicides of iron, copper, manganese, aluminium, calcium, or double silicides of these metals—as well as silicon alloys. The process may be described with respect to the manufacture of ferrosilicon. It has been known that considerable quantities of carborundum have been used in the iron and steel industries instead of ferrosilicon in order to introduce silicon into the steel, but a disadvantage of using carborundum is that at a temperature ordinarily obtained in the manufacture of steel, only limited quantities of silicon will be absorbed. It is therefore preferable not to use carborundum directly, but to use it as starting point for making ferrosilicon. For this purpose the inventor has devised a process in which the high temperature of the electric furnace is used. A typical reaction is the following, in which the charge consists of 30 parts iron and 40 parts carborundum, and there is formed a ferrosilicon containing 33 per cent silicon

$$Fe + SiC = (Fe + Si) + C$$

The carbon liberated in this reaction is preferably used as reducing agent for the simultaneous production of further amounts of silicon or iron. This is accomplished by adding to the charge corresponding amounts of the oxide of silicon or the oxide of iron. The following equation is typical of the process when iron oxide is added to unite with the carbon from the carborundum:

$$Fe + 3SiC + Fe_2O_3 = (3Fe + 3Si) + 3CO$$

The following equation is typical of the process when silica is added for the purpose of uniting with the carbon liberated by the carborundum, thus producing a further quantity of silicon, which forms part of the alloy:

$$Fe + 2SiC + SiO_2 = (Fe + 3Si) + 2CO$$

When silica is added to a mixture of the oxide and carborundum, the following equation is typical of the process:

$$5SiC + SiO_2 + Fe_2O_3 = (2Fe + 6Si) + 5CO$$

If it is preferred to use with the carborundum only the oxide of the metal the silicide of which it is desired to produce, a mixture of the oxide and carborundum can be made such that the carbon liberated will be of the exact amount necessary to reduce the oxide, and a typical reaction may be expressed as follows:

$$Fe_2O_3 + 3SiC = (2F + 3Si) + 3CO$$

The process may be applied also to the production of double silicides or to silicon alloys containing several metals—as, for example, an alloy containing silicon, manganese, and aluminium. For example, by using a mixture containing 200 parts carborundum, 87 parts manganese dioxide, and 102 parts alumina, there is obtained an alloy consisting of 140 parts silicon, 55 parts manganese, and 54 parts aluminium. This reaction may be expressed as follows:

$$5SiC + MnO_2 + Al_2O_3 = (5Si + Mn + 2Al) + 5CO$$

Electrolysis of Fused Electrolytes in Induction Furnace.—Leonard Waldo, 833,357, Oct. 16. Application filed June 23, 1905.

The apparatus is essentially a combination of the electric induction furnace with a cell for electrolyzing fused electrolytes. This is accomplished by arranging the electrodes at opposite points of the annular secondary of the induction furnace containing the fused charge. The current induced in the secondary from the primary is used to melt the charge, after which the primary circuit may be opened. The current between the electrodes need only be sufficient to produce the desired electrolytic action. In general, it will be necessary to close the primary circuit only intermittently, and then for no longer than is necessary to keep the charge at a temperature of fluidity. The main advantage claimed for the apparatus is that both the electrolyzing and heating currents may be controlled independently of one another, so that each may be adjusted to the particular needs of the moment.

Electrolytic Production of Hydrazo Derivatives.—O. Dieffenbach, 833,513, Oct. 16. Application filed March 28, 1906.

The manufacture of hydrazo bodies, as hydrazobenzene and its homologues, by electrolytic reduction of the corresponding nitro, azoxy, or azo bodies suspended in the solution of an alkali or of an alkali salt according to the hitherto known process (see, for instance, the German Patents No. 120,899, Nov. 24, 1899, and No. 121,900, Dec. 1, 1900) shows the inconvenience that during the process the reduction products are precipitated in a solid form and are apt to stick to the cathodes and diaphragms so that they can only with difficulty be removed from the apparatus and at the expense of the durability of the latter. Besides, this coating or partial insulating of the cathode often causes a considerable rise of electric tension. To overcome these disadvantages, the present inventor adds to the electrolyte convenient solvents for the reduction products, which, however, are not themselves soluble in water, as for instance, benzene, toluene, etc. These solvents take up the reduction products, and at the end of the electrolysis they float on the electrolyte and can easily be removed from the cathode compartment, together with the reduction products, without removing the electrolyte itself. The following specific example is given: The cathode compartment of an electrolytic vat is charged with 100 cubic centimeters of caustic soda lye of 3 per cent strength and 100 grams of nitrobenzene, which are dissolved in 300 grams of benzene, while the anode compartment is charged with caustic soda lye of 5 per cent strength. The anodes and cathodes are of iron or nickel. The cathode liquid is strongly agitated, and about 150 amp-hours are passed at a current density of 2 to 4 amps. per square decimeter of cathode surface, whereby about 70 to 85 grams of nitrobenzene are converted into hydrazobenzene, while about 15 to 20 grams only are reduced to azoxy benzene. Both reduction products dissolve in the benzene, and when the reduction is completed they can be decanted easily from the surface of the cathode liquid. By adding the necessary quantity of hydrochloric acid to the solution in benzene, the hydrazobenzene is converted into benzidine hydrochloride, which is readily soluble in water, while the azoxy benzene remains dissolved in the benzene, and therefore can easily be separated from the aqueous solution of benzidine hydrochloride. The solution of azoxy benzene in benzene is then returned to the cathode department, together with a fresh portion of nitro benzene. Any other hydrazo derivative and the corresponding benzidine derivatives may be converted similarly into hydrazo benzene and benzidine in the above manner.

Electrically Treating Air or Other Gases.—J. H. Bridge, 832,767 and 832,768, Oct. 9. Applications filed April 3, 1906, and May 25, 1906, respectively.

832,767 covers the process, and 832,768 the apparatus. The

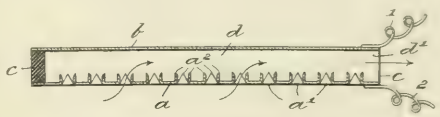

FIG. I.—PRODUCTION OF OZONE.

principle of the apparatus is shown in Fig. 1, where *a* is a metal plate with openings *a'* surrounded by projections a^2; *b* is another metal plate, while *c* is a U-shaped frame of insulating material, forming with the plates *a* and *b* a chamber *d* which has the outlet *d'*. By the connections 1 and 2, the two plates *a* and *b* are charged to opposite potentials. The air is passed through the openings *a'* and flows in the direction indicated by the arrows, being simultaneously subjected to the influence of electric discharges whereby the oxygen of the air is transformed into ozone.

On account of limitations of space, due to the convention report in our present issue, a large number of electrochemical patents had to be reserved for our next issue.

BOOK REVIEWS.

The Principles of Qualitative Analysis, from the Standpoint of the Theory of Electrolysis Dissociation and the Law of Mass Action. By Wilhelm Böttger. Translated by William Gabb Smeaton. Philadelphia: P. Blakiston's Son & Co. Price, $2.00.

Several years ago Ostwald's "Scientific Foundation of Analytical Chemistry" called the attention of the analytical chemical world to the ionic theory, and showed that in some respects and within certain limitations, a clearer and better understanding of the analytical reactions is gained with this theory, as compared to the atomic theory. Böttger's book goes into all the necessary details and is a comprehensive work on qualitative chemistry.

While with the old theory the danger is prevalent that the analytic work is dissolved into a number of mechanical operations, without the necessity of thinking about the connection of the general facts with the special facts (this danger is removed by adopting the new theory although it cannot be denied that another danger arises, namely, that of assuming more reality in some generalizations of the theory as there is than there is in them in fact). On the basis of the dissociation theory, the student must learn the "why" of every reaction, he learns the reason underlying his manipulations according to the theory.

Bauxite Brick for Lining Rotary Cement Kilns.

In our February issue of this year (Vol. 4, page 52) we published an interesting paper by A. J. Aubrey on the refractory uses of bauxite in chemical and metallurgical work.

In this connection, some information recently made public on the use of bauxite blocks for lining the hot zones of rotary Portland cement kilns is interesting. The bauxite brick is made by the Laclede Fire Brick Manufacturing Co., of St. Louis, Mo. A test was carried out at the plant of the St. Louis Portland Cement Co. with the coöperation of Messrs. Struckmann and Taylor. On March 17, 1905, the hot zone of a 6-foot rotary, 60 feet long, burning pulverized coal, was covered with a 6-inch lining of bauxite. On Feb. 22, 1906, this lining was removed after a continuous run of eleven months and five days, or 8,208 hours, during which time at least 75,000 barrels of cement had passed over this lining.

On examination the blocks showed an average loss of about $2^1/_2$ inches, while a number of them had not worn much over 1 inch, and it is believed that the blocks would easily have lasted over the year if they had not been taken out.

The following features of bauxite brick make it suitable for cement kilns: First, bauxite blocks are basic, so that they are not chemically attacked by the cement mixture. They have a sufficient degree of hardness and porosity conducive to the adhesion of the cement coating, which affords protection from direct flames, and greatly insures its length of life. Further, bauxite has a very low heat conductivity, and therefore with about the same losses of radiation a 6-inch lining of bauxite brick can be used in place of a 9-inch fire brick lining. Finally, the life of the bauxite lining is twice that of the fire brick lining.

These advantages, of course, involve a very appreciable reduction in the cost of freight and cost of erection and maintenance.

Ball Filling Material for Sulphuric Acid Towers.

In an article in our Vol. II., page 347, we referred to the great progress which has been made in recent years in connection with the chamber process for making sulphuric acid, mainly as a result of the strong competition of the new contact process.

One of the most important details of a chamber plant is the filling material for the Glover and Gay Lussac towers as well as for the intermediate towers. Coke has been used to a large extent, but since its shortcomings were early recognized, earthenware material in brick and other forms were largely introduced. Attempts, however, continued to find a more suitable form of the earthenware filling, the main principle being to get as large a surface of reaction as possible.

To some extent the problem was solved by the introduction of the perforated hollow balls of Oscar Guttmann in London, which in their outer and inner surfaces presented as large a surface of reaction as may be obtained in practice. But their disadvantage was that they had to be made by hand, and were, therefore, quite expensive. Nevertheless, it speaks well for their suitability as a filling material that they came into extensive use, especially in German chemical works, and chiefly in nitric and hydrochloric acid plants.

An important practical progress has recently been made by the Deutsche Steinzeugwarenfabrik, of Friedrichsfeld, in Baden, Germany, who succeeded in producing the Guttmann hollow balls by machinery. This means a very uniform manufacture and at the same time a greatly reduced price. They are now making balls of 100 mm. (4 inches) diameter with ten holes. This makes them available for the whole sulphuric industry.

In order to be able to give exact advice on the questions connected with the sulphuric acid industry, the Deutsche Steinzeugwarenfabrik has made an arrangement with Mr. Rudolf Heintz, in Hanover, to give such advice to all prospective users of the Guttmann hollow balls. Mr. Heintz has given special attention to the subject for many years, and from a paper, which he published in the *Chemical Trade Journal*, Vol. XXXVIII., No. 992, the following abstract of his exposition of the advantages of the Guttmann hollow balls is made:

The balls are moistened with a very thin layer of acid, both outside and inside. The gases enter the holes of the balls under a certain pressure and are at once diffused in them and brought into intimate contact with the moistened inner surface. Ball towers require no special work of erection. The balls are simply poured in, since they are equally efficient in any position, and the settling or deviation from the vertical of the tower, as well as the settlement of the balls themselves, does not influence the action in any way.

Heintz makes the following comparison of balls as filling material, with the best other kind of filling, namely, plates. In a preliminary Gay Lussac tower, which consists of twenty layers of plates, of sixteen plates each, the space occupied by plates and accessories has an area 27 x 27 meters. The plates being only 100 mm. apart, that the height is 2.3 meters and the cubical content is 10.77 cubic meters. Each plate has a surface of 0.6 x 0.6 meters, and the total contact surface of the plates is 230.4 square meters. The bearers have a surface of 76.8 square meters. Thus the whole tower has a contact surface of 307.2 square meters.

If we now fill such a tower with Guttmann hollow balls of 100 mm. diameter, it will be found that to get the same contact surface as before 5,040 balls are required, and the volume required by them is 3.6 cubic meters, that is 4.7 times less than before. In other words, if it is desired to retain the same height of the tower of 2.3 meters (which may be advisable), the cross-section may be made 4.7 times smaller. Heintz shows that under such conditions there is no danger of an obstruction of the draft.

From a practical standpoint the reduced cost is of greatest importance. While coke filling is very much cheaper than the Guttmann balls, yet the construction of the tower itself for coke filling is very much greater than for ball filling, so that it will be found on the whole that there is an appreciable saving due to the use of the Guttmann balls, which at the same time are far more efficient.

Concerning the use of Guttmann balls in different towers, Mr. Heintz makes the following remarks: Intermediate towers can, of course, be made in any shape or size, but the best rule is to make the height of that part which is filled with balls at least equal to the diameter of the tower. With the use of Guttman balls, Gay Lussac towers may be built with much smaller area and much inferior height. In fact, it will never be necesary to make them higher than 30 feet. The section of these two towers can be so calculated that the cubical content of the filled space amounts at the utmost to one fifth of that in a coke tower. Thus, for example, a Gay Lussac tower of 2.4 x 2.4 x 12 meters will be reduced to 1.48 x 1.48 x 7.50 meters, and for this 15,475 balls will be required.

Except with sulphuric burners, Glover towers will never be filled in their lower part with balls, since these would fill up with mud by the dust. On the other hand, it will be undoubtedly advisable to use balls in about the upper third of the denitration zone. There is much time and space given on account of the good distribution of the liquid, and the shape of the balls allows a considerable temperature to be reached. For this reason also a smaller number of balls will be sufficient.

The Deutsche Steinzeugwarenfabrik, as manufacturer of the Guttmann hollow balls, is represented in this country by Messrs. Fred. Bertuch & Co., of New York City.

Chemical Abstracts.

In accordance with the recent vote of the American Chemical Society the publication of a semi-monthly abstract journal will be commenced Jan. 1, 1907. The corps of those in charge of the various divisions of the journal is already well organized and work upon the abstracts has been commenced. It is intended to include in the journal abstracts of all new work in chemistry published in the world after Oct. 1, 1906. Chemical patents issued in the United States, Germany, France and England after July 1, 1906, will be included. The abstracts will be classified under the following divisions, the selection of abstractors and the oversight of each division being placed in the hands of the persons named:

Apparatus—W. H. Walker.
General and Physical Chemistry—G. N. Lewis
Photography—L. H. Friedburg.
Electrochemistry—W. R. Whitney
Radioactivity—H. N. McCoy
Inorganic Chemistry—Alexander Smith.
Analytical Chemistry—L. M. Dennis.
Mineralogical and Geological Chemistry—W. F. Hillebrand.
Metallurgy—J. W. Richards, Henry Fay.
Acids, Alkalies and Salts—T. L. Briggs.
Glass and Pottery—G. E. Barton, Albert V. Bleininger.
Cements and Mortars—Harry Drew.
Fuel, Gas, Coke—J. D. Pennock.
Organic Chemistry—M. T. Bogert.
Petroleum, Asphalt, Turpentine, Wood Products—S. S. Sadtler.
Cellulose, Paper—A. D. Little.
Explosives—C. E. Munroe.
Dyes, Textile Fabrics, Bleaching, Inks—L. A. Olney
Pigments, Resins, Varnishes, India Rubber—A. H. Sabin.
Fats, Fatty Oils and Soap—W. D. Richardson.
Sugar, Starch and Gum—C. A. Browne, Jr.
Leather, Glue—J. H. Yocum.
Biological Chemistry—L. B. Mendel.
Foods—W. D. Bigelow.
Nutrition—C. F. Langworthy.
Water, Sewage, Disinfectants, Insecticides—L. P. Kinnicut.
Fermented and Distilled Liquors—Robert Wahl.
Pharmaceutical Chemistry—A. B. Stevens.
Soils and Fertilizers—F. P. Veitch, J. H. Pettit.
Patents—W. H. Seaman.

The dues of the Society for 1907 will be increased to $8.00, to pa yfor the expense of the new publication.

The abstract journal will be published twice a month.

The *Journal* of the American Chemical Society will, of course, be continued and will contain original articles, book reviews and reviews of recent progress in the various fields of chemistry.

Notes.

Obituary.—We regret to report the death of Cav. MARIO MICHELA, President of the Societa Italiana di Elettrochimica, in Rome, on Oct. 10. Through the courtesy of the late Cav. Michela we were enabled to give an illustrated description of their works in our Vol. I, page 453, September, 1903.

University of Pennsylvania. The new building for the engineering departments of the University of Pennsylvania was dedicated on Oct. 19. The magnificent building is located at Locust and Thirty-third Streets, in West Philadelphia, and is splendidly equipped to meet all modern requirements.

Philadelphia Section of the American Electrochemical Society.—The first monthly meeting of the Philadelphia Section for the season 1906-7 will be held on Nov. 2, at Soula's Café, Fifth and Ludlow Streets, Philadelphia, at 8 P. M., preceded by an informal dinner at 7 o'clock.

Filter Papers.—We have received samples of ashless, chemically pure, toughened and starch-free filter papers made by Max Dreverhoff, in Dresden, Germany. The American representative is George D. Dodd & Co., in Philadelphia.

Chemical Courses.—Among the chemical courses for "extension students," to be held at the Polytechnic Institute, in Brooklyn, College of Arts and Engineering, during the season of 1906-7, there are courses by Prof. Olsen in general chemistry, qualitative analysis, quantitative analysis and advanced quantitative analysis, and courses by Prof. Lamb in organic chemistry and coal-tar dyes.

Tropenas Converter Steel Process.—Messrs. Powell & Cohne, of New York City, agents for the Tropenas converter steel process, have just closed a contract with the Massachusetts Steel Casting Co., Everett, Mass., for a second converter, after the first had proven very successful. Two other converters have been ordered by the Penn Steel Casting & Machine Co., Chester, Pa., and one by the Duquesne Steel Foundry Co., of Coraopolis, Pa.

Peroxides.—A pamphlet issued by the Roessler & Hasslacher Chemical Co. contains a reprint of a paper of R. von Foregger and Herbert Philipp, read before the New York Section of the Society of Chemical Industry, on earth alkali and allied peroxides; properties and applications. The authors deal with calcium peroxide, strontium peroxide, magnesium peroxide and zinc peroxide, and discuss the bleaching of oils, sterilization and preservation, the ageing of alcoholic beverages, sterilization of drinking water and of milk.

"Things Chemical."—The October issue of *Things Chemical* contains the portrait of Dr. J. E. Teeple, director of the Industrial Laboratories of the Charles E. Sholes Co.

Chemical Patents and Free Alcohol.—The October issue of the *Patent Review for Busy Manufacturers*, published by Messrs. Mason, Fenwick & Lawrence, of New York City, contains a list of chemical patents issued in the years 1889 and 1890, expiring in 1906 and 1907. They are classified under the following headings: Organic compounds, carbon lyes, dyes, dyeing processes, pharmaceutical preparations. The same issue contains a list of the important patents granted in the United States during the last twenty years for processes and apparatus for the manufacture of alcohol. This is specially interesting, since, as is pointed out, the free alcohol bill passed June 7, 1906, the provisions of which become effective Jan. 1, 1907, will have a tremendous influence upon chemical manufacturers in the United States, and especially upon the manufacture of many organic products now almost entirely imported from Germany. In the manufacture of dyes, transparent soaps, and in hundreds of processes where extraction with alcohol is necessary, the wonderful stimulus of free alcohol will be felt. Besides this, many new uses for alcohol in the household will be found and the production of suitable burners will also afford great scope to the inventor. Commissioner Yerkes is now in Europe gathering data to be used in the promulgation of rules for the manufacture and denaturization of alcohol.

Deoxidizing Alloys.—For some time past high percentage silico-spiegel alloys have been made in the electric furnace by the firm of George E. Blackwell Sons & Co., of Leverett. The cheap grades are, first, about 70 per cent manganese and 20 to 25 per cent silicon, this grade, as a rule, running about 4 or 5 per cent iron. Second, 50 to 52 per cent manganese and 20 to 22 per cent silicon, the balance being iron. Third, 30 to 40 per cent manganese, and 20 to 22 per cent silicon, the balance being iron. These alloys are largely used as deoxidizers in the manufacture of steel, but there has been a demand for perfect deoxidizers, which contain no iron or carbon. The metallurgists of George D. Blackwell Sons & Co. have been working in this direction, and have succeeded in producing an alloy containing practically 30 per cent silicon and 70 per cent manganese. The content of carbon is less than 1 per cent. The alloy, as claimed by the manufacturers to be one of the most perfect deoxidizers yet introduced into the steel industry.

Coke Ovens.—The October issue of the *Proceedings of the Engineers Society of Western Pennsylvania* contains a paper by W. M. Judd on coke-oven construction. The paper is restricted to a description of the beehive type of oven, and is fully illustrated. Concerning the waste of heat in such ovens, it is said that a portion of the waste heat is now successfully utilized at some of the more recent plants. A very effective demonstration of the power wasted by the beehive oven is given at a plant recently completed, where, from the waste heat of fifty ovens all the electric power required, both for mining operations and at the coke ovens, for two plants aggregating 800 ovens is being generated at the present time; and at this same plant, within a short time, 100 ovens will furnish the power for the operation of five mines and coke plants aggregating 1,500 ovens. If a market for the power could be secured a further development of the use of waste heat for power purposes would doubtless bring about the making of other changes in the construction of the ovens.

Personal.

Mr. Woolsey McA. Johnson, who has been acting for some time in a consulting capacity to Mr. Samuel W. Traylor, of the Traylor Engineering Co., is now also connected as metallurgical engineer with the Tri-Bullion Smelting & Refining Co., 43 Exchange Place, New York. This company owns three large mines, chief of which is the Kelly mine, Magdalena, New Mexico. The Tri-Bullion Smelting & Refining Co. intends to erect a 300-ton concentrating plant, producing 100 tons of high-grade zinc concentrates, as well as lead and copper concentrates. They will also erect either in the Kansas gas field or in the New Mexican coal fields its own reduction plant, to turn their high-grade zinc ores and concentrates into spelter.

The business of advertising the products of engineering concerns, heretofore conducted by Mr. H. M. Baxter, in Pittsburg, Pa., will hereafter be carried on under the name of Brooke & McConnell, both gentlemen having been connected in the past with Mr. Baxter in the work of the firm.

Digest of U. S. Patents.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

Ozone (*Continued.*)

No. 636,304, Nov. 7, 1899, Alexander Vosmaer, Haarlem, Netherlands.

The ozonizer comprises a metallic casing sub-divided into compartments, the walls of which constitute one of the electrodes. The other electrode or discharging surface comprises a large number of blades or points arranged parallel to these walls and in close proximity thereto. Both electrodes consist of steel, copper, brass, aluminium, or other suitable metals. The construction effects some economy of space as compared with one employing opposing blades or points, and there is said to be less liability of sparking. High-tension, continuous, or alternating currents are employed, and in some cases a condenser is arranged in shunt and an air gap in series with the discharge.

No. 636,868, Nov. 14, 1899, Henry Tindal, Amsterdam, Netherlands.

The ozonizer comprises a semi-cylindrical vessel or conduit of cast iron, to the inner surface of which is applied a thin layer of enamel, and a cover of plate glass suitably secured to the cast iron and supporting a plurality of electrodes, between which and the cast iron conduit the discharge occurs. These electrodes comprise semi-circular plates of steel arranged parallel with each other and concentric to the inner surface of the conduit. The air to be ozonized enters and escapes at opposite ends of the conduit and all of the electrodes may be arranged at an inclination to the line of flow in order to produce a whirling or like motion of the gas. It is stated that cooling is unnecessary, and that ozone of high concentration is obtained by passing air through the extended, even and dense field of discharge provided by the apparatus.

No. 640,694, Jan. 2, 1900, Marius Otto, Neuilly-Sur-Seine, France.

The discharge occurs between opposing points or edges, carried by the walls and partitions of a suitable cast iron casing. In order to prevent arcing between the points there is interposed between each pair of electrodes a rotating plate, the purpose of which is to interrupt the discharge at short intervals. Several constructions are described. In case the electrodes are in close proximity the rotating plate is of insulating material, having portions in the form of sectors cut away, the discharge being interrupted over any given portion of the electrodes as they are covered by the solid portions of the plate. In some cases the electrodes may be arranged beyond the normal discharging distance, and the rotating plate will then be of conducting material; in such case the discharge will occur as the solid portions of the plate pass a given portion of the field. In either case the sectors, instead of being entirely open can be provided with a very thin sheet of insulating material, thereby making a more complete baffle for the circulation of the air and increasing the efficiency of the apparatus.

No. 642,663, Feb. 6, 1900, Charles G. Armstrong and Wm. D. Neel, Chicago, Ill.

The ozonizer comprises a casing of insulating material, as glass, sub-divided into three compartments by two vertical insulating partitions. The electrodes consist of copper wires incased in heavy glass tubes, which are mounted in the central compartment in parallel relation and are supported by the partitions. The tubes carrying the electrodes of one polarity pass through one of the partitions, and are mounted in shallow recesses in the other, this construction being reversed for the tubes carrying the wires of opposite polarity. The wires constituting the electrodes of each polarity are brought into the respective lateral compartments and suitably connected. Perforated distributing plates are arranged at the inlet and discharge ends of the casing, the flow of air being transverse to the electrodes.

No. 648,764, May 1, 1900, Joshua H. Lamprey, London, England.

One electrode consists of a plate of metal covered on each side by sheets of dielectric material, as glass. The electrodes of opposite polarity are supported by these sheets, and comprise metallic wires having metal beads or discs strung upon them and spaced by insulating beads. The insulating beads are of larger diameter than those of metal, whereby contact of the latter with the dielectric is avoided. No casing is shown or described.

No. 653,078, July 3, 1900, William Elworthy, London, England.

A casing is sub-divided into two compartments of unequal size by means of a horizontal partition, the lower compartment being the larger. A large number of glass tubes, open at both ends, are so mounted in this partition as to project slightly into the upper compartment. A metallic rod is inserted in each tube, the several rods being connected together and constituting one electrode or discharge surface. The opposite electrode comprises a coil of metallic wire, cord or chain encircling the portion of each tube which projects into the lower compartment, being spaced from the tube by suitable means, as, for instance, a reversely coiled dielectric strip. Air is admitted to the upper compartment, passes through the several tubes, is deflected upwardly through the lower compartment and discharged at the upper portion thereof, thus passing twice through the field of discharge, once within the tube and once between the tube and the exterior coil. All electrical connections are made in the upper compartment, and are, therefore, not subjected to the action of the ozone.

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J. W. RICHARDS, PH. D.....*President*
E. F. ROEBER, PH. D.....*Editor*

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Laws Governing the Promotion of Mining Companies.

In our November issue we showed how anxious the general public was to speculate in mining stocks, if once imbued with the idea that the cards were dealt fairly. Although there has been a great deal of endeavor on this part, nevertheless it has been fruitless of any widespread uniform legislation, but fruitful of an active public interest in the question. How far the work should be done by States and how far by the Federal Government is the great issue formulated by Jefferson and Hamilton a century and a quarter ago. At the same time the consideration is brought up how far a republic, which in its nature is individualistic, should lean towards paternalism.

* * *

Undoubtedly some protection of the lambs against the wolves is necessary. But the endeavor should be in the line of educating the public so that the public protects itself. Any gross misstatement in the prospectus of a mining company is in essence an attempt to obtain money under false pretenses and thus a criminal offense—the offender to be prosecuted at the public expense. However, in the mining business it is so hard to estimate the "prospective value" of a mine, so hard to "look into the ground," that many a loophole would be left for the unscrupulous promoter. If we could always rightly discriminate between the true and the false, human life would become very tame. But something must be done to restrict the fleecing of the public by the use of advertisements and the mail, just as has been done by the national government in the case of the lotteries. A curb on those practicing the gentle art of swindling would be in order.

* * *

A possible remedy suggests itself. Perhaps a national commissioner of mines with power to pass on mining enterprises offered the public, and to keep the statements to something like the truth, to eliminate pure "wildcats" and still leave the legitimate prospect, would help clear the atmosphere. But we frankly confess that this is a most difficult question, and the needed reform can only come as a result of much discussion. It should be remembered that the public are more willing to gamble on a good prospect if assured of a fair deal than to invest in a developed mine

Electric Steel, the Government and the Promoter

In another column of this issue will be found an abstract of an article from the New York *Sun* which deals with the unfortunate results which have followed the semi-official or unofficial newspaper reports of the government's experiments at Portland, Ore. We have already had occasion to refer to

these experiments and the erroneous impression they evidently made in the mind of the lay public. All misinformation is harmful, but the harm done is increased ten-fold when the erroneous information is apparently stamped with the government's approval. As is well known, one of the most widespread fallacies is the belief in the infallibility of governmental enterprises. In this case, the extravagance of the assertion that the government's experiments demonstrated that steel could be made directly from black sand at a cost of $12 per ton, is probably sufficient warning to the more intelligent layman that there must be a mistake somewhere. Therefore, the Black Sand and Gold Recovery Company shows much wisdom in addressing its appeal for subscriptions to the "plain people" and the "small investor," for in them is found the most profound belief and the most enduring faith in the infallibility of governmental work of this character.

* * *

It is possible that if the results of the government's work had been first communicated to the public through the regular channels and had not been exploited in advance in the columns of yellow journalism, more conservatism would have been shown in their presentation. We have no doubt that when the official report of Dr. Day appears, it will be found to be quite a conservative document. The metallurgical purpose for which the Geological Survey's research was carried out—in which the electric smelting experiments formed only an incident—may then appear in a new light; and for this reason we hope that the appearance of the report will not be delayed too long. Even so, the question might still be fairly asked: What good end has been served by the government's experiments with the electric furnace? Any one with the most elementary knowledge of the electric furnace and the metallurgy of iron would know that the production of steel in a furnace like that used in the government's experiments is wholly impracticable. While it is true that a low-carbon iron may be produced in such a furnace, it is not practicable to manufacture in it those irons, having definite carbon contents, which we call steel. As for the production of pig iron in the government's furnace, it is extremely doubtful that any useful information could be thereby obtained. Experiments in this direction have been carried out on a comparatively large scale by others who possessed a far better equipment as regards apparatus and experience. Moreover, the results obtained are easily available.

Among the opinions which have been expressed as to the object of the government's experiments, we notice one, stated with delightful frankness, in our Denver contemporary, *Ores and Metals*, which takes us to task for failing to perceive the true significance of the government's work. It is pointed out that Dr. Day's experiment "will make it less difficult to secure capital when conditions are ripe, and as a result many engineers who have known all about the facts for years will be given employment." It would be difficult to gainsay this in view of the [illegible] but in spite of the astonishingly rapid growth of the government's ubiquitous activities, we doubt if any department is as yet designed to assume the duties of a promoter of steel companies or an engineers' employment agency.

The Metal Markets.

As was clearly indicated in our July editorial on this subject, the fall metal markets have followed the precedent of the last year and displayed unusual strength after early buying. The demand for metals results from basic reasons for expansion. Solid and substantial buildings and factories require metal in all their parts. New factories require new machines. Everywhere is iron, steel, copper, zinc and lead required in new construction. The electrical business in all its complexities requires chiefly iron and steel, but copper most of all. It also uses zinc for primary batteries and lead for storage batteries and for lead-covered underground cables. But the rapid expansion of all the electrical trades has put the copper producers under tremendous physical strain for simply quantity. The pig iron and steel markets are wisely held at constant points by the denominating interests, but of late have shown a great natural strength. Possibly the statistical position in the iron and steel markets is the strongest at the present time.

* * *

Lead market is especially strong, for the scarcity of lead ores is most actual. Spelter is in great consumptive demand, and has risen forty points since September. The presence of several smelters with inadequate financial resources tends to hold spelter down. However, the great demand for galvanized sheets, due to high prices of lumber, for which they are a substitute, and the tendency to put up substantial structures, as well as the demand on the spelter producers by the other galvanizing industries and the brass trade, have given the spelter market great basic strength. Considered in their entirety, the metal markets are all in good shape. The demand, though at times excited, is in main healthy, because of the great growth of this country. The mining and metallurgical engineers are certainly doing their share for the good of the country in producing the metals which are the framework of industrialism.

Progress in Reverberatory Copper Smelting

In the past three years there has been a marked tendency to revert to the reverberatory copper furnace for the purposes of smelting copper ores or concentrates with the production of a copper matte carrying the precious metals. While the vertical blast furnace has made great progress, especially in the "pyritic" line, nevertheless, the open flame reverberatory has made greater irrelative advances. The rivalry is a survival of the old struggle between the "Welsh" and "German" style of smelting described in both the works of "Percy" and "Peters."

* * *

The shaft furnace has the advantage of continuity of operation and the reduced labor charge resulting therefrom. While its fuel is more expensive, yet the proportion of fuel to charge is so much less that with the combustion of the sulphur and iron of the charge the advantage in this respect is on the side of the Teutonic method. The cost of repairs is also less. Here the superiority of the shaft furnace ceases and that of the reverberatory commences. Perhaps the latter's greatest advantage lies in the fact that its flue dust losses are nearly nothing, while in the other style the losses are considerable with high blast pressures now used, and with large proportion of the flue dust necessarily recharged. This results in a cycle of

"fines," and thus increases smelting charge. In the matter of slags the Welsh method is far superior. With a proper mechanical mixture there is no danger of freezing, for the heat of the furnace can be regulated at will—an innate point of superiority of the intermittent type. There is also a far wider latitude in slag composition, which circumstance was used to advantage in "Highland Boy" plant in the Bingham district. Due to the fact of high heat and large pool, more or less quiescent, the slags are usually cleaner. There are possible places where the reverberatory smelting practice can be improved, such as by the use of the "Eldred" process for delaying combustion and the return of part of the sensible heat of chimney gases now used for raising steam by recuperators and by improved refractory brick for arch.

* * *

The reverberatory smelting process was mainly introduced into this country by a man with Welsh experience, Prof. Richard Pearce, who used it with great success at the "Argo" plant in Denver. His furnaces once considered large are now small compared to the mammoth reverberatories of the Anaconda "Washoe" works. The installation of reverberatories is rapid, and the "Phelps-Dodge" interests and the Nevada Consolidated Copper Co., among others, are taking active interest in this subject. It will be used largely for treatment of roasted fine concentrates and "flue dust" from shaft furnaces or in localities where coal is cheap and coke dear, on ores which when mixed produce a slag of high "fluidity-point." Put in its proper place or combined with the shaft furnace it is a highly efficient metallurgical instrument for extracting values from copper, silver and gold ores.

A National Department of Mining and Metallurgy

One of the marked facts bearing on our commercial life to-day is the increased yield per acre of corn, wheat, cotton and other staples in the great agricultural States. This in part is due to the better financial position of the farmers, but it is more due to the diffusion of agricultural science. The Department of Agriculture at Washington spends some seven million dollars each year in scientific and practical research. The different State agricultural stations, connected often in the West with a magnificent State university, supplement the work. In farming, the base of all wealth, for we are all dependent on our stomachs, the ideals of the twentieth century, the application of broad truths of science to the amelioration of human life are manifest. And the direct realization of these ideals is accomplished through a great national system for the acquiring of new knowledge and the distribution of information. Although this may not have the effect of doubling the yield per acre in less than ten years, the enthusiastic claim of Secretary Wilson, yet it is a fact that it is making now a national saving that is measured in the hundreds of millions of dollars.

* * *

Next to farming, the mining of ores and getting their valuable contents in marketable shape is the most important basic source of wealth. While there is much done by the United States Geological Survey and the Commissioner of Mines or State geologist in the several States, yet the organization is far from perfect and not to be compared to Secretary Wilson's splendid force. The work of the Coal Testing Plant of the United States Geological Survey, at St. Louis, is an example of what might be done in all forms. The tests on gas producer working on lignites have shown possibilities in the Western coal deposits of the greatest [illegible]

While the work at St. Louis is good, yet it has been necessarily of a restricted nature and in only one industry. The smelting of iron ores, the manufacture of steel, the smelting of copper ores and the refining of copper, are great branches of the mineral industry. In these and in the treatment of zinc, lead, silver and gold ores the elimination of waste and the recovery of by-products are being accomplished, but by a slow and painful progress. If all the attempts at progress were reported and filed at Washington, if systematic scientific investigations were made of the chemical reactions of the different processes, if all this were done by one great bureau with a view to the practical application, the increased efficiency in mining and metallurgical operations would be astounding, because it is easy to make changes when comparatively few men's minds must be changed. Of course, the consolidation of experience and the direction of research when done at Washington is a work of great magnitude. It would be supplemented by the work at the universities and technical schools and engineering societies, as well as by the research of the engineering staffs of the large corporations, and if rightly laid out would not interfere with the endeavor of any one. But it would rather prevent the reduplication of experiments and would co-ordinate metallurgical experience.

* * *

Canada has done just about what we have outlined, but it is easier to effect such a result in the smaller country. The admirable reports on electric iron smelting and steel making, and the later report on the zinc resources of British Columbia, which we reviewed in our November issue, are examples of the good work done by the Dominion Government in this line. Germany also is most provident in her utilization of national resources, but her governmental direction and interference would not be allowed in this country. Anyhow, her celebrated Physikalisch-Technische Reichsanstalt has found successors in this country and in England, and now a Chemische Reichsanstalt is earnestly contemplated. The chemical industries represent perhaps the most flourishing commercial activity of Germany, and this great national success has been essentially due to the fact that for the German chemical manufacturers commercial chemistry has been an applied science. Their most valuable raw material was and is the army of trained chemists turned out every year by the German universities. Under existing conditions, German chemists naturally favor the manufacture of the more complicated products of organic chemistry. Reversely, in this country, with its abundance of valuable ore resources, applied chemistry will naturally remain metallurgy for years to come, though its future metallurgical developments will probably be guided more systematically by scientific principles than has been the case in the past. At all events there is a crying need in this country for a great national bureau to take over and amplify the work of the Geological Survey, and increase the operating efficiency of the processes of extracting economic values out of the ground.

The Iron and Steel Market.

Buying of pig iron and finished steel products was relatively light during November.. The market had been well sold up and the very heavy buying could not be continued indefinitely. The chief interest now is in specifications against old contracts and the success the furnaces and mills meet in endeavoring to fill them.

As was expected, October established new records of production in a great many departments and aided by good weather the pace was maintained in November. Production of coke and anthracite pig iron in October was 2,196,808 gross tons, a gain of 23,000 tons over March, 1906, the previous record month. The furnaces controlled by steel companies made a greater gain than this, the merchant furnaces falling some 30,000 tons behind. The furnaces operating in March did not make as much pig iron in October, the gain being due to the completion of new furnaces. As more of the older furnaces are brought into the producing class, new high records can be made. We estimate the production of all grades of pig iron in 1906 at about 25,300,000 tons, but the furnaces now completed can make a considerably larger tonnage in 1907, and there are other furnaces now nearing completion.

There being no considerable stocks of steel-making pig iron, production of steel has been equally large. The Illinois Steel Co., for instance, made in October, 212,547 gross tons of steel ingots, against its best previous record, made in September, of 192,385 tons. Individual finishing mills have not shown a proportionately large number of new records, for the reason that all finishing mills have been full of business, and the steel supply had to be distributed. When only a portion of the finishing mills demand the greatest output they can be better supplied with steel. Finishing capacity is always in excess of steel capacity.

The Ore Situation.

On Friday, Nov. 2, an informal arrangement was made by ore sellers as to prices on Lake Superior ores for the forthcoming season. Nominal advances were made of 75 cents on Bessemer ores and 50 cents on non-Bessemers, but the base iron content, maintained without change for years, was reduced, so that the actual advances were larger than the nominal. The reduction in the base was forced by the increasing leanness of the ores. In the old days only the choicest deposits were mined. As demand increased, less desirable deposits had to be mined, while the older deposits are becoming exhausted. It may be estimated roughly that the average iron content of all Lake Superior ores shipped is decreasing by between 1 and 2 per cent per year at the present time. No other outcome could be expected. Shipments this year will be in the neighborhood of 37,000,000 tons, and predictions for next year are at 40,000,000 tons and more. In 1899 they amounted to only 18,251,804 tons, and in 1800 and 1892 they were a trifle over 9,000,000 tons, the years here mentioned having been record years in their time, so that the output has doubled twice in sixteen years.

Until last year the basis was uniform as follows: On Bessemer ores, 63 per cent iron when dried at 212°, and 10 per cent moisture in the natural state, making the iron content, natural, 56.7 per cent; on non-Bessemers, 60 per cent iron, when dried at 212°, and 12 per cent moisture in the natural state, making 52.8 per cent iron in the natural state. Last year one merchant firm departed from the custom and guaranteed 53 per cent iron on Mesabi non Bessemers, the innovation being accepted by most sellers. In establishing a new basis even figures were selected for the natural, instead of the dried state, although the moisture content is unchanged, the new bases being, in the natural state, Bessemers, 55 per cent., non Bessemers, 51.5 per cent. In the following table is shown the 1906 season price, the 1907 season price, on the new basis, and the price which an ore of the previous guaranty would bring on the 1907 price basis:

	1906 Base.	1907 Base.	1906 Base at 1907 Price.
Old Range Bessemer.....	$4.25	$5.00	$5.154
Mesabi Bessemer.....	4.00	4.75	4.897
Old Range non-Bessemer...	3.70	4.20	4.306
Mesabi non-Bessemer.....	3.50	4.00	4.116

These prices are f. o. b. lower lake port. Contracts are usually made for a season's supply, to be shipped during the season of lake navigation, while payments are made in twelve equal monthly installments, May 1 to April 1. Only a comparatively small percentage of the Lake Superior output is sold on this basis; more than half the ore is mined by consumers, while a considerable part of the balance has been sold on long-time contracts.

Most consumers promptly bought ore on the announcement of prices. A few have held off, being unwilling to commit themselves so far ahead, as the consuming season for ores shipped next season runs to about July 1, 1908.

Pig Iron.

The market has continued to move upward since last report. The pressure for prompt delivery continues as the great increase in production, although aided by imports at the rate of 10,000 to 20,000 tons per week, nas not filled the demand, and prompt lots are bought at higher and higher prices. Furnaces being well sold through first quarter, and in many cases also through second quarter, are disposed to exact prices for such deliveries pretty closely in line with prices for prompt. For later delivery the market is naturally on a lower basis. Buying is not heavy. For prompt shipment we quote foundry iron at close to $25, valley furnace, and $22 to $23, Birmingham. A few sales of Bessemer and basic have been made for prompt and first half delivery at from $21.50 to $22.60, valley, while for second half the market on these grades is about $21, valley, although no large sales have been made.

Billets and Sheet Bars.

There is no regular market on billets. All consumers are covered more or less fully by contracts and only buy as they can supplement their deliveries by odd lots at reasonable prices. Sales have been made at somewhat higher prices, and Bessemer billets may be quoted at $29 and open-hearth at $30 to $32. Sheet bars are about $30, Pittsburg, for long lengths.

Finished Material.

Since last report the following advances have been made:

Tin plates, 15 cents a box to $3.90.

Black sheets, $2 a ton to $2.60 for 28 gauge.

Galvanized sheets, $2 a ton to $3.65 for 28 gauge.

Blue annealed sheets, $1 a ton to $1.80 for 10 gauge.

Painted corrugated roofing 10 cents a square to $1.85.

Galvanized corrugated roofing 5 cents a square to $3.15.

Wire products $1 a ton to $1.90 for wire nails, $1.75 for plain wire and $2.35 for galvanized barb wire.

Light rails $1 a ton to $32 for 20 to 45-pound sections.

Merchant steel bars $2 a ton to $1.60

An advance in plates from the $1.60 basis is likely to be made at any time to $1.70. The Eastern mills about the middle of November began quoting on a basis of $1.70, Pittsburg.

Beams and channels remain at $1.70 for 15-inch and under and $1.80 for larger sections, and are not likely to be advanced.

Electric Steel

The New York *Sun* of Nov. 21 publishes an article on steel direct from ore, covering almost four columns of its editorial pages.

The article is different from anything that has been pub-

lished in daily newspapers on electric steel. While sensational in its arraignment of the scheme of the Black Sand & Gold Recovery Co., the article appears to be extremely careful in the statement of facts, and is at the same time conservative in the best sense of the word in the discussion of the possibilities of electric furnace methods in iron and steel metallurgy. The anonymous writer in the *Sun*, who is evidently an electric furnace expert, intimately connected with the iron and steel industry, expresses the following "safe and sane" views on the possibilities of making iron and steel in the electric furnaces.

Concerning competition of the electric furnace with the blast furnace for pig iron production, he says that such competition is impossible where coke is cheap. Only under exceptional conditions where electric power is cheap and coke dear it may be possible to produce pig iron at a lower price than the blast furnace product can be marketed.

Concerning the manufacture of steel in the electric furnace, reference is made to two types of furnace which have proven commercially successful. One is that of Héroult and Keller; the other the induction furnace of Colby and Kjellin. For the manufacture of high-grade tool steels (such as are now made by the crucible process) the electric furnace has already distinct advantages, and it is possible that, with the development of the art, the electric furnace may in time also encroach upon the field of the open-hearth furnace.

The main portion of the article in the *Sun* is, however, devoted to a severe criticism of the methods employed in the exploitation of a scheme dealing with the production of steel direct from ore. The article quotes largely a prospectus issued by the Black Sand & Gold Recovery Co., with head offices in Chicago, which prospectus bears on its front cover the following statement:

"This pamphlet tells how the Government of the United States has recently made steel in commercial quantities for $15 per ton less than the Steel Trust can make pig iron." The Black Sand & Gold Recovery Co. proposes to work on the black sand on the Pacific Coast, this sand containing considerable quantities of magnetic oxide of iron, and to produce steel in electric furnaces. The principal basis of its appeal for subscription to its stock is certain work carried on recently under the auspices of the United States Government.

In connection with the black sand deposits of the Pacific Coast, Congress made an appropriation of $25,000, afterward increased by a second appropriation to $50,000, for their investigation. This investigation was carried out under the supervision of Dr. David T. Day, chief of the Division of Mining and Mineral Resources, United States Geological Survey. The complete official report by the government on this work has not been issued, but certain semi-official articles have appeared giving some information as to the government work. These articles have chiefly appeared in daily newspapers and their general tone is highly sensational.

Special reference is made to an article in the issue of the Chicago *Inter-Ocean* of Sunday, Jan. 21, 1906. This article has received the endorsement of Dr. Day in a letter to the business manager of the *Inter-Ocean*, which is reprinted in the Black Sand & Gold Recovery Co's prospectus. In this letter Dr. Day says that the article in the *Inter-Ocean* "requires but little in order to make it perfect. * * * The black sand subject is really one of very great interest, and is going to aid very much good citizenship on the Pacific Coast, for it is not a matter of speculation, but simply of untiring industry, with all the personal improvements of character which come by that kind of work as contrasted with the usual speculation so frequent in the mining industry."

From this article in the *Inter-Ocean* the writer in the *Sun* quotes the following passages:

"Dr. Day, after demonstrating that many of the sands tested contained as high as 600 pounds of iron to the ton, erected a 10-ton electric furnace, and in one short hour, by adding lime and broken coal to the iron which had been separated from the black sands, smelted the iron into high grade steel." The immense importance of the steel industry on the Pacific Coast is discussed, the construction of "battleships from sand" is prophesied, and so forth. Finally, the article is summarized in a series of results which "Dr. Day and the work of his associates have demonstrated beyond question." Among these results are the following:

"That in order to utilize this iron (magnetite) commercially it must first be magnetically separated from the sand without drying."

"That the magnetic iron (oxide?) when separated from the sand can be reduced in an electric smelter to commercial iron and steel ready to supply Pacific Coast markets."

"That the steel so produced, smelted with cheap electricity generated from water power, need not exceed $12 per ton, while the present cost of pig iron on the Pacific Coast is over $27 per ton—a saving of $15 per ton over present prices of pig iron alone."

These claims are most severely criticised by the writer in the *Sun*, who refers to an article in the *Technology Quarterly*, written by Prof. R. H. Richards, who was associated with Dr. Day in the black sand investigation. There Prof. Richards states that the furnace "is still in the experimental stage, as to the means of controlling the contents of the steel in silicon carbon, manganese, sulphur and phosphorus." The writer in the *Sun* points out that it is obviously absurd to talk of manufacturing steel when the contents in carbon, silicon, etc., cannot be controlled.

Many investigators have made extended experiments to make steel directly from the ore in one operation, but all have failed so far in solving this problem, and "we may feel confident that this was not done in the crude experiments made by the Government at Portland," especially if the type of furnace used in these experiments is considered, which is in no way suitable for making steel. "Under these circumstances it would be a waste of time to discuss the possibility of turning out steel from an electric smelter at $12 a ton."

Concerning magnetic separation, the writer in the *Sun* makes the following criticism:

"Now, as regards the magnetic separation of the magnetic oxide of iron from the black sand the Black Sand & Gold Recovery Co. publishes a letter, apparently written on the United States Geological Survey's paper by Mr. J. F. Batchelder, who is now the company's superintendent, in which Mr. Batchelder asserts that as a result of certain tests made at the request of Prof. Richards, it was found that the Lovett magnetic concentrator was a most excellent machine. This machine is said to be controlled by the Black Sand Co. It is interesting to note that in a letter, signed by Dr. Day and printed in the *Engineering and Mining Journal*, the work with Prof. Richards is referred to, and it is asserted that no machine owned by the Black Sand & Gold Recovery Co. was tested. In another part of his letter Mr. Batchelder says that one of the strong features of the Lovett machine is its ability to handle the material wet. Thus it is not necessary to dry the sand, 'which must be done in the case of every other separator.' This is absolutely false, for there are other wet separators."

In taking up again the pamphlet of the Black Sand Gold Recovery Co. the writer in the *Sun* says: "But it is not only the fatuous experiments of the Government's employees that are used to excite the plain people to buy stock in the company, but various forms of subtle misrepresentation are also employed. For example, the prospective investor is induced to infer, after reading the literature furnished him by the company, that one C. F. Wilson, who is described as 'the foremost furnace electrical expert of this country' is T. L. Wilson, whose name is intimately associated with the development of the calcium carbide industry in America. Even if the blame for this misapprehension can be shifted from the company to its

dupes, no excuse can be found for another ambiguous suggestion. One of the illustrations in the company's pamphlet shows 'pig iron melted in the electric furnace,' and the reader is led to infer that this pig iron was manufactured by Dr. Day and his associates or by the Black Sand & Gold Recovery Co. The illustration actually had its origin in a photograph of pig iron taken during the experiments made under the auspices of the Canadian Government in Sault Ste. Marie during January and February and March of this year. Perhaps reference to those experiments is avoided because from the results it has been estimated that under favorable conditions crude pig iron might be produced at about the cost advertised by the Black Sand & Gold Recovery Co. for finished steel. But then such facts as this would be of no assistance in the sale of stock to gullible 'plain people.'"

"An air of stability is given to this questionable enterprise by the fact that on the Pacific Coast are found some of the conditions that favor the use of the electric furnace, such as a high price for fuel, possibilities for cheap development of water power and somewhat higher prices for finished products than are found in the East. Probably it is considerations such as these which have induced such well conceived enterprises as that of Mr. Noble, of the Northern California Power Co., who is putting up an experimental electric furnace plant, to make pig iron from the ore of the Shasta Iron Co." (Concerning this undertaking, in which the Héroult furnace will be used, probably in essentially the same way as in the Sault Ste. Marie experiments, some information was given in our November issue, page 441.)

The writer in the *Sun* concludes as follows:

"It is lamentable that just when the electric furnace is emerging from the laboratory into a useful metallurgical apparatus it should have to bear the odium of such irresponsible wildcat exploitations as that of the Black Sand & Gold Recovery Co. But it is still more deplorable that such schemes should find a basis in the inexpert activity of governmental bureaus."

We comment on the article on another page.

CORRESPONDENCE.

Secretaryship of American Electrochemical Society.

To the Editor of Electrochemical and Metallurgical Industry

SIR:—As Secretary of the American Electrochemical Society, I desire, through your columns, to lay before the membership of the Society a matter that has given me some thought for months past. Although the duties of the office of Secretary have been very congenial to me, and the acquaintances and friendships made while occupying the post, very greatly appreciated, I feel that I will have to decline to be a candidate for re-election at the next annual meeting. To do the work justice requires constant thought as well as work, and the growing demands of my professional business are now pressing for more of my time than is available under the present arrangement.

I thought it best therefore, to give this timely notice, that the members might find some one for the position who could keep in touch more constantly and thoroughly with the lines of research and industrial development in the field of electrochemistry. In this way the high standard of the Transactions and membership may be kept up and the well being of the Society in general promoted. SAMUEL S. SADTLER.

PHILADELPHIA, PA.

[Prof. J. W. Richards, of Lehigh University, declared in a letter to this journal, February, 1904, when a similar question arose as to the secretaryship of the Society, that "he is willing to serve the Society in any capacity in which the members think he can best aid it and promote its interests." We feel sure that Dr. Richards has not changed his attitude towards the Society, and that his acceptance of the secretaryship would probably represent the best possible solution of the problem now confronting the Society.—EDITOR.]

The Mining Boom.

To the Editor of Electrochemical and Metallurgical Industry:

SIR:—I have read with interest the editorial on the "Mining Boom," in your November issue, and especially that part which refers to the increase in the values of mines due to the advance in the art of metallurgy.

I could have wished that you had made particular reference to the zinc business. To my knowledge, two zinc properties, the "Kelly" mine, in New Mexico, and the "Iron-Silver" mine, in Leadville, went begging, the latter four years ago, the other two years ago. The figure the mines were offered for was in each case less than $100,000. Later, advance in handling complex sulphide ores by modern methods of concentrating and in handling zinc ores high in iron by Kansas zinc smelters has brought it about that these mines are each netting several hundred thousand dollars per year, and are worth a good many million dollars besides. I mention these two, for they are the most conspicuous examples in the zinc business; of course, there are many similar instances. The list might be extended greatly, and in the future we can expect similar great enhancement in the value of mines now considered worthless, due to "the advance in the state of art."

New York City. WOOLSEY MCA. JOHNSON.

Specific Heat of Iron.

To the Editor of Electrochemical and Metallurgical Industry:

SIR:—I do not know whether your readers have yet seen the very valuable determinations of the specific heat of iron by Mr Harker[1]. They are of particular interest in that they confirm in general the early results obtained by Pionchon, when the latter are corrected in accordance with our present values of some of the constants which he used.

I enclose a copy of Mr. Harker's results, together with those of Pionchon as originally given and as recalculated by Mr. Harker:

TOTAL HEAT OF IRON BETWEEN 0° AND T°.—O_o^t.

T	*Pionchon's Values from Formula.*	*Pionchon's Values Recalculated.*	*Harker's Results.*
200	23.5	23.5	23.5
300	36.8	36.8	37.0
400	51.6	51.6	51.3
500	68.2	66.0	66.9
600	87.0	83.2	83.8
700	108.4	102.2	104.1
800	135.4	125.0	127.8
900	157.2	146.7	148.0
1000	170.0	160.0	155.7
1100	.....	.....	168.8

MEAN SPECIFIC HEAT BETWEEN 0° AND T°. - S_o^t.

T	S_o^T.	*T.*	S^T.
200	.1175	700	.1487
250	.1204	750	.1537
300	.1233	800	.1507
350	.1257	850	.1647
400	.1282	900	.1644
450	.1311	950	.1642
500	.1338	1000	.1557
550	.1361	1050	.1542
600	.1396	1100	.1534
650	.1440	....	.....

Dr. Harker's paper appeared in the *Philosophical Magazine* for October, 1905. HENRY M. HOWE.

Columbia University.

[1] An abstract of Dr. Harker's paper appeared on page 136 of our vol. III. We are glad, however, that Prof. Howe has been good enough to again call attention to this work, and to send us the above tables, which are fuller than those published before in our columns on the subject.—EDITOR.

The Extraction of Metallic Sodium.

BY C. F. CARRIER, JR.

(*Concluded from page 446.*)

ELECTROLYSIS OF FUSED SODIUM CARBONATE, 669.733.374.3.

Only the process of H. Becker can be classed in this group. This inventor claims to increase the efficiency, decrease the cost and obviate explosions by using a mixture of sodium carbonate and sodium hydroxide as the electrolyte. He assumed that the carbonate would be decomposed in the presence of the hydroxide. This process is carried out in an apparatus that is very similar to the Castner apparatus, except that there is no metal screen diaphragm between the anode and the cathode, and the metal is collected in a water-cooled collecting hood. A full description of the process and apparatus has already been published in this journal, and it has also been shown by the author in an investigation undertaken in the laboratory of Prof. Max Le Blanc, that the fundamental assumption of the process is false, no carbon dioxide being liberated by the electrolysis of a mixture of sodium carbonate and hydroxide. If a pure bath of carbonate is used the melting point is so high that the sodium liberated will reduce the carbonate, leaving free carbon in the bath. (Private communication from Prof. Haber's laboratory, Karlsruhe.)

References:

This Journal 1; 574 (1903) and 2; 357 (1904).
U. S. P. No. 663,719 of Dec. 11, 1900.
Also patents in Eng., Ger., Fr. and Switz.

ELECTROLYSIS OF FUSED SODIUM HYDROXIDE, 669.733.734.4.

In this group come the only processes that, up to the present time, have been commercially successful in producing sodium. Like natural phenomena, invention first tends to follow the line of least resistance, so it is quite natural to expect the first successful solution to be by use of the easily fusible sodium hydroxide. Although the reduction of sodium from sodium hydroxide has been in commercial use for a number of years, the exact mechanism of the chemical reactions is still a matter of controversy.

It was observed by Janeczek in 1875 that both oxygen and hydrogen are obtained by electrolysis of an anhydrous bath of fused potassium hydroxide. He also noted that water condensed in his gas receivers, and concluded that OH ions must have been discharged at the anode. The evolution of hydrogen he attributed to the action of potassium on potassium hydroxide. It seems more reasonable to assume, however, that the reaction must be between water and potassium, for the OH ions decompose as soon as discharged, forming water and oxygen. A portion of the water escapes as vapor, as noted above, but all the water cannot be expelled by heating either potassium or sodium hydroxides to their melting points, or slightly above. This portion of water remaining in the bath will, therefore, be decomposed by the current in preference to the alkali base, water requiring but 1.69 volts, while sodium hydroxide requires 2.2 volts. It would, therefore, seem probable that a portion of the hydrogen at least is produced primarily by electrolysis.

Reference:

Berlin. Ber. 8; 1,018 (1875).

The electrolysis of fused sodium hydroxide with a cathode just touching the surface of the bath was investigated by Lorenz and Sacher in 1901. They made determinations of the voltage of decomposition a problem, which was more thoroughly investigated by Le Blanc and Brode, although the results of both substantially agree. It was found that the voltage of decomposition for anhydrous sodium hydroxide is 2.2 volts, but when water is present there is also a lower decomposition point at 1.3 volts. The formation of water from two OH ions at the anode was also demonstrated, and at high temperatures it was noted that hydrogen also was produced at the anode. This they attributed to the action of sodium that had diffused from the cathode to the anode, where it reacted with the water formed from the OH ions.

References:

Lorenz and Sacher, Zeit. f. anorg. Chem. 28; 385 (1901).
Lorenz, Zeit. f. El-ch. 8; 873 (1902) and 9; 155 and 333 (1903).
Le Blanc and Brode, Zeit. f. El-ch. 8; 697, 717, 817 and 930 (1902), and 9, 230 (1903).

Hambuechen has shown that with alternating electrolysis it is possible to separate sodium from sodium hydroxide by means of an alternating current. Haber has shown the electrolysis of the same compound in the solid state.

References:

Hambuechen, Trans. Am. El-ch. Soc. 4; 165 (1904).
Haber, Zeit. f. anorg. Chem. 41; 407 (1904).

As early as 1866 an English patent (Dickson, No. 2,202 of 1866) specified the use of sodium hydroxide as a raw material in the production of sodium, but it remained for H. Y. Castner to devise the first practical electrochemical process for the production of sodium. His process has been in commercial

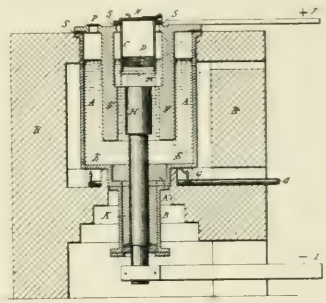


FIG. 5.—CASTNER CELL.

commercial operation from the time of its first appearance until now. The process consists in electrolyzing fused caustic soda in a specially constructed cell, at not more than 20° C. above its melting point. This cell consists (Fig. 5) of an iron melting pot A, suitably mounted in masonry. The cathode H is of metal, and is introduced into the cell through the bottom by means of the sleeve-like extension B, which is filled with solid electrolyte to insulate the cathode from the pot, hold the cathode in place and seal the joint. The anode E is an annular mass of metal surrounding the cathode, and is also supported from the cover. Suspended above the cathode is a cylindrical iron vessel C, whose lower edge supports a cylinder of iron or nickel gauze that completely surrounds the cathode and separates it from the anode. The sodium is collected in the vessel C. For full description of the Niagara Falls plant using this process, see this journal 1; 15 (1902).

The advantages of this process are:

(a) Low melting point of the bath.
(b) Simplicity of the apparatus.

The disadvantages are:

(a) High cost of raw material.
(b) Low ampere efficiency.
(c) Small size of the units.
(d) Constant explosions during process.

The two advantageous factors tend to decrease the wear and tear and thus the maintenance charges. Sodium hydroxide (78 per cent Na_2O) costs about $45.00 per ton, while an equivalent quantity of salt is worth about $6.00. As by-products are obtained in the electrolysis of sodium hydroxide, while the chlorine from the electrolysis of salt can be marketed for a greater or less profit according to current conditions. The

ampere efficiency is theoretically only 50 per cent, since for every atom of sodium liberated an atom of hydrogen must also be formed. In practice the efficiency is probably not over 40 per cent. and by some authorities is placed as low as 35 per cent. Another great drawback is the small size of the units. Owing to the constant explosions of oxygen and hydrogen in the cell, it would be dangerous to make the cells of any great size. This tends to lower the efficiency and increase the charges for maintenance and attendance. The metal is removed in dribblets by dipping out with a spoon, so the latter item may be a considerable factor.

For comparison with a process to be discussed later, the following estimate is made of the cost of producing sodium by the Castner process under the following conditions: Daily production, 1 ton of metallic sodium; ampere efficiency, 40 per cent; fall of potential 5 volts per cell; capacity of cells, 1,200 amps; capitalization, $50,000; 360 days per year; 24 hours per day:

Costs Per Year.

650 tons Sodium hydroxide at $45.00	$29,250
725 horse-power years at $20.00	14,500
Thirty men at $2.00 per day per man	21,600
Management and expert labor	9,000
Interest and depreciation 15 per cent	7,500
General expenses and selling, 5 per cent of sale price	9,000
Miscellaneous supplies	1,000
Total	$91,850

Yearly production = 720,000 pounds.

Approximate cost per pound = $0.127.

As the market price of sodium shows no inclination to fall below $0.25 per pound, there seems to be ample margin for a handsome profit.

References:

This Journal 1: 15 (1902).

Zeit. f. El-ch. 8: 412 (1902).

Eng. & Min. Jr. 57: 580 (1894).

U. S. P. No. 452,030 of May 12, 1891. Also patents in Eng., Ger., Fr. and Switz.

The processes used by the Aluminium Industrie Aktengesellschaft and the Chemische Fabrik Griesheim Elektron, of Bitterfield, are somewhat similar, both making use of the surface contact electrode for the cathode. The Bitterfield process was devised by Rathenau and Suter and their apparatus is of very simple in construction. The caustic soda is fused in an iron vessel. The anodes dip into the electrolyte, but the cathodes just touch the surface where the sodium gathers in a globule. These inventors do not consider it necessary to use a diaphragm. The metal has to be gathered in even smaller dribblets than in the Castner process. Ger. P. No. 96,672 of March 15, 1896.

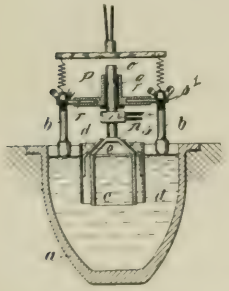


FIG. 6.—BOELSTERLI CELL.

The Aluminium Industrie process was invented by J. Boelsterli. In his apparatus the metal is liberated exclusively at the surface of the electrolyte. The cathodes b (Fig. 6) have rounded ends, and are arranged in a ring around the anode, or cathodes and anodes may be arranged alternately. The anode c dips some distance below the surface of the electrolyte and is surrounded by a sheath d, which retains the oxygen and causes it to pass to the surface within the space which it encloses. This process has the same disadvantage as the Bitterfeld process with regard to the collection of the metal in small amounts, and has the further drawback that the solid shield between the electrodes must greatly increase the required voltage. U. S. P. No. 589,523 of Sept. 7, 1897. Also patents in Ger., Eng. and Switz.

An apparatus described by Dronier may have been intended for the electrolysis of fused sodium hydroxide, but as both electrodes dip deeply into the electrolyte and have no diaphragm between them, it does not seem to be very well adapted to the purpose.

Reference:

Zeit. f. El-ch. 4; 419 (1898).

George P. Scholl, of Philadelphia, Pa., proposed to reduce the voltage required for the preparation of sodium by adding sodium sulphide to the bath. He states that the sulphur liberated at the anode unites with the sodium hydroxide to form sodium sulphide, and thus by adding fresh quantities of the hydroxide as required, it is possible to have the continuous formation of sodium at the low voltage of decomposition of sodium sulphide. The patent specifications say that less hydrogen is liberated than in the Castner process, but it does not state what becomes of the hydrogen in the hydroxide that must be constantly added. The writer is inclined to believe that the sulphide would be oxidized to sulphate at the anode, and that this sulphate would diffuse in the bath and be reduced at the cathode, thus depolarizing at both electrodes and reducing the ampere efficiency. The presence of the sulphate in the bath would also raise the melting point until a point would finally be reached where the temperature would be so high that the preservation of the liberated sodium would be a matter of considerable difficulty. U. S. P. No. 679,997 of Aug. 6, 1901.

To raise the low ampere efficiency of this class of processes, Thomas Ewan, assignor to the Cassel Gold Extracting Co., suggested the use of a porous diaphragm to separate the containing vessel into anode and cathode compartments. The diaphragm is made of alumina or sodium aluminate. The water formed in the anode compartment is removed by passing air through the electrolyte. The difficulty lies in the doubtful durability of the diaphragms in the fused bath. (See Darling process under next class for fuller discussion of this difficulty.) In the "Analysis of Current Patents," this journal, 2; 70 (1904), some comments were made by the writer of the analysis which do not seem quite justified by the facts. The comments were as follows: "He omits, however, in propounding this novel theory of the electrolysis of fused caustic soda, to state where the water, which he claims is formed at the anode, is to come from. Blowing air through or over the caustic melt would result in the formation of a good deal of carbonate with consequent thickening of the melt." The first portion of these objections has been met by the theory of the process as explained in this article, and the second is not of such vital importance. It has been shown by the writer (this journal 2; 358 [1904]) that the presence of about 40 per cent of sodium carbonate does not raise the melting point of sodium hydroxide. This carbonate will not be decomposed by the current, but some time would be required to convert 40 per cent of the bath to carbonate. As the above-mentioned research was not published until after the "Analysis" was written, its writer cannot be held responsible for facts of which he had no means of knowing. U. S. P. No. 745,958 of Dec. 1, 1903. Also patents in Eng., Ger. and Fr.

Electrolysis of Fused Sodium Nitrate. (90 733,374.5)

Some experiments on the electrolysis of fused sodium nitrate were made by Faraday in 1833 and Hittorf in 1847, but their results are of no importance from the technical standpoint.

References:

Faraday, Philosoph. Trans. (1833).

Ostwald's "Klassiker," No. 86: 44.

Hittorf, Pogg. Ann. d. Phys. 72: 481 (1847).

The Darling process seems to be the only attempt on record toward the technical solution of this problem. In many ways

it is a very attractive proposition. The melting point of the nitrate is low; the raw material is not much more expensive than sodium hydroxide, and nitric acid, a valuable by-product, is obtained, a by-product so valuable in fact that in a normally efficient process it would pay the cost of the raw material and the power consumed; the theoretical ampere efficiency is high, and the properties in general would seem to lend themselves to treatment by a fusion process. In spite of these advantages, however, the technical difficulties are even greater than in the electrolysis of fused salt. The electrolyte is very corrosive and is a powerful oxidizing agent, while the metallic sodium is an equally efficient reducing agent. It is, therefore, quite evident that metallic sodium cannot be liberated in a bath of fused sodium nitrate, and that to accomplish it some form of diaphragm must be used.

The Darling cell consisted essentially of a cylindrical iron melting vessel within which was a large cylindrical porous cup of special construction. This cup was made up of two concentric, perforated sheet steel shells, the space between which was filled with magnesia fritted in the electric furnace. Later, a mixture of magnesia and Portland cement was used for the filling. The cathode compartment within the cup contained fused sodium hydroxide. The melting vessel itself acted as anode, and the fused nitrate was contained in the space between the melting vessel and the porous cup. The nitrous fumes formed at the anode were conducted away and absorbed in water, to form nitric acid. In order to protect the metal walls of the diaphragm they were placed in shunt with the anode, thus making them cathodes. In order to prevent the migration of the nitrate into the cathode compartment it was necessary to thicken the bath by adding a little alumina. This cell produced sodium with a fairly good efficiency, but the life of the diaphragm is only 400-450 hours, making very heavy charges for maintenance and attendance. A further disadvantage was that the fall of potential was about 15 volts per cell. Experiments were made on a large scale and large quantities of sodium were produced, but for the reasons given above, it did not prove to be a commercial success. This lack of success seems to have deterred others from making any similar experiments, and the Darling process stands as the one attempt toward the technical solution of this problem.

References:

Jo. Frank. Inst. 153; 65 (1902).
El. W. & Eng. 39; 136 (1902).
Zeit. f. El-ch. 9; 369 (1903).
U. S. P. No. 517,001 of March 20, 1894.
U. S. P. No. 590,826 of Sept. 28, 1897.
U. S. P. No. 641,276 of Jan. 16, 1900.
U. S. P. No. 641,438 of Jan. 16, 1900.
Also patents in Eng., Ger., Fr. and Switz.

Compound Electrolysis, 669.733.374.6.

This class includes processes in which the electrolysis takes place in two steps instead of one, both steps proceeding simultaneously and involving the use of a liquid bipolar electrode, or its equivalent.

The best common example of a technical application of a liquid bipolar electrode is the Castner process for preparing sodium hydroxide. Its chief advantage over the other processes for the preparation of sodium hydroxide lies in the complete isolation of the anode and cathode products from each other. This is the most perfect method of preventing depolarization by the products liberated at the other electrode, and, if it works so well with solutions, why cannot the same principle be applied to fused baths? We have already seen that the direct electrolysis of fused sodium chloride brought forth nothing but failure from the technical standpoint, largely because it proved to be impracticable to liberate both chlorine and sodium in the same bath, even where there were partitions partially separating the two compartments from each other. If, now, a Castner cell is imagined in which fused sodium chloride takes the place of the [illegible] substituted for the mercury and the cathode is fused hydroxide instead of solution, or the [illegible] feasible to produce metallic sodium instead of sodium hydroxide in the cathode compartment.

The germ of this idea is given in the Castner patent (U. S. P. No. 528,322 of Oct. 30, 1894) in which it says: "The same principle may be applied to the electrolysis of fused salts using a molten metal such as lead, tin or the like, or an alloy of various metals in place of mercury." It is possible that the inventor might have developed this idea himself had it not been for his untimely death.

A fused metal alloy as a soluble anode has been made use of in the refining of several metals. The refining of crude lead in a bath of fused lead chloride is described in Blount's "Practical Electrochemistry, p. 92, second edition. The same principle has been used by William Hoopes for the purification of aluminium. U. S. P. No. 673,364 of April 30, 1901.

The first attempt to apply this idea to the preparation of sodium was that of E. A. Ashcroft. His process has been recently described in this journal (4, 220 [1906]), and before the Ithaca meeting of the American Electrochemical Society (Transactions 9, 123 [1906]), so it is not necessary to repeat in full. The author, however, has done considerable work with a furnace of 1,000 amps. capacity, similar to Ashcroft's, and is therefore in a position to add something to the literature of the subject, pointing out from "commercial-scale" experience some of the advantages and disadvantages of this method.

The Ashcroft process, as described in the above-mentioned reference consists in electrolyzing a fused bath of sodium chloride with a fluid lead cathode and carbon anodes. The lead alloy is then circulated through an "equalizer," where it is cooled to about 350° C., into a second compartment, where it acts as a fused soluble anode in a bath of fused sodium hydroxide. The depleted alloy then returns through the equalizer, where it is warmed by cooling the hotter alloy fresh from the anode compartment. The lead is circulated by passing magnetic lines of force at right angles to the flow of the current.

In the apparatus used by the author, the lead was first circulated by pistons (U. S. P. No. 830,051 of Sept. 4, 1906), but later by means of a screw propeller. The alloy was circulated directly from one compartment to the other, therefore, the temperature in the cathode compartment was about as high as in the anode compartment. At that high temperature (about 800°), no sodium was obtained when fused sodium hydroxide was used as the electrolyte, but by using a mixture of the chlorides of sodium, potassium and calcium, sodium was recovered without any very great difficulty, the chief precaution necessary being the protection of the separated metal from the air. The construction of the cathode compartment presented some difficulties, but none which cannot be overcome. The current density at the cathode was about 215 amps. per square decimeter (2,000 amps. per square foot), and at the anode about 270 amps. per square decimeter (2,500 amps. per square foot). If the current density be allowed to exceed 300 amps. per square decimeter (2,800 amps. per square foot), using graphite anodes in a bath of fused chloride at 700-800° C., the rate of corrosion becomes very rapid. At 270 amps. per square decimeter the electrodes were used several days without even rounding the corners.

The cooling and heating of the lead by the regenerative principle, as used in the Ashcroft process, is the source of some loss of heat, but this factor is not sufficiently large to interfere materially in the efficiency of the process. Assuming that 100 pounds of lead must be circulated for every pound of sodium produced, and that the lead returning to the anode compartment has to be heated 100°, the cost of supplying the lost heat by electrical energy is 0.05 cent per pound of sodium. The two great disadvantages are the high voltage required and the excessive wear and tear on the apparatus.

The fall of potential has been given by Ashcroft as 9 volts per furnace, and this figure corresponds with that found by the author in his furnace. To produce 1 kg. sodium by the Castner process at 40 per cent ampere efficiency requires 2,920 amp-hours at 5 volts = 14.6 kw-hours. To produce the same quantity of metal by the Ashcroft process at an ampere efficiency of 90 per cent, requires 1,297 amp-hours at 9 volts = 11.67 kw-hours. Thus while the ampere efficiency of the Ashcroft process is about 125 per cent greater than that of the Castner process, the energy efficiency is but 25 per cent greater. Ashcroft stated in the above reference that the energy efficiency of the two processes is approximately the same, but the author is of the opinion that he overestimated the energy efficiency of the Castner process. The power factor is, however, of much less importance than the cost of the raw materials, and it is in the difference in the cost between sodium hydroxide and salt that the advantage of the Ashcroft process rests.

For high temperature furnaces, both the original cost and the wear and tear are higher than for the simple cells which can be used for the electrolysis of fused sodium hydroxide. The lead used for the cathode costs about $50.00 for every 1,000 amps. capacity, and must be included in the cost of the furnace. While, theoretically, no lead is lost, there is some loss in practice.

Mr. Ashcroft has estimated the cost per pound of sodium by the Castner process at 10.5 to 14.5 cents, and by his own process at 5 to 9 cents, according to the cost of power. From the data determined by the author, a yearly estimate of the costs has been made, under the following conditions: Production of 1 ton per day; ampere efficiency, 90 per cent; fall of potential, 8 volts per furnace; capacity of cells, 5,000 amps.; capitalization, $60,000; 360 days per year; 24 hours per day.

Costs Per Year.

960 tons of salt at $5.00	$4,800
600-hp. years at $20.00	12,000
Thirty men at $2.00 per day per man	21,600
Management and expert labor	9,000
Interest and depreciation at 20 per cent	12,000
General expenses and selling at 5 per cent of sale	9,000
Miscellaneous supplies	1,000
	$69,400

Yearly production 720,000 pounds.

Approximate cost per pound = 9.63 cents.

From the above costs must be deducted any profit that can be realized from the utilization of the chlorine, but as the chlorine from fusion cells is not in as good condition for the production of bleaching powder as that which comes from solution cells, the profit from this source may be quite small. Under similar conditions, this makes an advantage of about 3 cents per pound in favor of the Ashcroft process. Whether the Castner process will be abandoned or not depends upon numerous other factors than those above discussed, but it seems hardly probable that any new plants will be erected using the Castner process even after the Castner patent expires.

References:

This Journal 4; 220 (1904)
U. S. P. No. 788,560 of May 2, 1905
U. S. P. No. 801,199 of Oct. 10, 1905
Also patents in Eng., Ger. and Fr.

Electrolysis of Solutions. 669.733.375

There are two general ways in which metallic sodium can be prepared from sodium compounds in solution; first, by electrolyzing such a solution with a mercury cathode and distilling off the mercury from the amalgam so obtained, and, second, by using a solvent for the sodium compound which is to be electrolyzed, which will not react with sodium.

It is not within the province of this article to describe the numerous apparatus available for the preparation of sodium amalgam, even though every such a one has in it a possible preliminary step in the separation of metallic sodium. Numerous inventors have incidentally suggested the use of the amalgam for the preparation of sodium, but there seem to be only two inventors who have intended that the metal should be isolated continuously in this manner. Baker and Burwell proposed to produce sodium amalgam continuously in the conventional way. The amalgam was simultaneously distilled and the condensed mercury vapor returned to the mercury cell. This process has a high ampere efficiency, so far as the formation of the amalgam goes, but there is one serious if not fatal objection to the process. There are few, if any, mercury cells that produce sodium amalgam containing more than 0.1 per cent of sodium. It would, therefore, seem doubtful whether it would pay to distill off half a ton of mercury in order to obtain 1 pound of sodium.

References:

U. S. P. No. 734,499 of July 28, 1903.
U. S. P. No. 739,139 of Sept. 15, 1903.
U. S. P. No. 782,893 of Feb. 21, 1905.

The commercial prospects of the electrolytic deposition of sodium from organic solvents, or other solvents that will not react with sodium, do not seem very bright. An interesting paper on this subject was presented by Patten and Mott before the Cleveland meeting of the American Chemical Society, June 29 and 30, 1903. Extracts therefrom may be found in this journal 1; 419 (1903).

Sodium Alloys. 669.733.6.

The alloys of sodium have been used more as an intermediate product in the production of sodium hydroxide than in the preparation of metallic sodium, and the preparation of the amalgam especially is more closely related to the alkali industry than to the metallurgy of sodium, but since the preparation of lead-sodium alloy involves electrolysis in the fused state, it falls more naturally with the other fusion processes, and will therefore, be treated here.

It is not possible to say positively who was the pioneer in the preparation of alloys of sodium with the heavy metals, but this honor is claimed by J. Walter, who states that he applied for a German patent on Aug. 24, 1887. The basic claim was on the preparation of sodium alloys by electrolysis using a fluid metal cathode with a constantly changing surface. In the *Zeitschrift fur Elektrochemie* 3: 385 (1897), he gives a resume of the original application, which shows such an advanced knowledge of the art that a full extract is given below:

"Neuerungen in der Herstellung von Metall-Legierungen auf elektolytischem Wege." On Aug. 24, 1887, the author applied for a German patent with the above title. The claim was on the preparation of alloys by electrolysis, using a fluid metal cathode with a constantly changing surface. This was denied, and German patent No. 42,418, cited. This patent is on the use of a moving cathode in electroplating. At the time this article was written, ten years after the above-mentioned application, several processes had been patented which would have been covered by this application. When one attempts to prepare sodium-lead alloy by electrolyzing a fused bath of some sodium salt with a fluid lead cathode, it is found that beyond a certain point the sodium is not absorbed by the lead as fast as liberated by the current, and escapes into the electrolyte. If the bath is allowed to cool it will be found that the top of the lead is very rich in sodium while the bottom is very nearly pure lead. The sodium-lead alloy is so much lighter than the lead that it mixes with difficulty. This is the great difficulty in the technical application of the process. This may be overcome by having the surface of the lead constantly renewed, which may be accomplished in various ways.

1. By mechanical stirring of the metal forming the cathode:
   a. By means of a neutral gas forced through the metal, or some vaporized neutral substance. This gas may be so chosen that it will depolarize the anode.

b. By means of mechanical stirrers, water or air cooled.
c. By turning the container or melting pot.
d. By raising a portion of the fluid cathode with an Archimedes screw or similar contrivance.
e. By an alternate flow of the cathode metal from one container to another.
f. By periodically adding fresh cathode metal.

2. By allowing the cathode metal to flow through the electrolyte by small falls or ridges or the like, the metal at all times, however, being connected with the cathode.

3. By hanging a solid bar of the metal into the melt. If the melting point of the metal is higher than that of the electrolyte and that of the alloy lower than that of the electrolyte, the alloy will drip off as formed, leaving a new surface of metal exposed.

4. By alternate enrichment of the bath near the cathode with one or more constituents in turn.

Examples of the application of this process:

a. To make sodium-lead alloy: Bath, sodium chloride; cathode, melted lead; anode, carbon. Cathode stirred by jet of nitrogen, hydrogen, carbon monoxide, methane, water gas, etc., which may also react with the chlorine and depolarize the anode.

b. Aluminium-zinc alloys: Bath, sodium and aluminium chlorides; cathode, a stream of zinc flowing over carbon.

c. Silicon-copper alloys: Bath, a fused mixture of alkali silicate and chloride. The cathode is a bar of copper. The temperature of the electrolyte is maintained at the melting point of the desired alloy. The alloy drips off the bar and collects in the bottom.

Whether this application was made in the form stated could not be confirmed by any literature available, but the author has assumed that the *Zeitschrift für Elektrochemie* would not have published the same had there not been some foundation in fact. It is a remarkable coincidence, however, that every process ever devised for the preparation of lead-sodium alloys is more or less completely covered by the above specifications. The author at any rate had a very clear perception of the difficulties involved and of the means for overcoming them.

The publication by Walter in 1897 was hardly as useful to the world at large as that of A. J. Rogers in 1889. The aim of his work was to produce sodium alloys to be used in the reduction of aluminium compounds. The "heavy metal" was placed in the bottom of a silicious crucible, which was heated in a wind furnace. By electrolysis of fused salt, using this heavy metal as cathode, he obtained alloys of lead and tin with sodium up to 50 per cent sodium with the tin and to 17 per cent sodium with the lead. In order to obtain pure sodium, these alloys must be distilled. The yield of sodium, as alloy, was 5-6 pounds sodium per horse-power per 24 hours. The crucibles are to some extent attacked by the alloys, but not so rapidly as by the pure alkali metals. A mixed electrolyte, composed of sodium chloride and cryolite, was also used. By applying the rich alloy thus obtained to the reduction of aluminium, the yield of aluminium was 1 pound per horse-power per 24 hours.

References:

Jr. Frank. Inst. 128; 486 (1889).
Proc. Wis. Nat. Hist. Soc. (1891).
Chem. News 60; 228 (1889).

The great year for investigation in this field was 1894, and the several workers of that period will be discussed in alphabetical sequence.

Borchers realized the necessity of keeping the lead cathode in motion, which he accomplished by allowing the fluid metal to flow down over a series of grooves arranged like small terraces, fresh lead being continuously added at the top and the alloy being continuously withdrawn from the bottom. The electrolyte was fused salt. U. S. P. No. 544,153 of Aug. 6, 1895.

The use of fused caustic soda as the electrolyte with a fluid metal cathode was suggested by Hetherington, Hurter and Muspratt. Unless the alloy obtained was to be used as an intermediary product in the preparation of the free metal, there is no use for the alloy thus formed. The author made qualitative tests, using a fluid lead cathode in the electrolysis of melted caustic soda, and found that hydrogen as well as sodium was liberated at the cathode. Eng. P. No. 5,831 of March 20, 1894.

The lead-sodium alloy collects on the surface of the lead, and if allowed to remain will finally reach a degree of concentration at which the sodium will be redissolved as fast as liberated by the current. To overcome this, Hulin proposed to continuously deposit a certain proportion of lead with the sodium. This was accomplished by having a soluble lead anode, as well as insoluble anodes, in a bath of fused sodium and lead chlorides. A portion of the current was allowed to pass through the lead anode, the quantity of lead dissolved being proportional to the number of amperes allowed to pass through. The general construction may be readily understood from the illustration (Fig. 7), and no special description is necessary. In the large experimental plant the following results were obtained: Fall of potential, 7 volts per furnace;

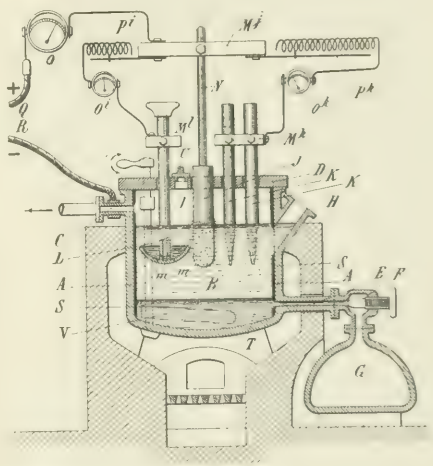


FIG. 7.—HULIN CELL.

current density at cathode, 70 amps. per square decimeter (650 amps. per square foot); yield, 1.24 kg. sodium per horse-power day (23 hours); ampere efficiency, 69.3 per cent; energy efficiency, 41.5 per cent. The low efficiencies said to be compensated for by the saving of evaporation charges and by the value of the by-product, lead peroxide which is obtained by heating the alloy to a low red-heat. The sodium is converted to oxide and then to hydroxide by treating with water.

References:

The Electrician 40; 623 (1898).
Eng. & Min. Jr. 67; 497 (1899).
U. S. P. No. 542,057 of July 2, 1895.
U. S. P. No. 587,830 of Aug. 10, 1897.
Also patents in Eng., Ger., Fr., Belg. and Switz.

The apparatus of Vautin at first consisted of a metal vessel lined with non-conducting material down to the surface of the fluid metal cathode. The alloy was drawn off intermittently. A later modification had a second vessel attached to the alloy-producing cell by means of a pipe. In this second cell the alloy was treated with steam to form caustic soda. No special means were employed for circulating the fluid metal cathode, an omission which was probably fatal to the commercial success of the process.

References:

Industries & Iron 16; 386 (1894).
Zeit. f. El-ch. 1; 250 (1895).
Jr. Soc. Chem. Ind. (1894); 448.
Zeit. f. angw. Chem. (1894); 528.
U. S. P. No. 541,465 of June 25, 1895.
U. S. P. No. 531,235 of Dec. 18, 1894.
Also patents in Eng., Ger. and Switz.

In 1899, Haag sought to obviate the re-solution of the sodium in the fused bath by placing a layer of glass wool, asbestos or similar material on the surface of the fluid metal cathode.

References:

Ger. P. No. 125,004 of April 6, 1899.
Ger. P. No. 125,337 of April 6, 1899.

The largest technical application of lead-sodium alloys is the Acker process for making caustic soda. This consists briefly in the electrolysis of a fused bath of sodium chloride between carbon anodes and a fluid lead cathode. The lead-sodium alloy thus formed is forced into a second compartment by means of a steam jet, which simultaneously converts the sodium to hydroxide. The lead and caustic soda separate from each other by gravity in the second compartment, and the lead returns continuously to the electrolytic compartment. This process has been described so fully in this journal 1; 54 (1902), and in the Transactions of the American Electrochemical Society 1; 165 (1902), that it is unnecessary to repeat in full here, simply noting in passing that the name "Acker" is as imperishably associated with the preparation of lead-sodium alloy as is that of Castner with the direct electrolysis of fused caustic soda.

References:

Zeit. f. El-ch. 9; 364 (1903).
El. W. & Eng. 39; 586 (1902).
Jr. Frank. Inst. 156; 221 (1903).
Jr. Soc. Chem. Ind. 19; 47 (1900).
U. S. P. No. 623,691-92-93 of April 25, 1899.
U. S. P. No. 649,565 of May 15, 1900.
U. S. P. No. 674,691 of May 21, 1901.
U. S. P. No. 687,709 of Dec. 3, 1901.
Also patents in Eng., Ger. Fr. and Switz.

The most recent attempt to utilize sodium-lead alloy in the preparation of caustic soda was that of H. S. Blackmore. For an adequate description of the process and apparatus, the original patent must be consulted. The surface of the lead is protected by a diaphragm composed of granular magnetite. The alloy diffuses to a second vessel with the aid of gravity, said vessel surrounding the electrolytic compartment, where the sodium is converted to caustic soda by the action of fused caustic containing a small quantity of water.

References:

U. S. P. No. 759,799 of May 10, 1904.
U. S. P. No. 809,085 of Jan. 2, 1906.
U. S. P. No. 809,088 of Jan. 2, 1906

Conclusion.

All the attempts to produce sodium commercially have failed except those involving the direct electrolysis of fused caustic soda. The only important use of the lead-sodium alloy is in the production of caustic soda. For further improvement the only hope seems to be in the electrolysis of fused salt, the field where the greatest number of failures have been recorded. It is quite probable that a successful solution of the problem has been found in the Ashcroft compound process. At the present price of sodium it seems probable that its use will still be confined to the preparation of sodium cyanide, sodium peroxide and a few minor applications, but it is to be hoped that the price may be further reduced and new uses found. Perfection has not yet been attained, and the author hopes that this brief and incomplete history may assist future workers in the improvement of the art.

Mixed Distillation and Combustion Gases.

(Translated from Baron Hann's Jüptner von Jonstorff's Chemical Technology of Energies.)

By Oskar Nagel, Ph. D.

If we subject natural uncarbonized fuel in proper apparatus (gas generators, also called gas producers) to incomplete combustion, mixed distillation and combustion gases are formed. In the upper layers of the producer the hydroscopic water is removed. In further going downwards the fuel (material to be gasified) is subjected to dry distillation, coke being the result of this process. The coke is burned incompletely in the lowest part of the producer, whereby, besides the heat necessary for evaporation and dry distillation, also CO is generated. The water which is introduced as moisture with the atmospheric air is also decomposed. A clear idea of these processes is given in the table below, without, however, taking into account the formation of tar, which is inconsiderable.

Composition of the coal used (bituminous coal of Ostran (Moravia) mixed with lignite of Leoben (Styria).

C =	64.92
H_2 =	2.50
N =	0.50
Chemically combined water	14.22
Hygroscopic water	12.42
Ash	5.44
	100.00
Combustible sulphur	0.52
Calorific value	6374 Calories.

(*a*) Process in the upper part of the generator (drying of coal): 100 kg. coal yield 12.42 water (steam), and 87.58 kg. dry coal.

(*b*) Process in the middle part of the generator (dry distillation of coal).

We suppose that the coke contains nothing but carbon, besides the ash, and that the gases of dry distillation contain no oxygen except as CO and H_2O (the latter supposition is not quite but sufficiently correct, since the gases contain CO_2 and other oxygen compounds). The formation of tar is not taken into consideration.

		Yield.						
87.58 *Kg. dry coal contain*		*Coke Kg.*	*Gases of Distillation Kg.*					
			H_2O.	CO.	CH_4.	H_2	NH_3	H_2S.
Ash	4.92	4.92						
C	64.92	58.73		5.07	0.52			
N	0.50						0.50	
S	0.52	0.12						0.40
H_2	4.08	0.035		0.17	3.44	0.11	0.025	
O	12.64	5.08	7.50					
Sum	87.58	68.77	5.715	13.23	0.69	3.44	0.61	0.425

Since only a small amount of N is present, we calculate the entire amount as NH_3 actually, however, but one-fifth of the nitrogen of coal is transformed into NH_3.

(*c*) Process on and just above the grate (incomplete combustion of the coke formed).

The coal analysis shows 5.44 per cent ash, while the table shows only 4.92 per cent, which is explained by oxidation, mainly formation of sulphates from FeS_2. The composition of gas shown in the last table results from the average composition of generator gas and the composition of the gases of distillation, which is given above.

The distribution of heat in the generator is shown in the heat balance, Table I.

It is understood that the composition of generator gas de-

Yields.

Components in Kg.	*Coke.*	*Air.*	*Sum.*	*Losses through Grate Openings*	*Gases.* CO_2.	*CO.*	H_2O.	*N.*
Ash	4.92	.....	4.92	4.92	.....	.....	.....	.....
C	58.73	.....	58.73	15.67	6.57	36.49	.....	.....
N	.....	211.63	211.63	.....	.....	.....	.....	211.63
S	0.12	.....	0.12	0.12	.....	.....	.....	.....
H_2O H_2	.....	0.25	0.25	.....	.....	.....	0.25	.....
H_2O O	.....	64.49	64.49	0.25	17.51	48.65	.....	.....
Sum	63.77	276.37	340.14	20.96	24.08	85.14	0.25	211.63

pends, besides the quality of fuel, on the size of same, height of fuel layer, construction of generator, and also temperature and air pressure during the operation. Table I. was prepared by Richard Akerman.

TABLE I.—HEAT BALANCE.

PRODUCTION OF HEAT AND NON-PRODUCED HEAT.	SINGLE		COMBINED.	
	Cal.	%	Cal.	%
I. Production of heat:				
1. Heat produced in generator by chemical processes	179666 4	26 67		
2. Heat introduced by coal and air on their temperature	3337 9	0 49		
			183004 3	27.16
II. Non-produced heat:				
1. Unburned coal falling through the grate	126613 6	18 79		
2. Heat capacity of generator gases	364028 0	54 05		
			490641 6	72.84
III. Heat losses:				
1. By fuel and ash falling through grate	2316 1	0 34		
2. By heat carried away by the gas produced	28282 0	4 20		
3. Loss by moisture of gas	12346 3	1 83		
4. By decomposition of water	8613 5	1 28		
5. (a) Radiation	94890 7	14 09		
(b) Heat necessary for gasifying coal	36553 5	5 42		
			183004.3	27.16
IV. Non-produced heat: By unburned coal falling through grate			126613 6	18 79
			309617 9	45 95
Heat gained			364028 0	54 05
			673645 9	100 00

A—GENERATOR GAS FROM WOOD OF FIR TREES.

KIND OF FUEL.			Trunks and Roots of Fir Tree.	Brushwood of Fir Tree.	Logwood of Fir Tree.	Sawmill Refuse.
Size	Thickness	mm	20–35	....	35–150	20–200
	Length	mm	500–750	maximum 200	maximum 890	maximum 340
Contents:						
Hygroscopic water, %			12.	16	27	60
Ash, %			0.9	0 6	0 5	0 3
Wood substance, %			87.1	83 4	72 5	39 7
Composition of wood substance:						
C %			53.0	?	51 0	?
H_2 %			7.1	?	6 1	?
O %			39.8	?	29.4	?
N %			0.1	?	?	?
Grate area, square meter, of gen			0 0	0 81	1 72	1.37
Cubic content, cubic meters, of gen			26.7	1 9	24 2	7 4
Consumption of fuel per day:						
Per sq. meter grate area		m^3	....	8 1	23 8	14.4
		kg	....	1654	8891	7909
Per generator		m^3	65 2	6 6	41.0	19.7
		kg	14866	1340	15293	10835
Number of charges per 24 hours			2 8	5 6	4 1	6.6
Length of time of presence of fuel in generator (hours)			8 6	4 3	5 9	3.6
Temperature of gas leaving generator, degrees C			180°	505°	147°	125°
Kg. tar in 24 hours			?	?	444	?
Composition of tar:						
C %			....	....	75 5	.....
H_2			....	....	7 4	.....
O			....	....	16 6	.....
N			....	....	0 5	.....
Volume composition of gases free of moisture and air:						
CO_2			3 8	6 2	6 0	11.3
CO			29 5	26 0	29 8	19.6
C_2H_4			0 6	....	0 3	0.9
CH_4			4 2	5 1	6 9	4.3
H_2			6.4	4.3	6 5	7.4
N_2			55 2	58.4	50.5	56.5

B—GENERATOR GAS FROM PEAT.

Origin. Quantity of Peat.			*Machine Good Peat*	*Air-dry Good Peat*
Intermediate analysis	Hygroscopic water %		....	36.0
	Gases, noncombustible		....	27 0
	Gases, combustible		39.0	24.0
	Fixed carbon %		....	24.0
	Ash %		2.8	5.5
Composition of peat substance	C %		57	61.0
	H_2 %		6.8	6 9
	O %		....	30.6
	N %		....	1.1
Grate area square meters			0.0	1.6
Cubic content, cubic meters	of generator		22.8	21
Daily fuel consumption	Per sq. meter grate area	Cubic meter	....	12.8
		Kg	....	5267
	Per generator	Cubic meter	20.6	20.4
		Kg	6262	8446
Number of charges per 24 hours			1.3	1.1
Length of time for which fuel remains in generator in hours			18.5	21.8
Temperature of gas leaving producer deg. C			80–125°	75–125°
Kg. tar in 24 hours			132	173
Composition of tar	C		79.6	79.8
	H_2		9.3	9.2
	O		11.1	9.6
	N		....	1.4
Composition of gases free of air and water	CO_2 Vol. %		6.6	6.8 – 7.4
	CO		29.6	27.6 –26.2
	C_2H_4		0.7	0.4 – 0.4
	CH_4		4.0	3.75– 3.70
	H_2		5.3	12.3 –13.5
	N_2		53.8	49.15–48.8

C—GENERATOR GAS FROM BITUMINOUS COAL.

Intermediate analysis	Hygroscopic water, %	7.6
	Gases, uncombustible, %	9.1
	Gases, combustible, %	13.6
	Coked coal, %	64.6
	Ash	5.1
Composition of coal substance	C %	70.0
	H_2 %	5.0
	O %	18.7
	N %	1.4
Limestone addition, %		3.4
Residue in ash-pit	Weight in % of coal	12.1
	Composition: C %	40.2
	H_2 %	1.0
	O_2+N_2 %	1.2
	Ash	57.6
Grate area, square meters, of generator		2.0
Cubic content, cubic meters, of generator		4.
Daily consumption of coal per	Cubic meter	1.7
	Sq. m. grate area Kg	1251
	Generator: Cubic meter	3.4
	Kg	2502
Number of charges per 24 hours		1.2
Length of time for which fuel remains in generator		20
Temperature of gas leaving generator, deg. C		300°
Composition of gases free of air and water	CO_2 Vol. %	1.8
	CO	27.3
	C_2H_4	0.4
	CH_4	4.3
	H_2	6.2
	N_2	60.1

D—GENERATOR GAS FROM LIGNITE.

Below are given results with a lignite generator.

Number of generators	3	
Grate area per generator	2.5 square meter.	
Duration of test	12	hours 45 minutes.
Coal charged	3000	Kg. Loiden (Styria) coal.
Composition of coal	C	61.55
	Volatile H_2	4.85
	N	0.23
	H_2O chemically combined	20.00
	H_2O hygroscopic	5.32
	Ash	6.28
	Combustible S	0.57
Calorific value		5440 Cal

Losses through grate				936.7	Kg.
Composition of losses { C				73.97	%
	Ash			26.06	
	I.	II.	III.	IV.	Avg.
Composition of the dry generator gas. { CO_2 Vol. %	5.3	5.4	4.2	4.4	4.64
O_2	0.3	0.8	0.6	0.8	0.65
CO	25.19	25.05	25.39	26.50	25.59
CH_4	0.29	0.15	0.51	0.40	0.38
H_2	10.20	10.65	11.20	11.60	11.11
N_2	58.63	57.95	58.01	56.30	57.63

The quantity gasified per hour and square meter grate area is:

Logwood and sawdust mixed	45–50 Kg.
Sawmill waste	200–330 Kg.
Logwood	370 Kg.
Loose peat (bad quality)	75–120 Kg.
Good fibrous peat	200–250 Kg.
Lignite	40–50 Kg.
Bituminous coal	60–250 Kg.

The Influence of Nickel and Carbon on Iron.

By G. B. Waterhouse.

(*Concluded from page 453.*)

Microscopical.

Transverse sections have been made of the bars in the rolled and other conditions, and also of K, P and Q, which contained respectively 0.41, 0.92 and 1.24 per cent carbon after they had been through the following treatment. They were placed in a Sauveur platinum resistance furnace, which was then luted up. In 10 hours they had reached 800° C., and in nine more hours they were at 1,080° C. The voltage was then lowered, and in 24 hours from the commencement of the operation the furnace was at 680° C. For 22 hours the temperature very slowly fell to 650° C., and then the current was cut off. The steels were removed after another 22 hours, being then at 100° C. The furnace temperature was controlled by a Le Chatelier thermo-electric pyrometer.

Rolled Steels.—These consist of ferrite, pearlite, cementite and temper graphite. K, as may be seen, has a small interlocking structure, of ferrite and pearlite. The pearlite increases in amount with increasing carbon, until with O and P it constitutes the entire field. Q, micrograph 4, with 1.24 per cent carbon, has a matrix of granular pearlite with thin cell walls of cementite, some of which is inside the crystals. The next sample, R, has more cementite, and a coarser structure than Q. S and T are represented by micrograph 6, which shows the globular amorphous temper graphite; the ferrite surrounding it; pearlite, and excess of cementite.

Steels Submitted to Treatment "A."—The specimens all show a coarsening of structure. The lower members consist of ferrite and pearlite. N A, micrograph 10, is almost entirely pearlite, and O and P are altogether composed of it. Q again consists of pearlite and cementite. The upper members of the series R, S and T, when polished, show the temper graphite, as seen in micrographs 12 and 14; and when etched present the rather unique combination of ferrite, pearlite, cementite and temper graphite as seen in micrograph 13.

Steels Submitted to Treatment "B."—In the lower members of the series annealing has brought about a complete separation of the pearlite into ferrite and cementite, as may be seen in micrographs 15, 16 and 17. Sample Q still shows cementite and pearlite. The higher members, when polished, have a great deal of temper graphite, micrograph 19. After etching the matrix is seen to consist of ferrite and free cementite, as in the case of K A or L A.

Special Treatment to Obtain Limpel.—Sample K, micrograph 20. This is peculiar to K A, but the much slower cooling has allowed more complete segregation of the cementite.

Sample P, micrograph 21. Here is seen well laminated pearlite, with an excess of cementite in thin meshes.

Sample Q, micrograph 22, consists of pearlite, with thick cell walls of cementite, some of which is also inside the crystals.

A sample of O in the condition after treatment B was taken for electrolysis, the method being that recommended by Arnold and Read. By means of a resistance in the circuit the current was kept at 1 amp. for 94 hours, when the piece was removed. The carbide was mostly attached to the bar. When removed it appeared as a grey sludge with a silvery sheen. The sample weighed originally 178.265 grams, and lost 31.23 grams by the operation. On analysis the figures given in Table V. were obtained.

Table V.

Analyses of Separated Carbides.

	Carbide from OB.	Carbide from QA.
Carbon per cent	6.52	10.07
Nickel per cent	1.86	2.79
Iron per cent	91.71	71.61
Silicon per cent	0.05	0.41
Total	100.14	84.88

The same treatment was tried on a sample of Q in the condition after treatment A. During the electrolysis there was a continuous liberation of gases from the piece, some of which were undoubtedly hydrocarbons, judging from their odor. The carbide appeared as a dark-colored amorphous powder, and had the analysis given in Table V. The result will be discussed under a later heading.

Pyrometric.

The temperatures in this section were all obtained with a Le Chatelier thermo-couple consisting of wires of platinum and platinum alloyed with 10 per cent iridium. A delicate dead-beat mirror-galvanometer made by Carpentier was used, and the deflections observed by means of a scale and lamp in the usual manner.

For calibration the boiling points of water (100° C.), naphthalene (218° C.), and sulphur (448° C.); and the freezing points of tin (232° C.), zinc (420° C.), aluminium (655° C.), and copper (1,084° C.) were used, plotting temperatures and the galvanometer readings as co-ordinates.

The critical points were determined by taking pieces of the steels 1¾ inches long, and drilling a 5/16-inch hole ⅝ of an inch deep. Into this a fire-clay couple cover just fitted, bringing the thermo-junction in the center of the piece. The arrangement is shown in Fig. 3. The piece was packed in well ignited asbestos in a crucible, and the heatings carried out in a small Fletcher furnace.

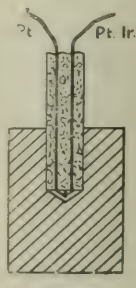


Fig. 3.—Arrangement of recalescence piece.

The results obtained on heating and cooling are given in Table VI.

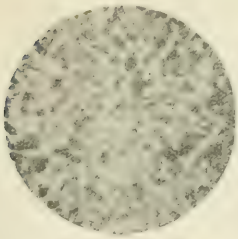
Table VI.

Recalescence Results.

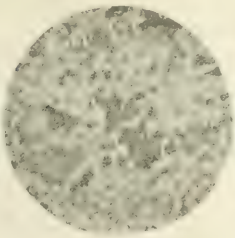
Mark.	C. C. per Cent.	Gr. Per Cent.	Ni. Per Cent.	Heating. Ac2. 3. Degrees C.	Heating. Ac1. Degrees C.	Cooling. Ar3. 2. Degrees C.	Cooling. Ar1. Degrees C.
KA	0.41	....	3.79	744	704–714	653–638	668–598
LA	0.51	....	3.79	734	696–714	640	613–605
MA	0.63	....	3.76	...	702–717	...	612–600
NA	0.79	....	3.81	...	708–718	...	612–602
PA	0.92	....	3.79	...	691–706	...	612–602
QA	0.97	....	3.75	...	701–716	...	612–602
QA	1.24	....	3.81	...	688–698	...	620–610
RA	1.21	0.31	3.82	...	683–693	...	621–611
SA	1.03	0.71	3.82	...	700–710	...	623–613
TA	0.91	0.01	3.79	...	696–711	...	627–606
RB	0.52	1.02	3.82	...	696–711	...	613–605
SB	0.21	1.43	3.82	727	697–712	...	626–606

The heating and cooling curves of the first three members of the series are given in Fig. 4. They, especially that of M, are typical of the curves of the whole series.

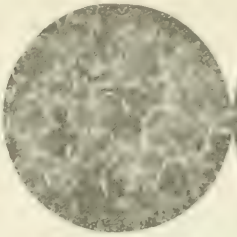
On the advice of Prof. Howe, Fig. 5 was prepared, which



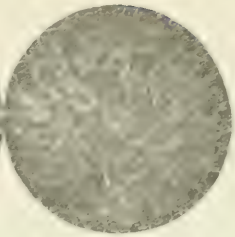
K No. 1. Rolled.
169 diameters.



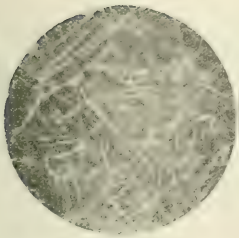
L No. 2. Rolled
169 diameters.



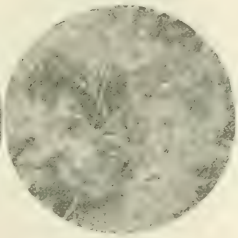
M No. 3. Rolled
169 diameters.



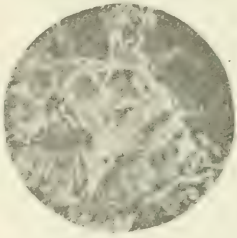
Q No. 4. Rolled.
169 diameters.



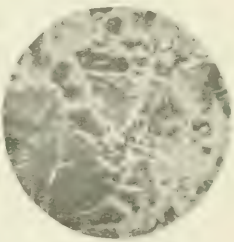
R No. 5. Rolled.
169 diameters.



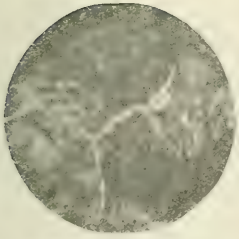
T No. 6. Rolled.
255 diameters.



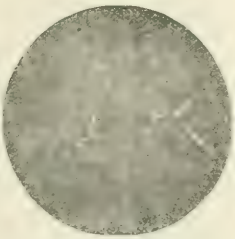
K No. 7. Treatment A
169 diameters.



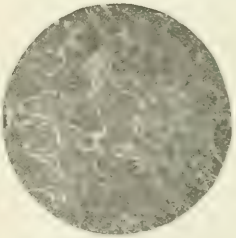
L No. 8. Treatment A
169 diameters.



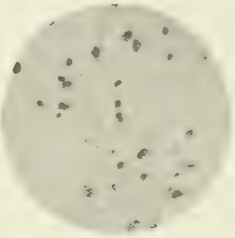
M No. 9. Treatment A
169 diameters.



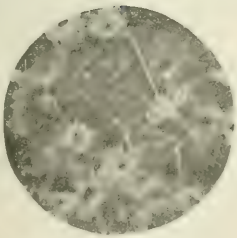
N No. 10. Treatment A.
169 diameters.



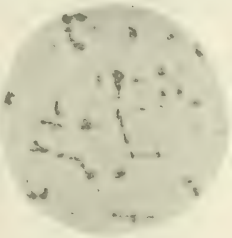
Q No. 11. Treatment A.
255 diameters.



R No. 12. Treatment A
199 diameters. Polished.



R No. 13. Treatment A.
255 diameters.



T No. 14. Treatment A
199 diameters. Polished.

K No. 15. Treatment B.
255 diameters.

M No. 16. Treatment B
255 diameters.

MICROPHOTOGRAPHS

shows the relation of the critical points in cooling of these steels containing about 3.80 per cent nickel, to those of the pure carbon steels of the Roberts-Austen diagram.

Dr. Mathews examined the steels electrically. Comparing the electrical resistance with that of carbon steels of equal purity, it appeared that the addition of about 3.85 per cent nickel raised the specific resistance about 7.5 microhms per cubic centimeters in the steels submitted to treatment "B," for all percentages of combined carbon.

In the hardened state the increase in resistance was greater, and varied from 12 to 17 microhms per cubic centimeter. The electrical resistance of the rolled steels differed but little from those submitted to treatment "B," excepting in the event of

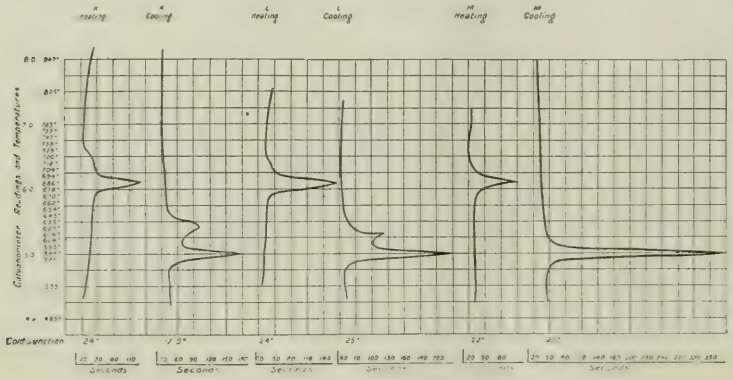


FIG. 4.—HEATING AND COOLING CURVES.

temper graphite being produced, 3.85 per cent nickel was almost without effect in increasing the coercive force of the steels, treatment "A," as compared with similar carbon steels in the same condition, but raised it considerably in the unannealed and hardened states.

Comparing samples K and R after treatment "B," when they contained nearly the same amount of combined carbon, they were found to have nearly the same coercive force and specific resistance, but R showed a lower hysteresis than K, and also lower tensile strength, elongation and reduction of area. The induction of sample K showed it to be the more permeable, but the difference was not great, and the general shape of the hysteresis loops was the same. In the rolled and treatment "B" steels, free from temper graphite, the hysteresis loss in ergs per cycle seemed to reach a maximum at sample Q. Treatment "B" reduced the coercive force in all cases, but the retentivity was often increased. The maximum induction for H-100 in this condition, was higher in some cases than in the rolled steels. This is not surprising when it is recalled that all bars were subjected to the same temperature, which might have been particularly good for some of them and bad for others. For 11-25 to 30 the induction was increased by treatment "B" in every case, excepting sample O, in which it was greatly reduced. The hysteresis loss in ergs per cycle was determined in six of the specimens by Mr. Morecroft as follows:

Sample	Rolled	Submitted to Treatment B.	Reduction.
K	14,660	32,200	27.5%
M	59,660	34,100	42.8%
O	64,660	45,260	30.4%
Q	65,700	47,660	28.5%
R	64,000	25,700	60.0%
T	64,200	21,200	67.0%

Consideration of Results.—Examining the table of the mechanical tests and its accompanying diagram, it is seen that the action of carbon and nickel on iron is very similar to that of carbon alone. In the steels after treatment "A," the tenacity, as shown by the elastic limit and ultimate strength, rises to a maximum with about 1.20 per cent carbon. Above this amount the tenacity drops in part, because some of the carbon is in the graphitic form.

The effect of the nickel is clearly shown by the great tenacity possessed by these steels as compared with carbon steel. To illustrate this the tests given by two similarly treated steels of Prof. Arnold's series may be given.

The elastic ratio, or the ratio of the elastic limit to the ultimate strength, is not greatly raised by the addition of the nickel, and, in the lowest member of the series, is slightly below that obtained from the carbon series. There is no doubt that in commercial nickel steels the elastic ratio is decidedly raised, which may be due to the large amount of manganese that they contain.

Corresponding to the increase in tenacity with the rise in carbon, there is a decrease in the ductility, as shown by the elongation and the reduction of area. At about 1.20 per cent carbon the ductility reaches a minimum, and then rises as the total carbon increases, which increase is accompanied by the separation of graphitic carbon.

The tensile properties are not greatly different from those of the pure carbon steels, and the influence of the nickel has been to increase the tenacity without lowering the ductility to any large extent.

The weakening effect of the separated temper graphite is well shown by the steels QA and RA, the tests of which are given in Table IV. They contain respectively 1.24 and 1.21 per cent of combined carbon.

The influence of treatment B has been very marked. The ultimate strength has been greatly lowered. It steadily rises with the increase in carbon until the latter is about 0.9 per cent, then rises slowly, attaining a maximum at about 1.2 per cent. The separation of the graphite carbon is marked by a drop in the curve.

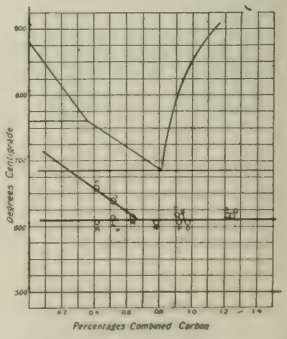


FIG. 5.—COMPARISON OF CRITICAL POINTS WITH THE LOWER PART OF ROBERTS AUSTEN'S DIAGRAM.

The elastic limit of the steels after treatment B is singularly uniform. It was determined in all cases by an extensometer, which, on account of its superior sensitiveness, gave its indications before the drop of the beam, to which, however, it closely corresponded. This interesting point will be referred to later. The ductility of the steel has been increased by the treatment B. It does not greatly decrease after the carbon has reached 0.90 per cent, and shows a rapid increase with the appearance of the graphitic carbon.

In the case of Mr. Hadfield's nickel iron alloys, annealing

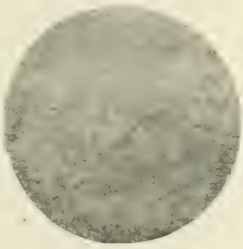
TABLE VII.
Comparison with Carbon Steels.

Carbon.	Nickel.	Elastic Limit.		Ultimate Strength.		Elastic Ratio.	Elongation Per Cent.	Reduction of Area Per Cent.
		Tons.	Lbs.	Tons.	Lbs.			
0.38	....	17.95	40,208	29.94	67,055	60.2	34.5	56.3
0.41	3.79	21.68	48,503	40.28	90,246	53.7	26.0	44.7
1.20	....	35.72	80,012	61.65	138,096	57.5	8.0	7.8
1.24	3.81	45.44	102,030	75.31	168,697	60.3	3.5	3.5

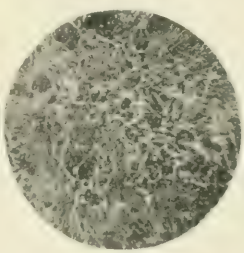
did not have much effect, and he says: "The annealed cast bending bars, forged bending bars, and forged tensile bars show very little difference to the unannealed specimens apparently showing that the annealing is of very little if any benefit." He expresses the wonder whether this would hold good with high carbon nickel steels, a question answered by these results in the negative.

The microscopical section shows that these steels belong to the first class of Osmond, and the pearlitic of Guillet, which is natural, because they lie wholly within the pearlitic area of his diagram. The constituents are those of carbon steels, namely, pearlite with ferrite as the excess substance in hypo-eutectoid or unsaturated, and cementite as the excess in the hyper-eutectoid or supersaturated steels. Graphite is also present. In the case of the rolled steels and those given treatment A, the free ferrite diminishes with increasing carbon, until P and O, containing 0.92 and 0.97 per cent of carbon, respectively, consist entirely of pearlite. Above this percentage cementite is found. In the steels which have been given treatment A the pearlite is of a sorbitic structure. In the higher members of the series the separated carbon has so lowered the percentage of combined carbon that the structure is mainly pearlite. The separated carbon is in small globules in the interior of the pearlite, and is surrounded by a layer of ferrite.

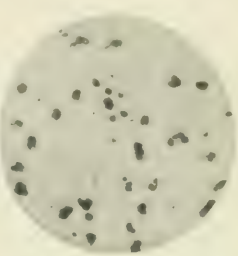
The annealing given in treatment B has had such an effect that the pearlite has separated into its constituents ferrite and cementite, and graphitic carbon is present in the higher members. In sample OB, containing 0.97 per cent carbon, the carbide contains 1.86 per cent nickel, and the ferrite has 1.89 per cent in solid solution. It would appear that the uniform elastic limit of these steels, and the fact that they are all chiefly ferrite and cementite, have a direct bearing on each other. Three specially annealed steels are particularly interesting. The lowest carbon steel has separated into ferrite and cementite; the one with 0.92 per cent carbon has well developed pearlite with meshes of cementite, showing that this composition is above the eutectoid ratio; while the highest member, with 1.24 per cent carbon, has a large excess of cementite.



O No. 17. Treatment B.
276 diameters.



Q No. 18. Treatment B
183 diameters.

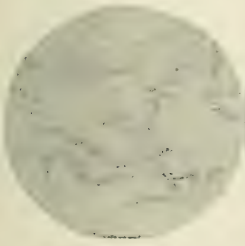


R No. 19. Treatment B
285 diameters. Polished

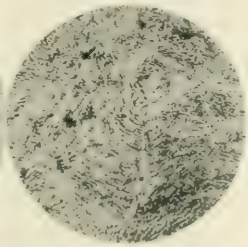
In the steels given treatment B the cementite has the following composition: 0.25 per cent C, 1.87 Ni, 94.51 Fe. By dividing these percentages by the corresponding atomic weights we get 0.54 for C, 0.034 for Ni and 1.63 for Fe. The ratio of the carbon to the nickel and iron is 0.54 : 1.004, which equals 1 : 3.08, so that the formula of the cementite is $Fe(Ni)_3C$, or, writing it simply, $Fe(Ni)_3C$.

In the steels which have been given treatment A the finely divided, almost emulsified, cementite of the sorbitic pearlite suffers decomposition under the method employed for its separation, a moderate amount of hydrocarbons being given off, and free carbon also being produced. This makes it impossible to say what the analysis of the carbide is in the rolled steels and those given treatment A.

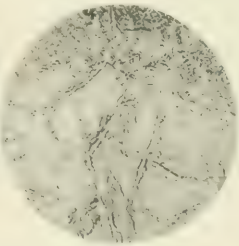
The results of the recalescence experiments show that the Ar_1 point has been lowered 20° C. for each 1 per cent of nickel; and that the eutectoid ratio has been lowered by the nickel to about 0.70 per cent carbon. It is only in the steels that have been given the special treatment in the electric furnace that the eutectoid ratio appears to be about 0.70 per cent. In the rolled steels, and those subjected to treatment A, it appears to be about 0.95 per cent, while treatment B was so drastic that the pearlite was broken up into its constituents, and time allowed for them to segregate.



K No. 20. 215 diameters.



P No. 21. 374 diameters.
TREATMENT IN ELECTRIC FURNACE.



Q No. 22. 215 diameters.

CONCLUSIONS.

The following conclusions may be drawn from the results:

1. Nickel decidedly raises the tenacity without materially lowering the ductility. The elastic ratio, in pure nickel-carbon steels, is only slightly greater than that of carbon steels.

2. Annealing has a marked influence. It lowers the tenacity without greatly raising the ductility.

3. The constituents of steels with low percentage of nickel in the unquenched state are: ferrite, pearlite, cementite and graphitic carbon.

4. The pearlite of these steels shows a great readiness to segregate into its constituents: ferrite and cementite.

5. In this condition the cementite has the formula $Fe(Ni)_3C$.

6. The eutectoid ratio in these steels appears to lie at about 0.70 per cent carbon, but in the rolled steels no free cementite shows until the carbon reaches about 0.95 per cent.

7. Nickel lowers the transformation points Ar3, Ar2 and Ar1 about 20° for every 1 per cent of nickel.

8. The cementite of these steels is very liable to precipitate its carbon as "temper graphite."

Metallurgical Calculations.

By J. W. Richards, Ph. D.

Professor of Metallurgy in Lehigh University.

THERMO-CHEMISTRY OF THE BESSEMER PROCESS.

The feature of the Bessemer operation which strikes the observer as most wonderful, is that cold air is blown in great quantity through melted pig iron, and yet the iron is hotter at the end than at the beginning. If the observer will reflect a moment, however, he can see that if nothing but fuel, on fire, was in the converter, it would certainly be made much hotter by the air blast; in similar manner, the oxidation or combustion of part of the ingredients of the pig iron furnishes all the heat required for the process. Ten tons of pig iron contains, for example, some 350 kilograms of carbon, which is all burnt out in the bessemerizing, furnishing heat equal to the combustion of some 400 kilograms of coke—a not insignificant quantity, since it is burned and its heating power utilized in a very few minutes.

Elements Consumed.

The usual ingredients of pig iron are:

Iron	90	to	95	per cent
Carbon	2.5	"	4.5	"
Manganese	0.5	"	4.0	"
Silicon	0.5	"	4.0	"
Phosphorus	0.01	"	4.0	"
Sulphur	0.01	"	0.5	"

Some of the unusual constituents are nickel, chromium, titanium, aluminium, vanadium, tungsten, copper and zinc; all of these are rare, and there is seldom present as much as 0.5 per cent of any one, except in unusual cases.

In the Bessemer operation, carried out with the usual silica lining, iron, carbon, manganese and silicon are freely oxidized, but phosphorus and sulphur remain practically unchanged. In the basic Bessemer, lined with burnt dolomite and tar, phosphorus is also freely oxidized at the end of the operation, but sulphur is only slightly diminished—the more, the more manganese is in the slag. After all the oxidizable impurities are eliminated, iron itself oxidizes in much larger quantity, occasioning great loss if the blast is permitted to continue.

Iron oxidizes during the blow mostly to FeO, which enters the slag as ferrous silicate and partly to Fe^2O^3. The brown fume which escapes in large amount if the blow is continued too long, contains iron as FeO.

Fe to FeO	1173	Calories per kg. of Fe		
Fe to FeO	1746	"	"	"
Fe to Fe^2O^3	1612	"	"	"

The amount of iron oxidized can be gotten from the weight and percentage composition of the slag, also from the comparison of the weights of materials used and weight of ingots produced, knowing the weight of other impurities oxidized.

Carbon oxidizes mostly to CO gas, and partly, especially in the early part of the blow, to CO^2 gas. The proportion of each of these formed can only be known by analyzing the gases produced at various stages of the blow. The proportionate volumes of CO and CO^2 express the proportionate amounts of carbon burning each to produce gas. A shallow bath allows more CO to pass, on essentially the same principle that a deep layer of fuel on a grate favors the production of CO. The heat evolved by oxidation of carbon is:

C to CO	2430	Calories per kg. of C		
C to CO^2	8100	"	"	"

Manganese oxidizes quickly and mostly to MnO. If the metal is a little overblown, Mn^2O^3 in small amount is found in the slag, while Mn^2O^3 is also present in the fumes. The heat evolved in these oxidations is:

Mn to MnO	1653	Calories per kg. of Mn		
Mn to Mn^2O^3	2300(?)	"	"	"

The last figure given is estimated; it has not yet been determined experimentally.

Silicon oxidizes rapidly and early in the blow to SiO^2, forming silicate slag with the metallic oxides formed. Its heat of oxidation has been usually taken as 7830 Calories per kilogram, but recent investigations have thrown doubt on this figure, Berthelot having found as low as 6414 Calories, and an American experimenter (research not completed; results not yet published) about 7000. Under these circumstances the best course is probably to use the middle value *ad interim*, and consider

Si to SiO^2..... 7000 Calories per kg. of Si

We hope that the exact figure will soon be determined.

Phosphorus oxidizes to P^2O^5, and only towards the end of the blow. It is practically completely eliminated, going into the slag as calcium phosphate:

P to P^2O^5..... 5892 Calories per kg. of P

Sulphur is not eliminated at all in the acid-lined converter. In the basic converter it is reduced in amount in the last few minutes, while phosphorus is disappearing, and partly escapes, mostly as SO^2 in the gases. The presence of a very basic slag is necessary, but the sulphur, while possibly going into the slag, does not remain there, but passes into the gases. The heat of oxidation of the unusual elements sometimes present are, as far as known,

Ni to NiO	1051	Calories per kg. of Ni		
Ti to TiO^2	5000(?)	"	"	Ti
Al to Al^2O^3	7272	"	"	Al
Zn to ZnO	1305	"	"	Zn

The figure for titanium is estimated, and those for chromium, vanadium and tungsten are also unknown, although, as a guess and first approximation, might be used:

V to V^2O^5	2000(?)	Calories per kg. of V		
W to WO^3	1000(?)	"	"	W
Cr to Cr^2O^3	3000(?)	"	"	Cr

Heat Balance Sheet.

Taking 0° C. (32° F.) as a base line, we may express the total heat contents of the pig iron, steel, gases, slag, blast, etc., from this temperature. This method is simpler than to take the bath at any one high temperature and to reckon from there.

The items of heat available during the blow are:

- Heat in the body of converter at starting.
- Heat in the melted pig iron used.
- Heat in the spiegeleisen or ferro-manganese added.
- Heat in hot lime added (sometimes in basic process).
- Heat in the blast, if warm on entering.
- Heat developed by oxidation of the bath.
- Heat developed by formation of the slag.

The items of heat distribution are:

- Heat in the body of the converter at finishing.
- Heat in the steel poured.
- Heat in the slag at finishing.
- Heat in the gases escaping.
- Heat in the fume.
- Heat in the slag or metal blown out.
- Heat absorbed in decomposing moisture of the blast.
- Heat to separate the constituents of the bath.
- Heat conducted away by supports, blast pipe, etc.

Heat conducted to the air in contact with converter.
Heat radiated during the blow.

These two columns should balance each other if all the items of each are correctly determined.

Heat in Converter Body at Starting.

If the converter were quite cold when the pig iron was run in and the blow started, this item would be zero. But such a case would result disastrously, since the heat absorbed by the converter during the blow would be more than any ordinary heat could afford to lose. It is, therefore, customary, when starting for the first time, to build a fire in the converter and turn on a little air blast, so as to bring the inside up to bright redness, say 900° to 1000° C. The outside shell would, under these conditions, be at about 200°, and the mean temperature of the converter lining, say 400°. If the converter were in regular operation, one charge being introduced as soon as the other was finished, then the heat in the body of the converter at starting could be practically regarded as equal to the heat in the same at finishing, assuming the heats are running regularly. An estimate of the heat in the converter body at starting is therefore only necessary when the converter is first started up, or when it is allowed to stand some time between blows.

Illustration: Assume a converter weighing, without charge, 25 tons, of which 5 tons is iron work and 20 tons silica lining. The mean specific heat of iron (for low temperatures) being $0.11012 + 0.000025t + 0.0000000547t^2$, and for silica $0.1833 + 0.000077t$, calculate the heat contained in the body of the converter.

(1) When the temperature of the outside shell is 200° and that of the lining averages 400° (converter warmed up for starting).

(2) When the temperature of the outside shell is 300° and that of the lining averages 725° (converter empty at end of a blow).

Solution: (1)

Heat in 5000 kg. of iron work:

$0.11012 + 0.000025\ (200) + 0.0000000547\ (200)^2 = 0.11731$
$0.11731 \times 200 \times 5000 = 117,310$ Calories.

Heat in 20,000 kg. of silica lining:

$0.1833 + 0.000077\ (400) = 0.2141$
$0.2141 \times 400 \times 20,000 = 1,712,800$ Calories.
Total heat in converter body = 1,830,110 "

It may be remarked that this would require the consumption of at least 250 kilograms of coke, with a calorific power of 8,000 Calories, to warm the converter to this extent.

(2) Heat in 5,000 kg. of iron work:

$0.12354 \times 300 \times 5,000 = 185,310$ Calories.

Heat in 20,000 kg. of silica lining:

$0.2391 \times 725 \times 20,000 = 3,466,950$ "

Total = 3,652,260 "

This condition is assumed as representing the converter when just emptied and immediately refilled. In this case, in regular working, the heat in the converter body is practically the same at the beginning and at the end of a blow, and the heat losses through it are only those due to conduction to the air and ground and radiation.

Heat in Melted Pig Iron Used.

This quantity depends on the temperature at which the pig iron is run into the converter. If the iron is high in silicon, which would tend to produce a hot blow, it may be run in somewhat cool; but if low in silicon it should be run in much hotter. The minimum quantity of heat contained in a kilogram of melted pig iron may be put at 250 Calories; the maximum, for very hot pig iron, 350 Calories; about 300 Calories would be a usual average figure. This may be easily determined experimentally in any given case by granulating a sample in water in a rough calorimeter.

Heat in Metallic Additions.

Melted spiegeleisen is usually not very hot when run into the converter. It may contain 250 to 300 Calories per kilogram; say an average of 275. Ferro-manganese is sometimes added red hot, at 800° to 900° C. At this temperature it would contain 120 to 135 Calories per kilogram. Assuming 4 per cent. manganese, heat of 0.15.

Heat in Produced Lime.

Taking the mean specific heat of CaO as $0.1715 + 0.00007t$, it would contain 154 Calories per kilogram at 700°, and 211 Calories at 900°. The heat content can be calculated for any known temperature at which the lime is used.

Heat in Hot Blast.

No Bessemer converters are run by hot blast, but the air pressure used is so great (20 to 35 pounds per square inch) that the blast is warmed by compression in the converter. If air at 1 atmosphere tension (ordinary air) be compressed to 2 or to 3 atmospheres tension, giving effective blast pressures of 1 to 2 atmospheres, the air is heated 60° or 103°, respectively, above its initial temperature. While some of this heat may be lost in the cylinder and conduits, yet the air, unless artificially cooled, passes to the tuyeres at 25° to 50° C. above the outside temperature, and thus imparts some heat to the converter.

Heat Developed by Oxidation.

We have already discussed the thermochemical data required for this calculation. To use the data we must find the weights of each ingredient oxidized (not forgetting the iron itself) and the nature of the oxide it forms. This is deduced from the analysis of slag, gases and steel produced, as compared with those of the pig iron and additions used.

Heat of Formation of Slag.

The slag is a mechanical mixture or mutual solution of silicates of iron and manganese, with a little alumina (up to 5 per cent) and lime and magnesia from nothing up to 5 per cent. In the basic process a large amount of calcium phosphate is present, representing over half of the entire slag, while magnesia is present in considerable amount, and alumina is almost absent.

Having, in the preceding column, calculated the heat of oxidation of all the metals oxidized in the converter, it remains to calculate the heat of combination of these to form slag. The silicate of alumina contributes nothing, since it occurred combined in the lining or lime added. The same can be said of the lime and magnesia in the acid process slags. The FeO and MnO are then to be considered, and the only thermochemical data we have are:

$(MnO, SiO^2) = 5,400$ Calories.
$(FeO, SiO^2) = 8,900$ "

These data are for 71 or 72 parts of MnO or FeO respectively, uniting with 60 parts of SiO^2. Since in acid slags there is always proportionately less of the bases, we should reduce the above heats of formation by expressing them per unit weight of MnO or FeO going into combination. These figures are:

Per kg. of MnO = 5,400 ÷ 71 = 76 Calories
Per kg. of FeO = 8,900 ÷ 72 = 124 "

From these figures the heat of formation of the slag can be calculated. Any FeO present in the slag would be calculated as its equivalent weight of FeO (by multiplying by 124 ÷ 76).

Illustration: A Bessemer slag contained by analysis:

SiO^2	47.25 per cent.
MnO	3.45 "
FeO	15.43 "
MnO	31.80 "
CaO, MgO	1.84 "

What is its heat of formation per kilogram?

Solution: The MnO, CaO and MgO are to be neglected,

for reasons already given. The heat of combination, per kilogram of slag, is therefore,

FeO uniting with SiO^2 = 0.1543 × 124 = 19.1 Cal.
MnO uniting with SiO^2 = 0.3189 × 76 = 23.2 "

Total = 42.3 "

In basic Bessemer slags the conditions are much more complicated. The content of P^2O^5 is not under 14 per cent, runs as high as 25 per cent, and averages 19 per cent; the CaO averages 45 per cent, limits 35 to 55; the silica is usually below 12 per cent, and averages 6 to 8 per cent; magnesia is present from 1 to 7 per cent, average about 4 per cent. In such slags we should first of all assume the P^2O^5 to be combined with CaO as $3CaO.P^2O^5$, containing 168 of CaO to 142 of P^2O^5 (1.183 to 1), and having a heat of formation from CaO and P^2O^5 of 159,400 Calories per molecule, or 1123 Calories per unit weight of P^2O^5. Next the iron and manganese present may be calculated to FeO and MnO respectively, and treated as to their combination with SiO^2, the same as in an acid slag. Alumina may be considered in such basic slags as an acid, and equivalent to 120/102 of its weight of silica. These allowances will leave considerable lime and either an excess or deficiency of silica; if an excess, we can assume it combined with lime and magnesia, with a heat evolution equal to 476 Calories per kilogram of silica; if a deficiency, we let the calculations stand without further modification.

Illustration: Slag made at a Rhenish works contained:

SiO^2	7.73 per cent.
P^2O^5	21.90 "
Al^2O^3	3.72 "
Fe^2O^3	1.00 "
FeO	4.73 "
MnO	2.05 "
CaO	50.76 "
MgO	4.00 "
CaS	1.71 "

What is its heat of formation per unit of slag?

Solution: The 21.90 parts of P^2O^5 would be combined with 21.90 × 168/142 = 25.91 parts of CaO. This leaves 50.76 − 25.91 = 24.85 of CaO as either free, dissolved CaO or partly combined, in the slag. The 2.05 MnO would correspond to 2.05 × 60/71 = 1.73 SiO^2; the 4.73 FeO to 4.73 × 60/72 = 3.94 SiO^2; the 1.00 Fe^2O^3 to 1.00 × 60/80 = 0.75 SiO^2; a total SiO^2 requirement for these three bases of 6.42 per cent. The SiO^2 present is 7.73 per cent, adding to which the SiO^2 equivalent of the Al^2O^3 present (3.72 × 120/102 = 4.38), we have 12.11 per cent of summated silica. The ratio of summated FeO to summated SiO^2 is considerably below the ratio 72 to 60, we can, therefore, consider the summated FeO as all combined with silica. The summated FeO is:

FeO	= 4.73 per cent.
FeO equivalent of MnO	= 2.08 "
FeO equivalent of Fe^2O^3	= 0.90 "
Total	= 7.71 "

And the SiO^2 combining with this as FeO. SiO^2 is

7.71 × 60/72 = 6.42 per cent

The excess of summated silica free to combine with lime is 12.11 − 6.42 = 5.69 per cent. As there is 24.85 of CaO and 4.00 of MgO for it to combine with, the heat of this combination must be calculated on the SiO^2 going into this combination.

We then have the formation heat of the slag as

P^2O^5 to $3CaO.P^2O^5$	21.90 × 1123	= 24,594 Cal.
MnO to MnO.SiO^2	2.05 × 76	= 156 "
FeO to FeO.SiO^2	5.63 × 124	= 698 "
SiO to $3CaO.SiO^2$	5.69 × 476	= 2,708 "
		28,156 "

This equals 281.6 Calories per unit weight of slag, forming a very important item in the heat balance sheet, particularly when the slag is large in amount.

Heat in Converter Body at Finishing.

This item reaches its maximum at the end of the blow, and would be equal to that calculated for the begining of the blow, with the exception that some of the lining, silica or dolomite has been corroded and passed into the slag, carrying with it its sensible heat.

Heat in Finished Steel.

This should be determined experimentally in each particular case. If not so determined an average value may be assumed, based on the following considerations: The finishing temperature averages 1650° C.; at this heat average Bessemer steel will contain a total of 350 Calories of heat per kilogram. If the temperature is determined by a pyrometer, a correction of 1/5 Calorie can be made for every degree hotter or colder than 1650°.

Heat in Slag.

Some experimental data are badly needed, concerning the heat in slags of different composition at different temperatures. At present it is necessary to make guesses, wherever the heat in the slag produced is not directly determined. At a finishing temperature of 1650° it is likely that the slag contains 550 Calories per kilogram; with a variation of 1/4 Calorie for each degree hotter or colder than 1650°.

Heat in Escaping Gases.

The amount of these gases can only be determined satisfactorily from their analysis and the known weights of carbon oxidized. Direct estimation from the piston displacement is of much less exactness, because the slip and leakage at these high pressures may reach 25 to 50 per cent, or even more. The temperature of the gases is only slightly less than that of the bath, that is, some 1350° at starting and 1650° at finishing. Outside in the air the Bessemer flame may be much hotter than this, but that is due to further combustion of CO to CO^2 outside the converter, and should be disregarded. Where extra air is blown upon the surface of the bath, as in "baby" converters, it is quite possible that the gases in the converter and the escaping gases may be considerably hotter than the bath itself. These variations must be taken into consideration. The most satisfactory condition is to insert a pyrometer tube into the opening of the converter and measure the temperature directly. The heat carried out by the gases can then be calculated accurately, using the proper mean specific heats to these high temperatures already used in these calculations.

Heat in Escaping Fume.

This is mostly oxides of iron and manganese, with sometimes silica. It is relatively small in amount. Its quantity being known, consider it at the same temperature as the gases, with a mean specific heat of 0.40 if free from silica, and 0.35 if siliceous.

Heat in Slag or Metal Blown Out.

These can be counted as equal to the heat in an equal quantity of slag or metal at the finishing temperature.

Heat Absorbed in Decomposing Moisture.

Knowing the amount of moisture blown in with the blast, a proper allowance is 29,040 × 0 = 3,227 Calories for every kilogram of moisture thus blown in. It is probable that this moisture is all decomposed, its hydrogen appearing in the gases. In the absence of data as to the hygrometric condition of the blast, the amount of moisture entering may be inferred and calculated from the amount of hydrogen in the gases.

Heat to Separate Constituents of Bath.

We here meet the question, how much heat of combination exists between the bath and the various ingredients which are removed—carbon, silicon, manganese, phosphorus, sulphur. Le Chatelier believes manganese to exist in the bath as Mn^3C, requiring 80 Calories per kilogram of manganese to decompose

it. Silicon and carbon have, as far as has at present been determined, no sensible heat of combination with iron. Sulphur requires 750 Calories to separate each kilogram from iron. Phosphorus requires, according to Ponthiere, 1397 Calories to separate each kilogram from iron, but the reliability of this datum is doubtful. Until more reliable tests are made it is perhaps better to omit this item than to use it, although it must be of great importance if as large as Ponthiere states it to be.

Heat Conducted Away by Supports.

This is a very difficult quantity to determine, being conditioned by the size of the supports, their cooling surface and the kind of connection they have with other objects. The heat which would pass to the blast pipe is practically returned to the converter by the incoming blast. The heat passing into the supports is perhaps best found by taking their temperature at different places, and calculating the heat loss from their surface by radiation and conduction to the air. This amounts practically to considering them as part of the outer cooling surface of the converter; the calculation of these surface losses is given in the two following paragraphs:

Heat Conducted to the Air.

This is a function of the extent of outside surface, its temperature, the temperature of the air, and the velocity of the air current. Measurement will give the extent of surface in contact with the air and the average velocity of the air current; the surface being rough iron, the coefficient of transfer conductivity may be taken as $k = 0.000028\ (2 + \sqrt{v})$, where v is the air velocity in c. m. per second, and k is the heat conducted per second in gram-calories, from each square c. m. of surface per 1° difference of temperature. The temperature of the outer surface should be carefully measured, so that a reliable average is obtained, and the air velocity likewise averaged, since it has considerable influence on the heat lost to the air.

Illustration: A converter has an outside surface of 50 square meters, at an average temperature during the blow of 200° C., the average air current being 1 meter per second, and outside air 30° C. What is the heat loss by conduction to the air in kilogram Calories per minute?

Solution:

Coefficient of transfer conductivity:

$$0.000028\ (2 + \sqrt{100}) = 0.000336.$$

Heat loss per 1° difference, per second, in gram-calories:

$$50 \times 10{,}000 \times 0.000336 = 168 \text{ calories.}$$

Heat loss per 170° difference, per minute, in kg.-Calories:

$$168 \times 170 \times 60 \div 1000 = \mathbf{1714} \text{ Calories.}$$

Heat Radiated During the Blow.

This is a function of the temperature of the outside shell, the mean temperature of the surroundings of the converter and the nature of the metallic surface. As the surface is oxidized iron, it would lose about 0.0141 gram-calories from each square centimeter per second, if at a temperature of 100° and the surroundings at 0°; or practically 1 gram-calorie per second from each square meter, for every 100,000,000 of numerical difference between the fourth powers of the absolute temperature of the radiating surface and its surroundings. (See Metallurgical Calculations, Part I., p. 185.)

Illustration: Assuming the surroundings of the converter at 30°, in the preceding illustration, what amount of heat is radiated per minute in large Calories?

Solution: The absolute temperatures in question are $273 + 30 = 303$, and $273 + 200 = 473$. The difference of their fourth powers is:

$$473^4 - 303^4 = 41{,}626{,}500{,}000,$$

which, divided by 100,000,000 gives 416.265 gram-calories lost per second per each square meter of radiating surface. The radiation loss for the whole surface per minute is, therefore:

$$416.265 \times 50 \times 60 \div 1000 = 1249.7 \text{ Calories.}$$

While the assumption made as to the temperature of the outside shell is doubtless only approximately correct, the temperature of the same is carefully determined the radiation loss can be accurately calculated. If the outside of the converter were polished, this radiation loss might be reduced nearly nine-tenths.

Problem 65.

From the data and results of calculation of Problem 62 (this journal, November, 1906,) we see that 22,500 pounds of pig iron and 2,500 pounds of spiegeleisen produced 24,665 pounds of steel, there being eliminated during the blow and recarburization:

Carbon	679.5 pounds	(140.7 to CO^2)
Silicon	203.2	"
Manganese	197.9	"
Iron	253.9	"

The gases contain:

CO^2	5.20	per cent.
CO	19.91	"
H^2	1.39	"
N^2	73.50	"

The slag contains:

SiO^2	63.50	"
Al^2O^3	3.01	"
FeO	21.39	"
Fe^2O^3	2.63	"
MnO	8.88	"
CaO	0.90	"
MgO	0.36	"

Make average assumptions for requisite data not given.

Required: A balance sheet of heat evolved and distributed.

Solution: The items of this balance sheet have already been discussed in detail. We will apply them to this specific case:

Heat in Body of Converter at Starting: Assuming that this is a blow in regular running, the heat may be taken at any reasonably approximate quantity, because the same quantity, with only a slight deduction, will be allowed as contained in the same on finishing. We will, therefore, take a figure already calculated, 8,034,970-pound Calories, as the heat in the converter body at starting.

Heat in Melted Pig Iron: We will take it at 300 Calories per pound, or a total of $300 \times 22{,}500 = 6{,}750{,}000$ Calories.

Heat in Spiegeleisen: $2{,}500 \times 300 = 750{,}000$ Calories.

Heat in Blast: This may safely be considered as warmed by compression and entering the converter at 60° C. The amount of blast received altogether is calculated thus:

Carbon oxidized = 679.5 pounds.

Volume of CO and CO^2 formed:

$$679.5 \times 16 \div 0.54 = 20{,}133 \text{ cu. ft.}$$

Volume of gases $= \frac{20{,}133}{0.2511} = 80{,}179$ "

Volume of nitrogen $80{,}179 \times 0.735 = 58{,}932$ "

Volume of air proper in blast = 74,408 "

Volume of hydrogen in gases:

$$80{,}179 \times 0.0139 = 1{,}115 \text{ "}$$

Volume of moisture in gases = 1,115 "

Assuming the blast at 60° C., the heat [illegible]

Air $74{,}408 \times 0.3040 = 22{,}615$ [illegible]

H^2O $1{,}115 \times 0.3799 = 423$ " "

$23{,}088$

$23{,}088 \times 60 = 1{,}385{,}280$ [illegible]

$= 86{,}580$ lb. Cal.

Heat of Oxidation:

C to CO^2	140.7 × 8100 =	1,139,070	Calories.
C to CO	538.8 × 2430 =	1,309,280	"
Si to SiO^2	203.2 × 7000 =	1,422,400	"
Mn to MnO	197.9 × 1653 =	327,130	"
Fe to FeO	2280 × 1173 =	208,150	"
Fe to Fe^2O^3	25.3 × 1740 =	44,170	"
		4,510,800	"

Heat of Formation of Slag: The 197.9 pounds of manganese oxidized forms 255.5 of MnO. Hence the weight of the slag is 255.5 ÷ 0.0888 = 2877 pounds. The slag, therefore, contains also 2877 × 0.6356 = 1829 pounds of SiO^2, of which 203.2 × 60/28 = 435 pounds came from the silicon oxidized, and 1794 pounds from the lining. The lining will also have lost the Al^2O^3, CaO and MgO in the slag, equal to 2877 × 0.0427 = 123 pounds. The total iron going into the slag, 253.9 pounds, is equivalent to 326.4 pounds of FeO.

The heat of formation of the slag will therefore be:

FeO	326.4 × 124 =	40,474	Calories.
MnO	255.5 × 76 =	19,418	"
Total		59,892	"

Heat in Converter at Finishing: This will be the same as at starting, less the heat in 1794 − 123 = 1917 pounds of lining, which was corroded and entered the slag. Assuming this to have been on the inner surface, at an average temperature of 1500°, the heat in it, using the mean specific heat of silica, will have been

1917 × 0.2988 × 1500 = 851,200 Calories.

And the heat in the converter body at the finish:

8,034,970 — 851,200 = 7,183,770-pound Calories.

Heat in Finished Steel: Taking its temperature as 1650°, with 350 Calories per unit, we have

24,665 × 350 = 8,632,750 Calories.

Heat in the Slag:

2877 × 550 = 1,582,350 "

Heat in Escaping Gases: These have already been calculated as consisting of

Nitrogen	58,932	cubic feet.
Hydrogen	1,115	" "
Carbon monoxide	15,964	" "
Carbon dioxide	4,169	" "

The first three have the same heat capacity per cubic foot, so assuming their temperature 1550°.

N^2, H^2, CO	76,011 × 534.5 =	40,627,000	oz. Cal
CO	4,169 × 947.1 =	3,948,250	"
		44,576,150	"
	=	2,786,000	lb. Cal.

Absorbed in Decomposing Moisture: The 1,115 cubic feet of hydrogen in the gases represent so much steam or water vapor decomposed. Since 1 cubic foot = 0.09 ounces, the heat absorbed is

1,115 × 0.09 × 29,090 = 2,914,160 oz. Cal
= 182,130 lb. Cal.

Heat Conducted to the Air: Assuming the conditions worked out in the silica tables under this heading, this item would be approximately, for 9 min., 10 secs., in lb. Cal.

1,744 × 2.294 × 9.167 = 34,630 lb. Cal.

Heat Lost by Radiation: Making similar assumption, we have

1,249 × 2.294 × 9.167 = 25,240 lb. Cal.

Recapitulation.

	Lb. Cal.
Heat in converter body at starting	8,034,970
" melted pig-iron	6,750,500
" spiegeleisen	750,000
" blast	80,580
Heat of oxidation	4,510,800
" formation of slag	59,890
Total on hand and developed	20,192,740
Heat in converter body at finish	7,183,770
" finished steel	8,632,750
" slag	1,582,350
" escaping gases	2,786,000
Heat absorbed in decomposing moisture	182,130
Heat conducted to the air	34,630
Heat lost by radiation	25,240
Total accounted for	20,426,870

Another way of expressing this balance is to itemize the avenues of heat evolution and utilization, as follows:

	Lb. Cal.
Received from converter body	851,200
Received by oxidation	4,510,800
Received by formation of slag	59,890
Total	5,421,890

	Lb. Cal.
Used, excess of heat in gases over blast	2,000,420
Used, excess of heat in steel and slag over pig iron and spiegel	2,714,600
Decomposition of moisture	182,130
Radiation and conduction	59,870
Total	5,556,020

[The next instalment of these calculations will discuss the relative heating efficiencies of different ingredients of the pig-iron, in the Bessemer process.]

Tests of Titaniferous Slags.

By Chas. N. Cox, Jr., and Loring C. Lennon.

There are found in several places in the United States large bodies of iron ore of high grade, but containing titanic acid in considerable amounts. This ore is not being mined at present because of the prevailing prejudice among blast-furnace men against ores containing titanic acid, which is supposed to be the source of several difficulties in the working of the blast furnace, such as high smelting point and viscosity of the slag and the formation of accretions in the crucible (hearth).

Mr. A. J. Rossi, of New York, has constructed a small blast furnace and conducted quite an extensive series of experiments on the feasibility and fluidity of actual slag mixtures containing varying percents of titanic acid. His general conclusions were that it was perfectly feasible to obtain mixtures in the blast furnace that would produce slags fully as fusible and fluid and in every way as satisfactory as those obtained when using ore free from titanic acid. He attributed the trouble that this substance caused, in the early days of the iron manufacture in this country, to the ignorance of the blast furnace men in regard to the material they were handling rather than the influence of the titanium. He thinks that the modern blast furnace man's fear of titanium-bearing ores is based almost entirely upon prejudice, because of the early mistakes in the methods of handling them.

In Vol. III of the Transactions of the American Institute of Mining Engineers, there is an article by Mr. Rossi, in which he states that in making up slags for a titanium-bearing ore, the titanic acid (TiO_2) should enter in as so much silica (SiO_2), and all calculations should be made with this in view.

In this article he gives the percentage composition of several slags which proved to be quite satisfactory, and which will appear later in this article as comparative tests. With these facts in view it was decided to make a number of tests on titaniferous slag mixtures in order to verify Mr. Rossi's experiments, and thus perhaps interest those who are in a position to do some experimental work on a commercial scale. Our tests were made in the Engineering Department of Colorado College.

Before attempting to outline the method of conducting these tests, it will be desirable to say something about the materials used in making up the slag. Of necessity the tests were conducted on a small scale, and so with the exception of the limestone and rutile (which formed the source of titanic acid), the ingredients were "C P" chemicals. Fortunately, a limestone was obtained, quarried in the Garden of Gods, which was, with the exception of 0.018 per cent $Fe_2O_3 + Al_2O_3$ and 0.0018 per cent MgO, pure $CaCO_3$. For the sake of simplifying the computations it was called 99.8 per cent pure, and the $Fe_2O_3 + Al_2O_3$ and MgO ignored.

The rutile, which was obtained from the American Rutile Co., of Washington, D. C., was found to contain:

Titanic acid TiO_2	91.32%
Silica SiO_2	4.70
Ferrous oxide FeO	4.02
	100.4

As there was no guide to follow in the matter of types of mixtures to use, it was decided to test and compare the monosilicates, bisilicates, trisilicates and sesquisilicates. As all metallurgists know, the meanings of these prefixes are simply that the ratio of the oxygen atoms in combination with the bases of the slags to the oxygen atoms in combination with the acid-forming elements are respectively 1 to 1, 1 to 2, 1 to 3 and 2 to 3, and they are represented by the general formulæ $2RO \cdot SiO_2$, $RO \cdot SiO_2$, $2RO \cdot 3SiO_2$ and $4RO \cdot 3SiO_2$ respectively, when R represents any divalent base or bases. Also, $2R_2O_3 \cdot 3SiO_2$, $R_2O_3 \cdot 3SiO_2$, $2R_2O_3 \cdot 9SiO_2$ and $4R_2O_3 \cdot 9SiO_2$ are the corresponding formulæ when R represents trivalent bases. In the calculations that follow it has been assumed that TiO_2 may take the place of SiO_2 in these formulæ, and also that the bases in the slag divide themselves between these two acids in proportion to the respective masses of the acids. It was decided to use slag containing the following percentages of titanic acid: ½, 1, 3, 5, 7, 10, 14. In these slags alumina and calcium oxide in the ratio of 1 to 3 was used. In the slags containing ½ per cent and 1 per cent of TiO_2 the c.p. titanic acid was used; in the others rutile, the analysis of which was given above.

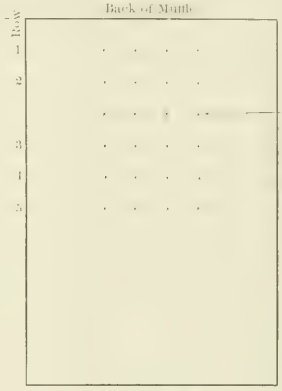
Two other sets of slags were tested, viz.: slags containing alumina, calcium oxide and magnesia, in the ratios 12 to 33 to 3, and the folowing per cents of titanic acid, 3, 7 and 14. Also slags containing alumina and calcium oxide in the ratio 1 to 4, and the following percents of titanic acid: 1, 3, 5, 7, 10, 14. The reasons for using these different ratios between the bases in the slags is that thus are obtained the percents of the bases usually found in ordinary blast furnace slags.

In the following tables will be found the results of computations for sixty-four different slags, and with the exception of the two columns following the one marked "total percent," they require no explanation. The column marked "percent rutile" was introduced for convenience in computing the weight of TiO_2 and FeO, required for a charge of any size of the slag. The column marked "percent raw," is inserted for similar reasons, the figures in this column being those representing the total percents of silica required less that contained in the rutile. All slags with 0.5 per cent and 1 per cent of TiO_2 were made up from c.p. TiO, without rutile.

To these tables is added one of four slag mixtures that are said by Mr. Rossi to behave in a satisfactory manner in the blast furnace; also the composition of a regular blast furnace slag from one of the Jones & Loughlin Steel Co's blast furnaces, making Bessemer pig iron, and two of Prof. H. O. Hofman's ferrous silicates. These were used as giving a standard of comparison with actual practice and as checks on the accuracy of temperature readings.

The testing of these slags was completed after two different methods, and with two distinct objects in view, viz.: to find the fusion points (temperatures) of the mixtures, and to find the comparative fluidity of the slags. At a temperature of 1,450° C. For the fusion point test cones were used, not quite properly speaking, triangular pyramids, as near the size of Prof. Orton's Seger cones as they could conveniently be made. This method of making the fusion tests was adopted because it seemed to be the one finding most ready acceptance at the present time for such work.

The first method tried for making the cones was mixing the material (ground to pass a 100-mesh sieve) with a sufficient quantity of a syrup (made by boiling dextrine in water) to make a stiff paste, and then pressing this paste into a mould with a rammer made for the purpose. This method was unsatisfactory, because of the sticking of the material in the mould, and resort was finally made, after considerable experimenting, to rolling the mixture out into thin rolls and cutting out the cones with a spatula. It was also found, after several trials, that gum tragacanth was a much more preferable binding material; when mixed with the slag-forming materials it made a sort of dough, which was much easier to handle than the brash paste formed with dextrine. Then it was found that when the gum tragacanth cones were thoroughly dry, they could be placed immediately in a very hot furnace without breaking, while dextrine cones had to be heated up gradually, and the organic matter given time to burn off.



PLAN OF CONE ARRANGEMENT

To test these cones they were placed erect in a dry fire-clay bed which was introduced into the muffle. The cones were arranged in rows, with a row of Prof. Orton's cones across the middle of the bed, as shown in the adjoining diagram.

The above figure shows the arrangement of the slag and standard cones in the furnace. Row No. 2 was usually the hottest part of the muffle, and the row of standard (Orton) cones occupied a position half way between row No. 2 and row No. 5, which is the coolest row. In making up the bed for a second test the slag cones in rows 4 and 5 were interchanged with those in rows 1 and 2, so that by averaging the results obtained for any slag no great error, due to location in the furnace, will enter in. In this manner eight to twenty cones were tested at once. Six cones of each slag were made and tested, two at a time, so as to eliminate as far as possible any variations in temperature in different parts of the furnace.

The fluidity tests were made first in pocket crucibles, but it was found impossible to keep the atmosphere sufficiently reducing to prevent their burning through. Platinum ware was too expensive, so it was decided to use the ordinary fire-clay crucible, after first lining it with a mixture that was to be fused in it, and then glazing it by a preliminary heating. This method gave excellent results. In making these tests four of the glazed crucibles were placed with their charges of 20 grams each in the furnace, and a No. 14 Orton Seger cone placed in

Table No. 1.
Ratio of lime to aluminum = 3 to 1.

Silicate Type.	TiO_2 %	*FeO* %	SiO_2 %	Al_2O_3 %	*CaO* %	*Total* %	*Rutile* %	*Raw* SiO_2 %	*Slag No.*
Mono	.5	.0	37.594	15.476	46.429	99.999	.0	37.594	1
Mono	1.	.0	37.17	15.46	46.37	100.	.0	37.17	2
Mono	3.	.132	35.456	15.353	46.059	100.	3.28	35.302	3
Mono	5.	.23	33.748	15.255	45.767	100.	5.475	33.491	4
Mono	7.	.307	32.040	15.163	45.49	100.	7.665	31.68	5
Mono	10.	.44	29.507	15.013	45.040	100.	10.950	28.992	6
Mono	14.	.644	26.058	14.825	44.474	100.01	15.331	25.337	7
Bi	.5	.0	54.455	11.258	33.785	99.998	.0	54.455	8
Bi	1.	.0	54.010	11.242	33.747	99.990	.0	54.010	9
Bi	3.	.132	52.207	11.16	33.501	100.	3.28	52.053	10
Bi	5.	.230	50.413	11.085	33.278	100.	5.475	50.156	11
Bi	7.	.307	48.617	11.014	33.062	100.	7.665	48.257	12
Bi	10.	.44	45.920	10.903	32.728	100.	10.95	45.414	13
Bi	14.	.644	42.320	10.753	32.274	100.	15.331	41.608	14
Tri	.5	.0	64.087	8.853	26.56	100.	.0	64.087	15
Tri	1.	.0	63.628	8.842	26.528	99.998	.0	63.628	16
Tri	3.	.132	61.776	8.773	26.319	100.	3.28	61.622	17
Tri	5.	.230	59.814	8.739	26.219	100.	5.475	59.557	18
Tri	7.	.307	58.086	8.652	25.955	100.	7.665	57.726	19
Tri	10.	.44	56.085	8.369	25.105	100.	10.95	55.57	20
Tri	14.	.644	53.290	8.014	24.043	100.	15.331	52.578	21
Sesqui	.5	.0	47.372	13.032	39.096	100.	.0	47.372	22
Sesqui	1.	.0	46.936	13.016	39.049	100.01	.0	46.936	23
Sesqui	3.	.132	45.17	12.952	38.773	100.	3.28	45.016	24
Sesqui	5.	.230	43.410	12.839	38.521	100.	5.475	43.153	25
Sesqui	7.	.307	41.652	12.760	38.281	100.	7.665	41.292	26
Sesqui	10.	.44	39.014	12.636	37.01	100.	10.95	38.499	27
Sesqui	14.	.644	35.491	12.466	37.399	100.	15.331	34.770	28

Table No. 2.
Ratio of magnesia to lime to alumina = 3 to 12 to 33.

Silicate Type.	TiO_2 %	*FeO* %	SiO_2 %	Al_2O_3 %	*CaO* %	*MgO* %	*Rutile* %	*Raw* SiO_2 %	*Slag No.*
Mono	3.	.132	35.939	15.232	41.880	3.808	3.28	35.785	29
Mono	7.	.307	32.518	15.044	41.370	3.761	7.665	32.158	30
Mono	14.	.644	26.524	14.708	40.447	3.677	15.331	25.803	31
Bi	3.	.132	52.708	11.030	30.359	2.76	3.28	52.554	32
Bi	7.	.307	49.111	10.895	29.861	2.726	7.665	48.751	33
Bi	14.	.644	42.811	10.636	29.248	2.659	15.331	42.090	34
Tri	3.	.132	62.21	8.664	23.827	2.166	3.28	62.056	35
Tri	7.	.307	58.516	8.544	23.496	2.136	7.665	58.156	36
Tri	14.	.644	52.043	8.328	22.902	2.082	15.331	51.322	37
Sesqui	3.	.132	45.827	12.76	35.090	3.19	3.28	45.673	38
Sesqui	7.	.307	42.197	12.624	34.715	3.156	7.665	41.837	39
Sesqui	14.	.644	35.981	12.343	33.946	3.086	15.331	35.26	40

Table No. 3.
Ratio of lime to alumnia = 4 to 1.

Type Silicate.	TiO_2 %	*FeO* %	SiO_2 %	Al_2O_3 %	*CaO* %			*Slag No.*
Mono	1.	.0	36.575	12.485	49.94	.....	.....	41
Mono	3.	.132	34.864	12.402	49.602	.....	.....	42
Mono	5.	.230	33.16	12.322	49.288	.....	.....	43
Mono	7.	.307	31.457	12.248	48.988	.....	.....	44
Mono	10.	.44	28.93	12.126	48.504	.....	.....	45
Mono	14.	.644	25.489	11.974	47.893	.....	.....	46
Bi	1.	.0	53.41	9.117	36.473	.....	.....	47
Bi	3.	.132	51.612	9.051	36.205	.....	.....	48
Bi	5.	.230	49.821	8.989	35.959	.....	.....	49
Bi	7.	.307	47.834	8.972	35.887	.....	.....	50
Bi	10.	.44	43.347	8.842	35.371	.....	.....	51
Bi	14.	.644	41.757	8.72	34.879	.....	.....	52
Tri	1.	.0	63.063	7.187	28.75	.....	.....	53
Tri	3.	.132	61.213	7.13	28.525	.....	.....	54
Tri	5.	.230	59.371	7.079	28.319	.....	.....	55
Tri	7.	.307	57.533	7.032	28.128	.....	.....	56
Tri	10.	.44	54.693	6.973	27.894	.....	.....	57
Tri	14.	.644	51.687	6.854	27.415	.....	.....	58
Sesqui	1.	.0	46.320	10.530	42.144	.....	.....	59
Sesqui	3.	.132	44.559	10.462	41.847	.....	.....	60
Sesqui	5.	.230	42.803	10.392	41.574	.....	.....	61
Sesqui	7.	.307	41.049	10.329	41.315	.....	.....	62
Sesqui	10.	.44	38.418	10.229	40.914	.....	.....	63
Sesqui	14.	.644	34.994	10.091	40.361	.....	.....	64

No.					
1	36.78	1.85	27.83	9.18	
2	25.11	3.46	26.72		
3	34.38				22.10
4	64.80	.90	.07		

Slags investigated by Prof. H. O. Hofman.

		40.81			
		31.55	24.15		

Gray slag from blast furnace making Bessemer pig iron, Jones & Laughlin's Steel Company, Pittsburg, Pa.

		.85	34.10	15.87	41.70		

their midst. The temperature was raised until the cone bent over, and a few minutes allowed for the mass in the crucible to become quiet, when it was removed from the furnace and immediately poured on a hot inclined slab. Tables VI. and VII. give the results of these tests.

The manner of noting the temperature of the fusion tests on the silicate cones has not been explained; it may be well to say that the cones were watched through a peep-hole in the door of the muffle, and the temperature of fusion of any particular slag determined by noticing which Orton-Seger cone was going down when it bent over. As will appear in the tables, the range of fusion points was within the limits of No. 7 cone (1,270° C.) to No. 11 (1,350° C.). It was also noticed that the slag cones bent much more rapidly than the Orton cones when they had started, so much so in fact, that if cones were put into a very hot furnace considerable difficulty was experienced in judging the fusion points.

In these experiments the standard cones used were made by Prof. Edward Orton, Jr., Columbus, Ohio. At this point it may be well to state that throughout the fusion tests the method set forth by Prof. H. O. Hofman, in his article entitled "Temperature of Formation of Various Silicates" (Trans. A. I. M. E., Vol. XXIX.) was followed.

At the outset of the work it was expected to make use of a Le Chatelier pyrometer, together with the Orton cones, but the only instrument at hand had the end of the porcelain tube broken off, so that the wires were exposed to the gases of the muffle. It was found that after the pyrometer had remained in the furnace for a little time, the readings would go down while the cone indicated that the furnace was as hot as before. Whatever may have been the cause of this drop in the pyrometer readings (it was ascribed to the absorption of gases by the wires) they were very unsatisfactory, even when attempts were made to correct them by cone tests, and so it was decided to discontinue their use.

Many difficulties were encountered in obtaining a temperature sufficient to do the work. First was tried a portable fire-clay assay furnace, designed for use with anthracite or coke, but the highest temperature obtained with it was about 1,200° C., as against 1,500° C. desired. Attempts to secure an electrical resistance furnace failed, as no dealer could be found to furnish one that could be relied upon. A gasoline furnace failed to answer the requirements though shipped with a guarantee.

Finally, a small Sturtevant blower was connected by a 4-inch pipe to the ash-pit of a stationary assay furnace with a deep fuel bed. With good coke packed around the muffle and forced draft from the blower, it was possible to attain a temperature of 1,500° in half an hour.

The high temperature used, especially in the fusibility tests, rapidly destroyed the muffle supports so that the muffles themselves were short lived, and much delay and many shut-downs were thus caused. Thus the work was at times discouraging, and it was impossible in the limited time to perform as many experiments as were desirable.

The two sets of tables that follow need but little further explanation, but it may be well to say that a number of tests were made at first with the cones which were so unsatisfactory that the results were not entered in the tables. Instead of giving the temperature readings in terms of the Orton-Seger cones, they have been translated into degrees Cent.

In the column headed "remarks," in Table No. VI., will be noticed at intervals the expression "in the front of the muffle." This is not intended to mean in the very front part of the muffle (for the muffle was never filled to its capacity, and the charges were always placed in the back half), but rather those cones nearest the front, and in the front of the row of standard cones.

The muffle was small (11 inches x 7 inches x 5 inches), and it was found that at times when the coke did not work down along its sides evenly, or when the blast became partially choked at one point, there would often be as much as [illegible] difference in a distance of 6 inches. This cooling occurred almost always in the front part of the muffle, because at this point more difficulty was encountered in keeping the coke packed, due to the fact that the muffles were sunk into the brick work of the furnace about 8 inches, which made it difficult to use a poker effectively. Then, too, the front part of the furnace was more troublesome to keep free from accretions, and so the front part of the muffle was almost always cooler than the back part.

These difficulties have made the results much less exact and less uniform than is to be desired from a scientific point of view, but it is believed that remembering the plan of arranging the cones in successive tests, the average of the separate cones of each mixture is close enough for commercial purposes.

$CaCO_3$ in place of CaO, was used in making up the cones, this being done to avoid the slaking effects of the CaO, since the cones had to be made up wet, and it was thought that upon drying they would absorb CO_2 from the air of CaO were used, causing expansion and cracking. In the [illegible] tests however, CaO was used, as these were to be made with dry mixtures. These latter were calcined at a temperature of 1,410° C. (No. 14 Orton cone), and the last 2 hours [illegible] plate, heated to about 900° C. and sealed [illegible] = 8, the flow being measured along the gate.

1,410° C. was taken as the standard temperature for the fluidity tests, because it was the highest temperature that could be maintained without serious detriment to the furnace. In Table VI. the temperature readings marked * were taken by means of the Le Chatelier, readings not so marked were taken by means of the Orton cones.

CONCLUSIONS.

From a study of the fusion tests it is noted that those cones containing magnesia fused at a temperature slightly less gen-

TABLE VI.

Slag No.	Cone No.	Temp.	Remarks.
1	1	1295°*	
1	2	1295°*	
1	3	1290°*	
1	4	1290°*	
1	5	1290°*	
1	6	1290°*	
2	1	1300°*	
2	2	1300°*	
2	3	1305°*	
2	4	1305°*	
2	5	1300°*	
2	6	1300°*	
3	1	1290°*	
3	2	1295°*	
3	3	1260°*	
3	4	1260°*	
3	5	1300°*	
3	6	1300°*	
4	1	1295°	
4	2	1300	
4	3	1280°	
4	4		
4	5	1290°	
4	6	1290°	
5	1	1292°*	
5	2	1292°*	
5	3	1290°*	
5	4	1280°*	
5	5	1305°*	
5	6	1320°*	
6	1	1300°*	
6	2	1320°*	
6	3	1270°*	
6	4	1270°*	
6	5	1270°*	
6	6	1270°*	
7	1	1320°*	
7	2	1320°*	
7	3	1270°*	
7	4	1270°*	
7	5	1330°*	
7	6	1330°*	
8	1	1310°*	
8	2	1330°*	
8	3	1320°*	
8	4		Broken.
8	5		Broken.
8	6		Broken.
9	1	1305°*	
9	2	1315°*	
9	3	1295°*	
9	4	1300°*	
9	5		Broken.
9	6		Broken.
10	1	1270°	
10	2	1270°	
10	3	1330°	
10	4	1330°	
10	5		Broken.
10	6		Broken.
11	1	1330°	
11	2	1330°	
11	3	1270°	
11	4	1270°	
11	5	1270°	
11	6	1270°	
12	1	1310°	
12	2	1310°	
12	3	1310°	
12	4	1310°	
12	5	1290°	
12	6	1290°	
13	1	1270°	
13	2	1310°	
13	3	1270°	
13	4	1270°	
13	5	1270°	
13	6	1270°	
14	1	1290°	
14	2	1290°	
14	3	1270°	
14	4	1270°	
14	5	1310°	
14	6	1290°	
15	1	1330°*	
15	2	1325°*	
15	3	1270°*	
15	4	1450°*	
15	5		Broken.
15	6		Broken.
16	1	1290	
16	2	1290	
16	3	1290	
16	4	1290	
16	5	1290	
16	6	1290	
17	1	1310	
17	2	1310	
17	3	1290	
17	4	1290	
17	5	1270	Cone put in hot muffle apparently [illegible]
17	6	1270	[illegible]
18	1	1310	
18	2	1310	
18	3	1270	
18	4	1290	
18	5	1290°	
18	6	1290°	
19	1	1270°	
19	2	1270°	
19	3	1290°	
19	4	1290°	
19	5	1270°	
19	6	1270°	
20	1	1310°	
20	2	1310°	
20	3	1270°	Furnace too hot when cones put in. Reading not satisfactory.
20	4	1270°	
20	5	1270°	
20	6	1270°	
21	1	1290	
21	2	1290°	
21	3	1270°	
21	4	1270°	
21	5	1270	Same as above.
21	6	1270°	
22	1	1290°	
22	2	1290°	
22	3	1330°	In front of muffle.
22	4	1350°	
22	5	1270°	
22	6	1270°	
23	1	1310°	
23	2	1310°	
23	3	1270°	
23	4	1270°	
23	5	1270°	
23	6	1270°	
24	1	1310°	
24	2	1290°	
24	3	1290°	
24	4	1290°	
24	5	1270°	Readings unsatisfactory, muffle too hot when cones put in.
24	6	1270°	In front row.
25	1	1290°	
25	2	1290°	
25	3	1290°	
25	4	1290°	
25	5	1290°	
25	6	1290°	
26	1	1290°	
26	2	1290°	
26	3	1290°	
26	4	1290°	
26	5	1290°	
26	6	1290°	
27	1	1290°	
27	2	1290°	
27	3	1290°	
27	4	1290°	
27	5	1290°	
27	6	1290°	
28	1	1290°	
28	2	1290°	
28	3	1290	
28	4	1290°	
28	5	1270°	
28	6	1290	
29	1	1310°	In front of muffle.
29	2	1310°	
29	3	1290°	
29	4	1290°	
29	5	1290°	
29	6	1290°	
30	1	1310	In front of muffle.
30	2	1310°	
30	3	1290	
30	4	1290°	
30	5	1270°	
30	6	1290°	
31	1	1290°	
31	2	1290°	
31	3	1290°	
31	4	1290	
31	5	1270°	
31	6	1270	
32	1	1310°	
32	2	1310°	
32	3	1290	
32	4	1290	
32	5	1270°	
32	6	1270°	
33	1	1310°	
33	2	1310°	
33	3	1290	
33	4	1290	
33	5	1290	
33	6	1290	
34	1	1290	
34	2	1290	
34	3	1290	
34	4	1290	
34	5	1290	
34	6	1290	
35	1	1310	
35	2	1310	In front of muffle
35	3	1310	
35	4	1310°	
35	5	1290	
35	6	1290	
36	1	1290	In front of muffle.
36	2	1310	
36	3	1290	
36	4	1290	
36	5	1290	
36	6	1290	
37	1	1290	
37	2	1290	
37	3	1270	
37	4	1270	
37	5	1270	
37	6	1270	
38	1	1290°	In front of muffle
38	2	1290°	
38	3	1270°	
38	4	1270°	
38	5	1270°	
38	6	1270°	
39	1	1290°	In front of muffle.
39	2	1290°	
39	3	1270	
39	4	1290°	
39	5	1270°	
39	6	1270°	
40	1	1290°	
40	2	1290°	In front rows.
40	3	1270°	
40	4	1290°	
40	5	1270	
40	6	1270	
41	1	1310°	
41	2	1310°	
41	3	1290	
41	4	1290°	
41	5	1270	
41	6	1290	
42	1	1310°	Reading unsatisfactory, bending of cone not noticed at first. When observed, seyer No. 9 was going down.
42	2	1310°	
42	3	1290°	
42	4	1290°	
42	5	1270°	
42	6	1290	
43	1	1310°	Reading unsatisfactory. Same as above.
43	2	1310°	
43	3	1290°	
43	4	1290°	
43	5	1291°	
43	6	1310°	
44	1	1310°	Same as 42, 1 and 2.
44	2	1310°	
44	3	1290°	
44	4	1290°	
44	5	1290°	
44	6	1290°	
45	1	1310°	
45	2	1310°	
45	3	1310°	
45	4	1310 °	
45	5	1310°	
45	6	1310°	
46	1	1310°	
46	2	1310°	
46	3	1290°	
46	4	1290°	
46	5	1310°	
46	6	1310°	
47	1	1310°	
47	2	1310°	
47	3	1350°	In front rows.
47	4	1350°	
47	5	1310°	
47	6	1310°	
48	1	1330°	In front rows.
48	2	1330°	
48	3	1290°	
48	4	1290°	
48	5	1290°	
48	6	1290	
49	1	1330°	In front rows.
49	2	1330	
49	3	1290	
49	4	1290°	
49	5	1290	
49	6	1290	
50	1	1330°	In front rows.
50	2	1330	
50	3	1290	
50	4	1290	
50	5	1290	
50	6	1290	
51	1	1290	
51	2	1290°	
51	3	1290	
51	4	1290	
51	5	1310	
51	6	1310	
52	1	1290	
52	2	1290	
52	3	1290°	
52	4	1290°	
52	5	1310°	
52	6	1310°	
53	1	1350°	
53	2	1350°	Only slightly fused.
53	3	1350°	
53	4	1350°	
53	5	1310°	
53	6	1310°	
54	1	1310°	
54	2	1310°	
54	3	1290°	
54	4	1290°	
54	5	1310°	
54	6	1310°	
55	1	1310°	
55	2	1310°	
55	3	1290°	
55	4	1290°	
55	5	1290°	
55	6	1290°	
56	1	1290°	
56	2	1290°	
56	3	1290°	
56	4	1290°	
56	5	1290°	
56	6	1290°	
57	1	1310°	
57	2	1310°	
57	3	1290°	
57	4	1290°	
57	5	1290°	
57	6	1290°	
58	1	1330°	
58	2	1330°	
58	3	1290°	
58	4	1290°	
58	5	1290°	
58	6	1290°	
59	1	1330°	
59	2	1330°	
59	3	1290°	
59	4	1290°	
59	5	1310°	
59	6	1310°	
60	1	1330°	
60	2	1330°	
60	3	1310°	
60	4	1310°	
60	5	1310°	
60	6	1310°	
61	1	1290°	
61	2	1290°	
61	3	1310°	
61	4	1310	
61	5	1290°	
61	6	1290	
62	1	1310°	
62	2	1310°	
62	3	1290°	
62	4	1290°	
62	5	1310°	
62	6	1310°	
63	1	1310°	
63	2	1310	
63	3	1290°	
63	4	1290	
63	5	1310°	
63	6	1310	
64	1	1310°	
64	2	1310	
64	3	1290	
64	4	1290	
64	5	1310	
64	6	1310	

Gray blast furnace slag Jones & Laughlin's 1 ur making Bessemer iron 1290

Silicates given in Prof. Hofmann's article

1	1	1290°	These temperatures agree exactly with ones given by Prof. Hofmann
1	2	1290°	
2	1	1310°	
2	2	1310°	

TABLE VII
FLUIDITY TESTS

Slag No.	
1	Glassy white and very viscous.
2	Glassy white and very viscous.
3	Glassy, nearly mottled, flowed 1½"
4	Mottled brown glass, flowed 1½"
5	Mottled brown glass, flowed 1 1 16".
6	Dark brown glassy, flowed 2½".
7	Dark brown, glassy, flowed 2".
8	Glassy, viscous, very slightly fluid.
9	Glassy, viscous, very slightly fluid.
10	Stony mass. Viscous, no flow.
11	Glassy, viscous, [illegible] in nickel
12	Glassy, viscous, crystalline and pearled
13	Glassy, viscous, [illegible]
14	Glassy, viscous, reading 1410. Not [illegible]
15	White, dead surface, no flow.
16	White, dead surface, no flow.
17	Glassy, viscous.
18	Glassy, viscous.
19	Glassy, light brown, very viscous

Slag No.	
20	Glassy, gray, very viscous.
21	Glassy, brown, viscous.
22	Stony, gray, no flow.
23	Glassy, green, viscous.
24	Glassy, brown, viscous.
25	Glassy, milky, viscous.
26	Glassy, dark brown, flowed 1½"
27	Glassy, dark brown, flowed 2 1 16".
28	Glassy, dark brown, flowed 1 15 16".
29	Glassy, very light brown, flowed 1½"
30	Glassy, mottled brown, flowed 1½"
31	Glassy, mottled brown, flowed 1½"
32	Glassy, gray, very viscous
33	Glassy, dark opaque, viscous
34	Glassy, mottled black, flowed 1½"
35	Gray, earthy mass, no flow.
36	Gray, earthy mass, no flow.
37	Stony, no flow.
38	Glassy, mottled green, viscous
39	Glassy, chrome mottled viscous

Slag No.	
40	Glassy, black; flowed 1¼".
41	Glassy, clear light green; flowed 1¼".
42	Glassy, clear light green; flowed 1½".
43	Gray, opaque mottled glass; flowed 1⅛".
44	Gray, opaque mottled glass; flowed 1⅛".
45	Dark glass; flowed 2".
46	Black stony mass; flowed 2¼".
47	Stony, gray mass, no flow In front row of muffle.
48	Stony, gray mass; no flow
49	Glassy, dark green, viscous.
50	Glassy, dark green, viscous.

Slag No.	
51	Glassy, brown, opaque; flowed 1"
52	Glassy, brown, opaque, flowed 1"

Rossi's Slags.

1	Gray, stony; flowed 1¼".
2	Black glass; flowed 2¼". Scoured crucible.
3	Dark gray earth mass; no flow.
4	Black stony mass; no flow.

Pulverized Gray Blast Furnace Slag.

1	Would not pour out of crucible.
2	Would not pour out of crucible.

erally speaking, than the others; also that the cones containing lime and alumina in the ratio 4 to 1 fused at a temperature slightly higher than those containing the same elements in the ratio 3 to 1. These differences were, however, slight, and since the fusion points of all the slags come within the range 1,270° C. to 1,350° C. (this being considerably below the temperature existing in the bosh and hearth of the iron blast furnace), we consider ourselves justified in saying that no difficulty would be encountered in fusing any of these mixtures in the ordinary blast furnace, but their fluidity after fusion is a question still to be considered.

In these cone tests very little, if any real differences, were found in the fusion points of the different silicates tested. The character of the slag was noted in each case, but this varies so largely with the location of the cones in the muffle, the time that they remained in (waiting for others of the same tests to fuse), and the maximum temperature reached before they were removed, that it can give no real information in regard to the comparative merits of the slags.

An idea may be had of the accuracy with which the temperatures of fusion were taken by a glance at the work on Prof. Hofman's silicates, given in Table No. VI.

From a study of the results obtained by the fluidity tests it is seen that the monosilicates are the most fluid, the bisilicates and sesqisilicates are quite viscous, and the trisilicates containing the higher percentages of TiO_2 were somewhat fluid.

Little difference was found in slags made up with varying proportions of bases; but it was found that the fluidity almost invariably increased with the increase of TiO. It is to be noted, however, that owing to the use of rutile with the increase of TiO_2, there was also a slight increase of the percentage of FeO, though the total amount of FeO was not higher than is frequently found in blast-furnace slags.

In no case were the comparison slags found to be more fluid than our own; and since these comparison slags have proved to be all that is required in blast furnace work, we see no reason why the more fluid, at least, of our slags might not be used successfully in a blast furnace.

Of course, it was impossible to judge accurately from the small charges used, what would be the action of large masses of a slag. In our case the small quantity of intensely hot material was bound to chill very quickly upon coming into contact with the air; and then to add to this, the hot plate we used was heated by gas, and 300° C. was about as hot as it could be kept, so that it was cold to the slag.

Our main object in performing these experiments was, if possible, to prove that TiO_2 in an ore, if properly handled, is no great detriment. There is good authority for the statement that blast furnaces have been run advantageously on ores containing as high as 5 per cent TiO.

Now we have found that apparently the fluidity of our slags increased with the increase in the percent of titanic acid. Therefore, we believe that TiO_2 in moderate quantity in ore, if properly handled in a blast furnace, should give less trouble than has been heretofore supposed.

Colorado College,
Colorado Springs, Col.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

CHEMICAL AND ELECTROCHEMICAL ENGINEERING.

Platinized Platinum Anodes for Chlorate Production.—In the electrolysis of sodium and potassium chlorides platinized platinum anodes are not yet used commercially, although Foerster found that not only the voltage at the same is lower by 0.5 or 0.6 volts than at polished platinum electrodes, but that the ampere-hour efficiency in the production of hypochlorites and chlorates is thereby considerably increased. The chief reason why platinized-platinum electrodes have not found favor in practice is that they were not durable. W. Geibel describes in the *Zeit. f. Elecktrochemie*, Nov. 9, experiments made in the laboratory of the well-known platinum works of Heraeus. The chief result is that if the platinum-black layer on the electrode is transformed into platinum-gray by careful heating, the layer sticks firmly to the electrode and the latter is sufficiently durable for use in practice. At the same time the platinum-gray electrode gives about the same increase of efficiency over the polished plate as can be obtained with platinum-black. The transformation of platinum-black into platinum-gray is best carried out in an electric furnace, the temperature of which may be exactly controlled. Since it is important that the effective surface is as large as possible, the heating temperature should not be too high.

Switchboard for Electrolytic Analysis.—In the *Engineering and Mining Journal* of Nov. 17, E. Larison describes a switchboard attachment which he has found useful for the electrolytic determination of copper. The source of current is a three-cell storage battery charged periodically from a 125-volt direct-current lighting circuit. To reduce the voltage a bank of twenty lamps of 16 cp. in parallel is placed in series with the battery. Two separate circuits in parallel are provided on the switchboard, each including a number of resistances in series. To obtain a close control of the current supplied to the individual solutions undergoing electrolysis, a second auxiliary switchboard is used. The connections are shown in diagram.

Electro-Analysis.—A paper by J. R. Withrow on the electrolytic precipitation of gold, with the use of a rotating anode, is found in the October issue of the *Journal* of the American Chemical Society. The author experimented with solutions of auric chloride in the presence of, first, potassium cyanide, and, second, sodium sulphide. Pure potassium cyanide was found to be a better electrolyte than sodium sulphide for this particular purpose.

Concentration of Sulphuric Acid.—It is a well-known fact that the concentration of sulphuric acid in platinum vessels of large size requires a very costly plant, while the use of porcelain pots is attended with considerable troubles, due to the breakage of the pots. A concentration furnace of modern construction, designed by G. Krell, which depends upon the use of cast iron pipes, is described by P. Schuberg in the *Zeitschrift für Chemische Apparatenkunde* May 15. The apparatus takes acid which has been concentrated previously to 62° B. in lead vessels. The concentration of the acid takes place in two horizontal cast iron pipes, which are half filled with the acid, which passes through them successively. The pipes are made of a special brand of cast iron, dense and free from blow holes,

and are arranged in a cast iron pan, which is filled with molten lead; the surface of the latter must be at least 30 millimeters above the cast iron pipes. The joint between the pipes and the wall of the cast iron vessel is made in such a manner as to allow for expansion and contraction, while at the same time preventing the escape of molten lead. Each pipe is 2,500 mm. long, 200 mm. wide and 20 mm. thick, and an even thickness at all parts of the pipe is of prime importance, as it allows the turning of the pipe around its axis so as to expose always fresh surfaces to the action of the acid. The acid attacks the first pipe to a slight extent, especially at the level of the liquid, but the second pipe suffers hardly any attack. At the fire-box side of the furnace the two pipes are joined by a cast iron pipe, which near the first pipe is provided with a baffle plate dipping below the level of the acid. By this means the distillate from each pipe can be collected separately. The apparatus is also provided with a sediment collector, and means are arranged for scraping the sediment out of the concentrating pipes. A considerable quantity of sediment is formed in them, and it has to be removed regularly and conveniently while the furnace is running. At several places, notably at the feed pipe and the discharge pipe of the acid, lead has been used, which is exposed to quite an elevated temperature, although it is cooled. If this were ordinary lead it would decompose in 60° acid at temperatures between 204° and 232° C. The lead used in this apparatus is, however, alloyed with some metal, which makes it more resistant towards hot sulphuric acid and at the same time increases its melting point to 320° C. without decomposition in sulphuric acid. The plant at Bruchhausen-Hüsten, Germany, which occupies only a small space and concentrates glover and waste acid to 92/93 and 97/98 per cent acid, produces in 24 hours 7,500 kg. of 92/93 per cent, or 3,500-4,000 kg. of 97/98 per cent acid. The last named figure is reached when the acid entering into the apparatus has a strength of 60-62° B., and a temperature of 170-180° C. The cast iron pipes last for 300-500 tons of 92 per cent, or 200-300 tons of 98 per cent acid. The cost of running decreases with the increasing number of furnaces, as one man can attend to three furnaces. The cost during a considerable period of practical operation, per 100 kg. finished 98 per cent acid, was as follows: With one double bath, labor, two shifts at $1.00, 5 cents; 19 kg. coal (100 kg. at 37.5 cents), 7 cents; wear and tear and repairs, 1.75 cents; a total of 13.75 cents. With two double baths the cost is decreased to 11 cents, and with three double baths to 10.5 cents. For the 92/93 per cent acid the cost decreases in accordance with the greater productive capacity of the furnace to 9 to 10 cents per 100 kg.

Deacon Chlorine Process.—In the well-known Deacon chlorine process a mixture of air and hydrochloric acid gas is led over a rough surface impregnated with cupric chloride, kept at a temperature of about 400° C. The cupric chloride acts only as a catalyzer, and may be replaced by the chloride of iron, nickel or cerium. The reaction is simply $4HCl + O_2 = 2H_2O + 2Cl_2$. The reaction never runs completely to an end, but approaches a state of equilibrium which, according to the conditions of temperature and concentration, fixes a definite limit to the yield of chlorine which it is possible to attain. G. N. Lewis has studied, in a paper in the October issue of the *Journal* of the American Chemical Society, this equilibrium in the Deacon process. In his experiments the catalyzer cupric chloride required always several days to reach a constant state, after which concordant results were obtained. The time required to bring the gases to equilibrium was greater the lower the temperature, and the smaller the percentage of oxygen present. Experiments were made at three temperatures, 352°, 386° and 410° C., and over a considerable range of concentration, the ratio of oxygen to hydrochloric acid in the final mixture being forty times as great in one case as in another. At constant temperature the results are in accordance with the mass law. The yield of chlorine from a given mixture is greater the lower the temperature, and the change of the equilibrium constant with the temperature is in good agreement with the equation of Van't Hoff. The results may be seen from the following table:

Temp. Degrees C.	Average Pressure in Atmospheres.	Mols. O_2 to 100 Mols. HCl in Original Gas.	Mols. O_2 to 100 Mols. HCl in Final Gas.	x.	K.
352	1.00	92.7	544.0	0.809	4.15
352	0.93	20.7	49.5	0.808	3.95
386	0.98	327.0	1070.0	0.845	2.94
386	0.96	48.8	146.0	0.804	3.01
410	1.08	327.0	1700.0	0.820	2.40

In this table x represents the fraction of the hydrochloric acid decomposed, or, in other words, the ratio of free chlorine to total chlorine in the final mixture. K is the equilibrium constant of the mass law, according to the equation:

$$K = \left(P_{Cl_2}^{\frac{1}{2}} \quad P_{H_2O}^{\frac{1}{2}}\right) \div \left(P_{HCl} \quad P_{O_2}^{\frac{1}{4}}\right)$$

An application is finally made to a calculation of the e. m. f. of the hydrogen-oxygen cell, which is found to be over 1.2 volt at 25°. This confirms the author's former conclusion that the value at present accepted for this e. m. f. is 0.1 volt too low.

Metallurgy.

Removal of Wood in Ore Dressing.—It is a well-known fact that the presence of wood in the ore occasionally gives rise to considerable trouble in the concentrating mills. Mr. A. H. Wethey, in the *Engineering and Mining Journal*, Oct. 20, discusses the method used at the Butte Reduction Works, Butte,

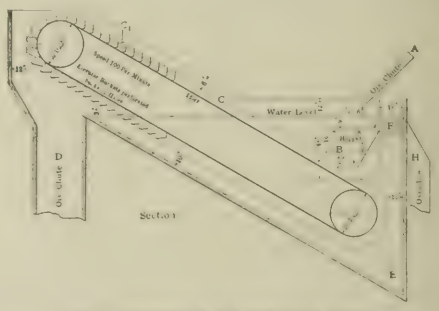


Fig. 1.—Separator for Removing Wood.

Mont., for getting rid of the wood. It has been found by actual experience at that plant that an ordinary mine wedge, 8 inches long, 4 inches wide, and 2 inches thick at the large end, when caught in a 9 x 15 inch crusher, will take as long to work its way through the crusher as a ton of ore. The ore at the above plant passes to a 15 x 24-inch crusher, set to crush down to a minimum of about 2½ inches. No trouble is caused by the presence of wood in this crusher. Previous to passing over this crusher the fine material has been screened off from the ore by means of a grizzley and any iron or steel, such as hammer heads, ends of drill steel, etc., have been removed by a magnet. The ore then passes through trommels, screening out the 1" inch material, the over size going to two 9 by 15 inch Blake crushers. The greatest difficulty was experienced with the pieces of wood in these crushers, and a novel wood separating device was therefore installed. It is shown in Fig. 1. The ore is delivered into the device by the chute A, which passes it into a hopper B below the water level. From there the ore passes into the conveyor C, which in turn brings it to the ore chute D. As the tank E is kept full of water, any wood in the ore, when the latter is discharged into the hopper B, will float to the top of the water. The elevator buckets C on the con-

veyor have perforated backs, through which the water flows out as the elevator lifts the ore out of the water. The water, therefore, rushes back into the tank with sufficient force to wash the floating wood into the screen F, which removes the water, the wood falling down the launder G and the water pouring over the overflow H. It has been found by actual experience that from 20,000 tons of ore there are removed by means of this device 40 tons of wood, and the author estimates that by removing the wood the capacity of the crushing plant is increased 15 per cent.

GOLD.

Tube Mill Practice.—The discussion of Mr. Darling's paper before the Chemical, Metallurgical and Mining Society of South Africa, mentioned previously in these abstracts, has touched several interesting questions connected with tube milling. In the September issue of the *Journal* of that Society, Mr. H. T. Brett explains the difference in the wear of the liners in Australia tube mills and those on the Rand. As far as the mills themselves are concerned, he remarks that the largest size mill in Western Australia is only 16 feet long, and only a few of that size are in use, while the majority are 13 feet long. This fact is due to the greater economy shown by the small tube mills. The in-take of these mills is bell shaped, the feed itself entering through a 2½-inch pipe tapered to 1½ inches at the end, so as to form a nozzle. This pipe forms the bottom discharge of a cone-shaped spitzkasten, usually 6 feet deep, the pressure forcing the feed through the nozzle into the mill. A round plate about 12 inches in diameter at the discharge end, and with long rectangular slots cut in it, was usually furnished with the mill, but it has been replaced by a ⅜-inch plate perforated with ⅛-inch holes. The liners are fastened to the shell with bolts. In the event of a bolt dropping through or working loose, a wooden plug is driven in, which keeps the liner tight until it is replaced. Eight hours' work of three men is required to take off an old set, replace it with new liners and to start the mill again. As far as the author is aware no silex liners have ever been tried. The chilled liners are 1 inch thick, cast in most cases at the mine; they last from four to six months. It is quite remarkable that these 1-inch chilled liners last longer than, say, a 4-inch silex or manganese steel liner, and the author traces the causes of this surprising fact In the first place he points out that there is quite a difference in the coarseness of the sand which enters the mill. Nearly all the Kalgoorlie mines are grinding the total produce to a slime, and the fine sand is returned again and again to the tube mill until 95 per cent of it passes a 150-mesh screen. On the Rand, however, the mills are grinding coarse sand to fine in practically one passage through the mill, and therefore the feed entering a Kalgoorlie mill, being composed of, say, one part of original coarse sand from the battery to ten parts of returned fine sand, is in a much finer state of division than that entering the large mills on the Rand. The author cites as a further proof of the influence of the coarse sand in the life of the liners, the fact that 1 inch liners last six months in some of the mines, where the coarse sand is broken down in grinding pans first and then finished off in tube mills. With reference to the tonnage passed through the mill per day as a factor in the wear of the liners, the author points out that the first experience in the running of the tube mills brought out the fact that with too small a feed the flints and liners begin to grind each other, and the life of the liners increased as the tonnage entering the mill increased. The mills, therefore, are kept loaded to their full capacity, and a tube mill, 13 feet long, grinding to a slime, constantly has a load of close on to 400 tons passing through it, containing about 40 to 45 per cent water. The difference in the wear of the liners between the Rand and Kalgoorlie is thus explained by the fact that in the first place the ore on the Rand is more abrasive than the Kalgoorlie ore, while in the second place the sand enters the tube mill in the latter place in a finer state of division and in larger quantities, taking the size of the mill into consideration. In the case of the grinding [illegible] appear, that is, the coarse sand entering the Rand mill [illegible] much quicker than the fine sand. However, even in the case of a pan, a good order, are found to wear quickly, it is accepted as a sure sign that the pan has not [illegible] pan acts, therefore, exactly opposite as compared with a ball mill, where the rapid wear and tear of the liners shows that the mill is under-loaded.

In a further contribution to the discussion of the paper under consideration, Mr. J. Thomas points out that the method of giving the speed of revolution of a tube mill as so many revolutions per minute is misleading, and he suggests that the speed be expressed in feet per minute at the internal circumference of the shell. It might be possible to vary the speed of the mills by means of slightly conical pulleys, so that the speed could be altered to compensate the wear of the liners. He mentions that on the Lupardsvlei Estate and G. M. Co., small tube mills have been installed, and that the experience with them so far bears out that of the El Oro Co., Mexico; that is to say, that small mills are more efficient per horse-power consumed than large ones.

The Computation of the Crushing Efficiency of Tube Mills.—As the use of tube mills increases it becomes of much interest to make comparisons between individual tube mills or groups of these machines. On this account it is very desirable that a method should be devised by which such comparisons might be accurately made. According to Messrs. S. H. Pearce and W. A. Caldecott, in a recent paper before the Chemical, Metallurgical and Mining Society of South Africa, and printed in the September issue of the *Journal* of that Society, the only method hitherto employed is based on the diminution of the + 60-mesh grade and the increase of the — 90-mesh grade, as is shown in the average gradings of the pulp leaving the battery and of the pulp entering the cyanide works. Thus, for instance, the figures in the following table show a difference of 42 per cent:

Grade	Battery Pulp	Final Pulp	Difference
+ 60	32%	10%	22
+ 60 — 90	12	14	
— 90	56	76	20
			42

This 42 per cent, multiplied by the tonnage of ore crushed in the battery during the month, is taken as an index of the tons of sand required. The authors call attention to the fact, that, though this method is certainly simple, it has the drawback of making no distinction between the products of batteries using different size screens, and assumes that there is the same work performed in reducing sand grains to pass a 60-mesh screen, whatever their original diameters may have been. They suggest a method for drawing conclusions from a grading analysis, which is based upon Rittinger's theory that the work done in crushing is proportional to the surface exposed by crushing. The matter thus resolves itself into the mathematical determination of the relative surface area of the grains of the pulp before and after passing through the tube mill, on the assumption that the average shapes of the particles are similar, and that the weights of particles of the successive diameters included in each grade are equal.

As for given weights of ore the total surface area of all the particles in each lot varies inversely as their average diameters, the relative surface exposed by each grade may be obtained by dividing its weight by the average diameter of particles in the grade. Thus the problem is one for the determination of the average diameters of the particles in the various grades into which the crushing product is divided in screening. Assuming now that the average diameter of the particles in the gradings would be represented by the arithmetical mean of the openings, the following result is obtained

		Before		After	
Grade.	Mean Diameter.	Weight.	Relative Surface.	Weight.	Relative Surface.
—.024 — .010	.017	.32	18.8	.10	5.9
—.010 — .006	.008	.12	15.0	.14	17.5
—.006 — .000	.003	.56	186.7	.76	253.3
		1.00	220.5	1.00	276.7

The additional relative surface exposed is thus found to be 56.2 or 25.4 per cent. The figures given under the heading "grade" refer to the apertures of the screen used, viz.: .024 inch, .010 inch and .006 inch. The authors remark that the defect in the above simple method is that the arithmetical mean of the particles in a grade is greater than the true average diameter, the latter being actually expressed by the following formula of E. Laschinger:

$$d = \frac{d_1 - d_2}{2.3026 \log \frac{d_1}{d_2}}$$

where d = the average diameter of the particles in the grade, d_1 the maximum, and d_2 the minimum diameter of these particles. When this formula is applied to the foregoing case, assuming that the smallest particles in the crushed product have a diameter of .0001 inch, there appears the following result:

		Before		After	
Grade.	Average Diameter.	Weight.	Relative Surface.	Weight.	Relative Surface.
—.024 + .010	.01505	.32	20.1	.10	6.3
—.010 + .006	.00783	.12	15.3	.14	17.9
—.006 + .000	.00144	.56	388.6	.76	527.4
		1.00	424.0	1.00	551.6

The additional relative surface exposed thus would figure up 127.6 or 30.0 per cent. If the above formula were adopted, a table might be constructed of average diameters between a series of maximum and minimum apertures expressed in thousands of an inch, which would simplify calculation.

Milling Plants of Summit County, Colorado.—An article in *Mining Reporter*, Sept. 6, deals with the status of the mills in the above county. It is stated that in the county there are upwards of twenty-five milling plants, having an aggregate capacity of 1,500 tons. More than half of these plants treat the ore by amalgamation and concentration. At several mills amalgamation or concentration alone is practiced, and at one plant, the Masontown mill at Frisco, cyanidation. At the amalgamation plants, stamp crushing is almost universally followed, the stamps varying in weight from 750 to 1,000 pounds. They drop anywhere from 5 to 7½ inches at the rate of 70 to 100 drops per minute. The battery screens have an average of 20-mesh, the plates are as a rule of the normal Colorado type, of an average length of 8 or 10 feet and 4 to 6 feet wide. Inside mortar amalgamation was used at a few of the earlier constructed mills. At one plant crushing mills are used, and at several plants Huntington and Chilean mills are set up. The stamp duty is from 2½ to 3½ tons per stamp per 24 hours. The Masontown mill at Frisco treats the ore by stamp crushing in weak cyanide solution followed by amalgamation, fairly successful results being obtained. Stamp crushing predominates in the amalgamation concentration plants, although at several plants roller mills have been installed, and at one, which is now under construction, a Lane mill will be tried. Regrinding is stated to have been attempted in rare instances, and with indifferent success. The straight concentration plants are stated to employ roll or stamp crushing, the former mainly for quartz-sulphide ores, and the latter for lead and lead-zinc ores. Jig concentration is adopted in the lead and lead-zinc plants, and table concentration at all the plants; the tables include about all known makes, the Wilfley predominating. Magnetic separation in the treatment of the zinc ores is stated thus far to have been simply experimental, it being probable that separators will be installed in several mills now in operation or under immediate or contemplated erection. Coarse crushing is the quite general practice on account of the sliming character of the lead ores. It is stated that in the plants using only amalgamation in treating the more free-milling quartz, a recovery of from 60 to 85 per cent of the gold values and 50 to 75 per cent of the silver values has been obtained on ores of value ranging from $10 to $40 per ton. The amalgamation and concentration mills make an average recovery of from 65 to 85 per cent, and the concentration plants recover about 80 per cent. Higher savings are, however, reported in all three classes of mills in several instances.

Present Milling Conditions in the Cripple Creek District, Col.—The present state of cyanidation in the Cripple Creek district is reviewed by G. E. Wolcott in the Sept. 6 issue of the *Mining Reporter*. According to him the progress in the local treatment of low-grade ores in this district during the past year by cyanidation without roasting has not in the main been very encouraging. Of the four mills which were actively engaged in the treatment of such ores by the cyanide process, none are now running. Without going into details of the different mills, he feels safe in saying that the majority of failures may be classified as due to two main causes, or rather to two different classes of people, namely, the unscrupulous promoter or the misinformed and unqualified enthusiast. As far as the treatment of low-grade Cripple Creek ores on a large scale is concerned, he feels safe in saying, that it will never be satisfactorily accomplished by simple cyanidation without roasting, or, for that matter, even by the aid of roasting. The Wishbone Co. is seeking a solution of the problem by following the Black Hills practice of amalgamation with subsequent cyanidation of the tailings. The author thinks, however, that while this may give better extraction by the saving of coarse gold, it is doubtful if the process will find any application to tellurides. Roasting, while converting the tellurides into free gold, practically amenable to cyanide treatment, will unavoidably, in the case of Cripple Creek ores, produce more or less coarse gold, which is practically impossible to save by the process. It would seem, therefore, to the author, to follow that the only method of cyanidation applicable to all classes of Cripple Creek ores must be one which includes in the process both roasting and amalgamation, and this necessarily precludes the possibility of the general treatment of low-grade ores. He thinks that recent events bearing upon the handling of Cripple Creek ores seem destined to bring about radical changes in milling conditions, and it is a matter of surprise that no consideration has been previously taken by Cripple Creek operators, looking to the treatment of their own ores.

The Use of the Filter Press for Clarifying Solutions.—A paper read before the Chemical, Metallurgical and Mining Society of South Africa, abstracted in the October issue of Electrochemical and Metallurgical Industry, by S. J. Truscott and A. Yates, in which they pointed out the advantages realized by the adoption of the filter press for clarifying gold-bearing cyanide solutions in a plant in Sumatra, called out a contribution from Mr. J. A. Jones, in which he describes the advantages of the use of filter presses, as ascertained at the new plant of the Meyer Charlton Co., on the Rand. When this plant was started two clarifying tanks, each 20 feet in diameter by 5 feet deep, were installed. These tanks had the usual filter bed with mat and jute, covered by 14 inches of coarse sand. It was soon found that their filtering capacity was too small for the requirements and the labor and time wasted in cleaning off the layer of accumulated slime was considerable. One tank, therefore, was removed and an old Johnson press installed which had thirty-four cells, 30 inches square and 1 inch deep, with a filtering area of 425 square feet. In the plant under consideration, cyanide solution is in use

throughout, and the aim is to keep down the gold value of the return mill solution. Therefore, the press was connected with the return launder by a 3-inch pipe, and with a head of 24 feet 500 tons of solution can be clarified daily and passed to the extractor boxes. The press is sufficiently high for a truck to be run under, and is cleaned out about every ten days. A spare set of cloths is then put on, so that the old ones can be washed out at leisure, and in about 2 hours the press is ready to start again. The cloths generally used are a good quality of unbleached calico. The press has a filtering area of 425 feet, but it only occupies a space 14 feet x 4 feet x 3 feet, where, as a clarifying tank, to have the same filtering area, would require a diameter of 23 feet.

The author has also carried out filter press clarification in another instance. The final wash in his large Dehne press is 10 minutes with water, the latter being obtained from the mine. Previously this water gravitated from a reservoir through a 3-inch pipe to a settling tank, and the suction of the washing pump entered the tank about 18 inches from the bottom. A large amount of red slimy ooze, however, accumulated behind the high pressure cloths, and in time interfered with the efficient washing and closing of the presses.

The author interposed one of the earliest patterns of Johnson presses between the reservoir and the settling tank, reducing the connection to 1¼ inches. With only twelve 24 x 1-inch cells this press gives, with 20-foot head, 150 tons of perfectly clear water daily, about three times more than the requirements, and the trouble with the accumulation behind the high pressure cloths has disappeared. In the author's opinion presses are certainly ahead of tanks for the clarification of liquids, inasmuch as, assuming that the first cost of a tank and of a press of a certain filtering area are equal, the labor and time in cleaning the press is much less; the clarifying capacity with even a 15-foot head is much greater; the space occupied by the press is only one-fourth that of the tank, and the solution contained in the press is only about one-sixtieth. This, in the case of gold solutions, is worth consideration.

Reduction Plant of the Meyer & Charlton G. M. Co., Transvaal.—The equipment of this plant, which has been erected by the Messrs. Denny, and embodies the latest ideas in the sliming of gold ores and the cyanide treatment of the slimes has been previously described. The following data are given in a note in the *Journal* of the Chemical, Metallurgical and Mining Society of South Africa, September, 1906, after the plant has been in operation for ten and one-half months. The stamp duty is stated to be 6.2 tons per stamp when using a 400-mesh screen, the stamps, nominally of 1,100 pounds, but having a net running weight of 1,050 pounds, dropping ninety-eight times 8½ inches per minute, the discharge being maintained as nearly as possible at 2 inches. The theoretical extraction during the above-mentioned period is stated to have been 94.2 per cent, and the actual recovery 94.8 per cent, the slight difference being probably due to the screen assay being a little too low, owing to the system in use.

Masonry Foundations for Mortars of Stamp Mills.—The matter of battery foundations built up from wooden blocks versus masonry foundations, has called forth a good deal of discussion, and there still exists much difference of opinion on the subject.

Mr. A. B. Foote, in the *Engineering and Mining Journal*, Nov. 10, makes a strong plea for the masonry foundation. He considers that because some of them, through faulty construction, have proved to be failures, there is no reason for considering masonry unsuitable for motor foundations. He holds that the old-time wooden built-up blocks have been the weakest point in the construction of stamp mills, as they required to be renewed in from eight to ten years at considerable expense, meanwhile causing a serious loss of profit during the time of renewal. In support of his opinion of the masonry foundations he quotes three instances where they have given good satisfaction. In one [illegible] masonry foundations were built up from the old foundations, and granite capstones, 6 inches wider than the bottoms of the mortars and 18 inches thick, were placed on the top of the masonry, the capstones being long enough to make a close joint under the centers of the battery posts. The latter were cut off and bolted to cast iron pedestals, which in turn were anchored to the masonry, the anchor bolts passing through holes drilled in the granite capstones. The old anchor bolts were used for holding down the mortars, also passing through holes drilled in the capstones. A thin sheet of lead was placed between the bottom of the mortars and pedestals and the capstones. When the mill was started some of the mortars "teetered" on their foundations, and it was finally ascertained that they were slightly convex on the bottom. The tops of the capstones were then dressed slightly concave, and a sheet of ¼-inch rubber substituted for the sheet lead under all the mortars and pedestals. No further trouble has developed, the anchor bolts do not work loose, and the rubber does not seem to wear. The mill has now 1,000-pound stamps dropping 8 inches ninety-six times per minute.

In a second case the mortar foundations of a ten-stamp mill were of concrete, in the proportion of 1 cement, 5 sand and 9 broken stone, nothing being placed between the mortars and the concrete to act as cushion. The top of this foundation went to pieces soon after the mill was started, but after removing the poor concrete to a depth of 2 feet, and building up again with the proportions of 1 cement, 2 sand and 3 broken stone, no further trouble was experienced.

The third case is that of a forty-stamp mill equipped with 1,000-pound stamps, dropping 8 inches ninety-six times per minute, which has now been running over two years. The mortars and the steel battery frame were designed especially for masonry foundations, the latter being of good quality rubble masonry, the top portion being higher grade, with mortar of the proportions 1 cement to 2 sand. The top surface was leveled off with great care, a sheet of rubber 1/32 inch thick being placed between the iron and the masonry. Not the slightest movement of any of the mortars or pedestals has been observed. The conclusions to be drawn are, therefore, that the masonry must be of good quality and the top must fit the bottom of the mortar properly. The more uneven the surface, the thicker must be the rubber cushion, but it is best to have a smooth surface and a thin rubber. The use of anvil blocks under the mortars is not to be recommended, as there is apt to be trouble in keeping the mortars from dancing around on top of the anvils. The author has not observed and does not believe that solid foundations increase the breakage of stems.

Metallurgical Plants and Practices of Yavapai County, Arizona.—An article in the *Mining Reporter*, Nov. 1, 1906, deals with the metallurgical methods in the above region, the aggregate daily capacity of the plants being over 6,500 tons, the seventy mills having a total capacity of 3,250 tons, while the six smelting plants have a combined capacity of 3,300 tons. The treatment at nearly all the mills consists in wet concentration, usually preceded by amalgamation, and in a few instances followed by cyanidation. It is stated that of the cyanide plants only at those where the concentrates or tailings material is subjected to roasting before delivery to the leaching vats has cyanidation treatment been attended with any marked success. the Congress plant, of 300 tons capacity, practicing successful stamp crushing to about 30-mesh, table concentration, roasting of concentrates and tailings, and leaching in a strong solution of potassium cyanide. At the several other plants which successfully practice cyanidation, the slimes and tailings, also after roasting are cyanided, the concentrates carrying precious metals constituting the bulk of the shipping product. The stamps, of which there are over 700 in the county, weigh from 800 to 1,150 pounds, the tendency being to use stamps of 950 pounds or more. They drop 90 to 100 times per minute, the [illegible] to 105 drops per minute, of from 7 to 8 inches, the

battery screens having from 20 to 40-mesh, according to the fineness of the gold contained in the quartz. Inside mortar amalgamation is stated to be almost entirely done away with, narrow straight-backed mortars being generally preferred. The amalgamating plates are, as a rule, of a length of from 8 to 12 feet, with widths of 6 feet. At several mills the plates, of a drop pattern, are as much as 18 and even 24 feet long. The duty per stamp is 3 to 4 tons, the recovery by amalgamation ranging from 30 per cent to 75 per cent on oxidized ores, while one plant claims to get a recovery by amalgamation alone of nearly 90 per cent on $15 and $20 highly oxidized ores. On sulphides the recovery by amalgamation is from 10 to 30 per cent. The ratio of concentration varies widely, the average being somewhere between 6 into 1 and 9 into 1. The Pfau mine, in the Cherry Creek section, follows the practice of amalgamation-concentration-cyanidation, the table tailings being reground in tube mills to a fineness of 120 and 160-mesh before being subjected to cyanidation. Wherever regrinding of the coarse table tailings has been done in roller mills of the ordinary type, the recovery by cyanide is stated to have rarely exceeded 75 per cent, whereas the saving at the Pfau plant is reported to have been between 85 and 90 per cent, in continuous operations of several months. The new mills are also equipped with improved mechanisms for a closer classification of battery pulp before it is being distributed to the concentrating machines, a necessity for successful work which was much neglected in the plants built previous to 1900, as it has been in a number of other districts in the West.

Copper.

Depreciation of Copper Smelting Plants.—In a communication to the *Engineering and Mining Journal*, Nov. 10, Mr. E. F. Matthewson, the manager of the New Reduction Works of the Anaconda Copper Mining Co., at Anaconda, Mont., expresses his opinion that the usually accepted depreciation of 10 per cent per annum is too high for modern copper smelting plants. He thinks that such a plant, consisting of the following departments, viz.: concentrating, roasting, reverberatory smelting, blast furnace smelting, converting, steam and electric powe plants, should last twenty years, assuming that repairs are kept up in good style. Therefore, only 5 per cent of the cost of the original plant should be written off yearly.

In regard to the question of the salvage obtainable when a plant is dismantled, Mr. Matthewson calls attention to the fact that a great many items of expense will yield no salvage. The principal items of expense are lands, buildings, engineering expenses, interest, machinery and the installation of the same. The values recoverable, for instance, from a typical concentrating and smelting plant for copper, if it were dismantled, are as follows: 44 per cent of the equipment would yield no salvage, 2 per cent would yield 2 per cent of its original cost, 11 per cent would yield 5 per cent of its original cost, 37 per cent would yield 10 per cent of its original cost, 2 per cent would yield 15 per cent of its original cost, 2 per cent would yield 20 per cent of its original cost, and 2 per cent would yield 25 per cent of its original cost; thus, 100 per cent would yield 5.49 per cent.

Lead and Zinc.

The Flotation Process at Broken Hill.—A concise review of the flotation process for the recovery of the zinc and lead sulphides at Broken Hill, Australia, by Mr. D. Clarke, in *Engineering and Mining Journal*, Nov. 24, furnishes some data concerning the present status of these processes. It is stated that in connection with the Potter process, the Gloyder Lungton plant has been discarded and an ordinary lead-lined spitzkasten, is being used. The tailings are led into this spitzkasten, most of the zinc sulphide and part of the lead sulphide rising as a thick scum, which continually overflows. The residues fall to the bottom and are continuously discharged. The solution used is sulphuric acid, rarely exceeding 2 per cent of acidity, and it is run in at a temperature near the boiling point, through a vertical pipe, which discharges the liquid near the bottom of the spitzkasten. It is claimed that from freshly crushed ores over 90 per cent of the zinc contents is obtained. In the Delprat process, while the plant is similar, the spitzkasten has two points, one a blind one called an appendix, and the other the usual discharge opening, the down stream of hot acidulated liquid being directed into the appendix. The solution is said to be common salt added to dilute sulphuric acid. The capacity of these acid treatment plants is stated to be enormous, the plant at the Proprietary mine being erected to treat 1,000 tons per week, while it easily treats double that quantity. The total cost of the plant is very low, as it is simple and easily worked, the feeding of the ore, the discharge of ore and concentrates and the washing of them being done automatically on Robins belt conveyors. The third process in operation, the De Bavay process, depends upon the cleaning of the particles of sulphides with acid, so that when the gangue and sulphides after this treatment are immersed in water, only the metallic substances will float. After the preliminary cleansing with acid, the powdered ore or tailings are fed upon an inclined belt, which gradually delivers them below the surface of a sheet of water; the gangue is then submerged, but a film of sulphides floats and is continuously drawn off. A fourth process, the granulation process, is stated to do good work at the Central mine on slimes; it consists in agitating tailings with a very small proportion of oil and dilute sulphuric acid, at about 100° F. After thorough mixing, the product passes over spitzkasten, the sulphides flowing over the top while the gangue is withdrawn below. The sulphides are cleansed from oil by washing with dilute caustic soda, the saponified solution being allowed to work on fresh ore, the oil being again liberated by the addition of more acid. Of the various theories which have been elaborated to explain the flotation processes, the latest one is promulgated by Mr. De Bavay, who holds that the presence of both colloidal silicic acid and sulphur is necessary for the process to be worked successfully. He claims that colloidal silicic acid is generated when ores containing fluorides are treated with acids, hydrofluoric acid forming silicon fluoride, which in aqueous solution decomposes into silicic acid and hydrofluosilicic acid. Sulphur is liberated from polysulphides. Both of these form colloidal mediums for holding the gas which is evolved, and which attaches itself to the sulphides, and when once a layer or scum of sulphides forms this medium binds them together, preventing them from falling back. Thus any gases evolved will push the layer from below and not drag up individual particles of sulphides.

Iron and Steel.

Granulation of Liquid Blast Furnace Slag.—The question of the efficient granulation of blast furnace slag is of considerable commercial importance, f. i., for the manufacture of slag cement. A new type of apparatus, which depends upon the use of a horizontally arranged cylinder, is described by Mr. G. Hofer in the *Giesserei Zeitung*, Sept 15. The apparatus is illustrated in Fig. 2, and consists essentially of a cylinder *d*, which is provided in its circumference with a series of ribs *c*. The liquid blast furnace slag is conducted to the cylinder by means of a runner *a*, and the granulation can be regulated by adjusting the relative position of the runner *a* with regard to the circumference of the horizontal cylinder, as well as by varying the opening of the runner or regulating the speed of revolution of the cylinder. The ribs *c* are preferably arranged somewhat ascending in relation to the periphery of the cylinder, so that the ribs do not come into contact all at once with the molten slag for their entire length. The lime and other ingredients required for the manufacture of slag cement can be mixed with the slag at the time of granulation, and they can be introduced by means of a runner *b*. This runner is preferably located directly above the slag runner, so that the materials are to a certain degree preheated by the heat of the slag. A construction of this description embodies

various advantages over the type of apparatus which is provided with a vertical shaft and a horizontal circular plate, by which the slag is thrown towards all sides. In an apparatus of this kind the circumferential speed of this horizontal plate at various points of its area is, of course, irregular, according as the point where the molten slag strikes it is more or less distant from the center. Those particles of slag which strike more towards the center, are thrown off with much less force and are apt to remain lumpy, instead of being thoroughly granulated. On the other hand, the circumferential speed of the apparatus described above is the same at all parts of the circumference of the cylinder. Moreover, with the latter type of apparatus it is comparatively easy to increase its capacity, which is done by simply lengthening the cylinder, while the widening of a horizontal plate of necessity is confined to very narrow limits on account of the resulting strain upon the vertical shaft and the consequent irregular, wavy motion of the apparatus. As a blast furnace plant usually possesses several blast furnaces, the apparatus can be so installed and the tapping of the furnaces so regulated, that a continuous operation is possible.

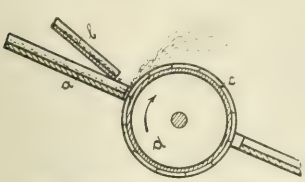


FIG. 2. PULVERIZATION OF SLAG.

RECENT METALLURGICAL PATENTS.

IRON AND STEEL.

Nickel-Silicon Steel.—A. de Dion and G. Bouten (836,567, Nov. 20) patent the composition of nickel-silicon steels with a high elastic limit and a great resistance to shocks, and said to be especially useful for shafts, gearings, springs, etc. Nickel steels are divided into three classes, according to their microstructure, whether pearlitic or martensitic or polyhedrous. If to a nickel steel of pearlite microstructure silicon is added in the proportion from 0.5 to 2 per cent, with respect to the steel, the microstructure remains pearlite, but the breaking load and the elastic limit are greatly increased, hardly without a diminution of the striction, the lengthening and the resistance to shocks. The following results were obtained with steels annealed at 900° C., and slowly cooled:

			Breaking Load.	*Elastic Limit.*
0.15 C	2 Ni	0 Si	41 kg	30 kg
0.15 C	2 Ni	1.5 Si	90 kg	65 kg
0.80 C	2 Ni	0 Si	90 kg	45 kg
0.80 C	2 Ni	1.5 Si	125 kg	75 kg

If to a steel of martensitic microstructure 0.5 to 3 per cent of silicon is added, the microstructure is not changed, but the breaking load and elastic limit are increased even more than in the above examples, the fragility of the steel being not increased.

Brick from Fines and Dust.—Kondo (836,586, Nov. 20) brings sulphide-ore fines and dust and smalls of coke into brick form for treatment in the blast furnace by pouring the fines together with a stream of molten matte into a mould. The smalls and the matte combine as a conglomerate mass and are later broken up into lumps of proper size.

SODIUM

Sodium for Electric Conductors.—According to E. A Ashcroft, the present annual production of sodium in the world is about 3,500 tons; of these 1,500 tons are used for cyanide making, and 1,500 tons for peroxide making. In this country metallic sodium is made at Niagara Falls by the Niagara Electrochemical Co., which [illegible] Hasslacher Chemical Co. [illegible] ide and other peroxides. [illegible] ing. The increase of the plant of the Niagara Electrochemical Co. in recent years was [illegible] mand both of cyanide and peroxides. On the other hand, it would seem that any possible further decrease in the price of metallic sodium might [illegible] other fields, notably as a strong [illegible] various purposes.

The very original idea of using metallic sodium for electric conductors is revealed in a patent of A. G. Betts (833,290, Oct. 16). It is based on the low specific gravity of sodium and its comparatively good [illegible]. Weight for weight, sodium has the highest electric conductivity of all metals. To use it as conductor it must, of course, be protected against air; it must be hermetically enclosed in a sheating of another metal. Betts forms a composite conductor by running sodium, while in a molten or plastic condition, into iron or steel tubes of [illegible] length. The greatest trouble in practice would probably be in making the joints. Betts describes several methods. One way is by plugs, which hermetically seal the ends; another way by sleeve-couplings, in which case the ends of the sodium rods in the two tubes are brought into close contact by heating the pipe until the sodium is fused and runs together. The following comparison is made with copper conductors: "A 2-inch wrought iron pipe has a tensile strength of about 37,500 pounds. Filled with sodium the total weight per foot is about 3.6 pounds. This would carry a current of, say, 860 amps., of which the iron carries about 70 amps. A soft, pure copper conductor of equivalent conductivity is about 1.15 square inches in cross-sectional area, weighs 4.4 pounds per foot, and has a tensile strength of about 32,000 pounds. For conductors of large current-carrying capacity the cost of the composite conductor, even at the present cost of sodium, is very much less than that of a copper conductor of equal capacity." The present price of sodium is, according to price lists, about 50 cents per pound. But that is the retail price. For large, steady contracts it is said that the price goes down to 25 cents per pound.

TIN.

Detinning Tin Scraps with Chlorine.—We have repeatedly referred in these columns to the development of the chlorine detinning process by Dr. Karl Goldschmidt, of the old electrolytic detinning firm of Theodor Goldschmidt, of Essen-Ruhr, Germany. The new chlorine process in its simplest form was described in our October issue, page [illegible]. A patent recently granted to Dr. Karl Goldschmidt and his chief chemist, Dr. Joseph Weber (836,400, Nov. 20), relates to an apparently small but particularly interesting detail of the process. It is essential to carry out the process with dry chlorine, and in such a way that as far as possible anhydrous chloride of tin is produced. In such a dry process the use of washing with water, as disclosed in the present patent, seems peculiar. The reason for the washing is as follows. If bundles of tin scrap have been decoated by means of chlorine or electrolytically with an acid solution they have a bright steel-gray surface, but after a [illegible] they rust and are then of no value for treatment in the open-hearth furnace. It has been considered to be one of the special advantages of electrolytic detinning in an alkaline solution that the bright surface of the iron produced thereby does not corrode. Goldschmidt and Weber have now found that the corrosion referred to before in connection with the chlorine process is due to a very thin but firmly adhering layer of chloride of iron. They state that the chlorine treatment completely removes the layer of tin and does not attack the iron. To explain the [illegible]

of the chloride of iron, they have recourse to the very small interconnecting layer between the actual sheet iron and the actual tin, and which may be a kind of alloy of iron and tin. This alloy is said to be attacked by the chlorine, with the result of formation of chloride of iron. If this is left on the surface of the iron the latter will rust, but rusting can be completely avoided by a washing process with water, the chloride of iron being dissolved; and by subsequently flushing the iron sheets in a weak alkaline bath a thin protective layer is formed, which prevents any corrosion.

Detinning Tin Cans.—A second patent of the same inventors (836,028, Nov. 13) is perhaps still more interesting from a commercial point of view. The detinning industry has already grown to considerable dimensions, but it is probably not generally understood that so far practically nothing but the scraps from tin-can factories have been detinned. The enormous amount of tin cans which have been in actual use was not available for successful economical treatment up to the present. The reason is to be found in the necessity of getting pure tin and pure iron as end products. Used tin cans are full of impurities, and anybody familiar with the practical necessity of maintaining an electrolytic solution pure will appreciate the difficulties which would be experienced in the electrolytic detinning of used cans. In this problem everything is to be considered as impure that is not either iron or tin; not only the rests of preserves (caviar, herring sauce, etc.) which were contained in the cans, but also the paper labels, the lacquer, the coloring matter, the solder and the thin caoutchouc ring which in folding the edges of the tins is placed in the fold to make the same tight. To remove all these impurities, Goldschmidt and Weber employ a process of two steps, the first of which destroys all adhering organic substances, not including the above-mentioned caoutchouc strips. This is done by dipping the cans in a hot 3 per cent solution of sodium hydroxide. The cans are then washed with water, and in the second step of the process the solder and the caoutchouc strips are removed by sudden exposure of the boxes to a temperature of from 600° to 800° C. The solder runs off and is recovered, while at the place of the caoutchouc strips there are narrow channels which will allow the detinning substance to circulate. The tin boxes are then ready for detinning. It is reported that this process is now being successfully used to some extent in the Goldschmidt works in Essen.

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

Electric Furnaces.

Soft Graphite.—E. G. Acheson, 836,355, Nov. 20. Application filed Sept. 22, 1906.

In our September issue (page 343) we noticed that Mr. Acheson has succeeded in finding a simple way for making soft graphite in the electric furnace. The artificial graphite heretofore made by the International Acheson Graphite Co. is hard, and is well suited for use for electrodes for electric furnaces and electrolytic cells, as a paint, pigment, etc. Soft graphite will be useful especially as a lubricant, a stove polish, and for electrotyping and other purposes. The pure, soft and unctuous graphite produced by the new method does not coalesce under pressure. The process consists in "heating in an electric furnace a charge comprising essentially carbonaceous material, such as mineral coal, coke, petroleum-coke and the like, and one or more carbide-forming material, the latter being in excess of the proportion of natural ash contained in any coal, but in less than the theoretical proportion required for production of a carbide, that is, the carbon is present in greater proportion than is required to reduce the metallic compound or carbide-forming material and to combine with the base thereof with formation of a carbide." The following specific example of the process is given. An electric furnace, having a length of 18 feet between terminal electrodes, was provided with a starting core consisting of a graphite rod 7/8 inch in diameter. The active zone, 18 inches in diameter, surrounding this core was filled with a mixture of carbonaceous material and carbide-forming oxide. The materials used in this specific instance were anthracite coal, ground to pass through a 1/2-inch mesh, mixed with sand, in the proportion of 65 per cent coal and 35 per cent sand, the ash contained in the coal being calculated as a part of the sand content of the mixture. Completely surrounding the active zone above referred to was disposed a mixture of anthracite coal and sand in the proportion of 1 part coal to 2 parts of sand, this mixture having a much higher resistance than that in the active zone, and serving as an effective heat retainer. The furnace being charged in this manner the electric current was turned on, and at the beginning registered 70 volts and 78 kilowatts. After 2 hours the register showed 203 volts and 800 kilowatts, and after 9¼ hours showed 135 volts and 800 kilowatts. The register at the of 15 hours still showed 800 kilowatts, while the volts had dropped to 70, as the result of decreased internal resistance, due to the formation of graphite. When cold the furnace was opened and 962 pounds of soft, unctuous and non-coalescing graphite were removed from the active zone.

Silica is preferred by the inventor as carbide-forming material, for the reason that oxides which form fluid carbides are more or less difficult of treatment in an electric furnace.

Terminals for Electric Furnaces.—F. J. Tone, 836,155, Nov. 20. Application filed Nov. 11, 1905.

To get a good contact between the end of the carbon terminals and the metallic clamping bars (which make connection with the external circuit) a layer of compressed artificial graphite powder is interposed between metal and carbon. The metallic clamping bars are cored out to provide channels for the circulation of cooling water. To protect the exposed portions of the carbon terminals from oxidation, they are surrounded with a layer of cemented refractory material, for which purpose various silico-carbides, such as crystalline and amorphous carborundum and siloxicon are found to be well adapted. This refractory material is mixed with a suitable binder, such as cement or silicate of soda, and the mixture is brought to the consistency of stiff mortar. These refractory jackets may also be provided with circulating ducts for cooling water.

Plastic Cylinder of Fused Silica.—J. F. Bottomley and A. Paget, 836,558, Nov. 20. Application filed Feb. 28, 1906.

A cylinder of fused silica is produced in an electric furnace containing a carbon or graphite rod as resistor, around which glassmaker's sand is placed. In order to produce an externally glazed product the rough cylinder is subsequently treated in a separate furnace, or the glassmaker's sand is contained in a jacket of carbon or platinum, which is further protected on the outside against heat losses by additional sand or other heat insulating materials. The jacket is made in two parts so as to be easily removed. The resistance core is also made removable. With respect to the further treatment of the plastic cylinder of fused silica when removed from the furnace, instructions for drawing the cylinder, for blowing it and for compressing it between rollers or dies are given.

Furnace Construction.—Frank J. Tone, 834,948, Nov. 6. Application filed Sept. 14, 1905.

The object is to increase the economy of operating electric

furnaces which have large dimensions in the direction of the path of the current, the current being introduced into the furnace through terminals at the two ends, and passing from end to end through a conducting core or through the charge.

To make the working more convenient, the furnace walls are made of removable and transferable sections, and the bottom of the furnace is artificially cooled, since otherwise at the existing high temperature the masonry composing the bottom becomes conductive, and current is shunted off through it and is wasted. One form of construction is shown in Fig. 1, giving a side view of part of the furnace; 2 is a permanent end wall supporting the electrode 3, while 4 is the furnace charge. The sections of the side walls are composed of bricks assembled in iron frames 5, and held therein by retaining lips 6 at top and bottom; 8 are popes in the bottom through which cooling water is passed.

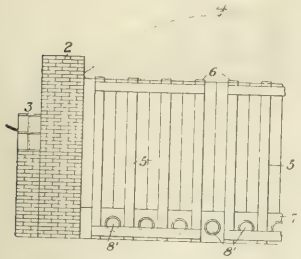


FIG. 1.—ELECTRIC FURNACE.

Molybdenum.—Fred. M. Becket; 835,052, Nov. 6. Application filed Feb. 1, 1906.

The object is to produce fused molybdenum, low in carbon, directly from the sulfide ore, molybdenite, in a single operation. For this purpose, a mixture of molybdenite, carbon and an oxide or other compound of an alkali or alkaline-earth metal is treated in an electric furnace. The reactions resulting if lime is used or calcium carbonate, are, respectively,

$$2MoS_2 + 2CaO + 3C = Mo + 2CaS + CS_2 + 2CO$$

and

$$2MoS_2 + 2CaCO_3 + 5C = 2Mo + 2CaS + CS_2 + 6CO$$

A modification of the method is to treat a mixture of molybdenite and a carbide of an alkaline metal such as calcium carbide, the reaction being in this case

$$5MoS_2 + 2CaC_2 = 5Mo + 2CaS + 4CS_2$$

The advantage of the process is that the produced molybdenum has a low carbon contact (not more than 0.2 per cent), and is free from molybdenum oxides; further that is produced from molybdenite in a single operation and with cheap reagents. For the production of ferro-molybdenum or a molybdenum-nickel or other alloys, the inventor introduces, of course, metallic iron, nickel, etc., into the charge. The process has an apparent outward likeness to the Huntington-Heberlein lead process, but in reality lime plays an essentially different part in this process (see our vol. III., pages 363 and 454) than in the above reactions. The latter have, however, many analogies with the work of Brown and Oesterle (see our vol. III., page 378).

Carbon Tetrachloride and Calcium Carbide.—J. M. Matthews, 835,307, Nov. 6. Application filed Jan. 26, 1904.

A mixture of calcium chloride and coke is heated in an electric furnace for the purpose of producing carbon tetrachloride and calcium carbide, according to the equation

$$2CaCl_2 + 5C = 2CaC_2 + CCl_4$$

The well-known corresponding equation of the ordinary calcium carbide process is

$$CaO + 2C = CaC_2 + CO$$

Calcium chloride may be had cheaply, since it is a by product of various chemical industries. On the other hand, carbon tetrachloride is a valuable product, having come into considerable favor as a volatile solvent of a non-inflammable and non-explosive character.

The carbon tetrachloride is conducted through cooled condensers, and thus brought to liquid form. It is necessary that the electric furnace reaction should take place in an atmosphere free from any oxidizing or decomposing influence. Moreover, in order to obtain carbon tetrachloride free from lower chlorides of carbon, the condensation of the highly heated vapors should take place in an atmosphere of chlorine. The electric furnace illustrated in the patent is of the well-known rotary calcium carbide furnace type, but is provided with a closed flue for carrying off the carbon tetrachloride vapors, and with an inlet pipe for the introduction of non-oxidizing gas. In the flue between furnace and condenser a chamber is interposed for condensing other volatile compounds that may be carried over from the furnace. Into this chamber also a current of chlorine gas is introduced.

Electric Furnace.—E. A. Story, Oct. 2. Application filed Sept. 13, 1905.

The furnace is designed primarily for fusing refractory substances, such as vitrifying enamel, etc. Fig. 2 shows a cross-section. The brick 21 to be fused rests on a little car running through the furnace chamber. There are two rods 13 passing through the left side-wall of the furnace (there being only one shown, the other one being behind it). These two rods are connected together by a bonded conductor 16. Through the right-hand side-wall of the furnace passes a corresponding pair of carbon rods 14, also bonded together by 16. A slab 17 of thoria, magnesia, or kaolin, which is substantially a non-conductor when cold, but becomes conducting with increasing temperature, forms the resistor which is directly

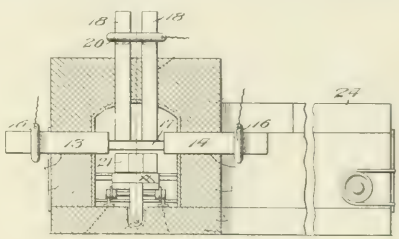


FIG. 2.—ELECTRIC FURNACE.

over the brick 21 to be fused. An additional pair of carbon rods 18 are vertically movable through the top of the furnace and are bonded together by means of 20. The electrodes 18 rest on the resistor 17. The electrodes 13 and 14, together with the resistor 17 form one conductor, while the electrodes 18 constitute the other conductor. At the rear end of the furnace a tunnel or hot-air chamber 24 is provided, which extends at a right angle to the length of the furnace. An endless conveyor receives the vitrifying material from the trucks in the furnace chamber and carries it to a point of exit.

Treatment of Zinc-Lead Ores.—F. T. Snyder, 834,644, Oct. 30. Application filed June 24, 1905.

The inventor has proposed to treat zinc-lead ores in electric furnaces from which air is excluded, the ore being smelted with carbon in the furnace to reduce both the zinc and lead, the lead being collected in liquid form, while the zinc is volatilized and condensed. Under these circumstances he found that a part of the lead also volatilized and was carried over with the zinc vapor into the condensers. He now provides an electric furnace through which the ore is passed gradually from the colder to the hotter end, all the lead being melted and carried away at the colder end, and zinc vapor being collected at the hotter end. The furnace is shown as having a hearth a to the electric smelting chamber. The temperature along this chamber is graduated by maintaining a bath of slag of varying depths within it and passing the heating electric current through the slag. Carbon electrodes [illegible] the upper end of the smelting chamber, while the lead col

lects in the well *e* at the other end and serves as the other electrode. This well *e* communicates with the exterior of the furnace by U-shaped passages. The current passes through the slag from *f* to *e*. The ore is introduced through the charging tubes *l k* and preheated by means of burning fuel in the furnace chamber *m*. This is done to start the process of reduction as far as possible and to economize in this way the more costly electric heat required in the smelting operation. The limit of preheating is that slag must not be formed in the charging tubes, since this would quickly destroy them. Since the cross-section of the body of slag is greater at the end near *e* that at the end near *f*, and since the slag at the

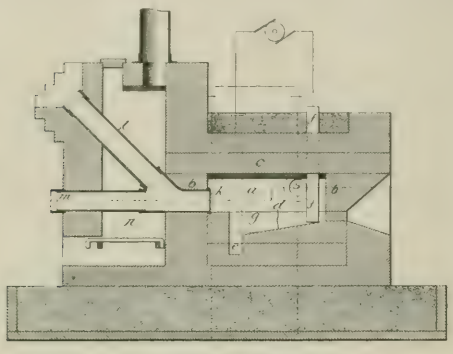


FIG. 3.—TREATMENT OF ZINC-LEAD ORE.

top is of higher resistance than the molten lead in *e*, the greatest heat is developed near the electrodes *f f*, where the slag bath is of least depth and will be gradually diminished in intensity toward the end *e* of the furnace. As the ore and coke are advanced into the smelting chamber *a*, the lead is first reduced and settles out below the body of ore and slag, running down the sloping hearth into the well *e*. The unreduced portions of the ore are then gradually advanced to the hotter parts of the furnace and zinc vapor passes off through the flues *s* into the condensers. Since the tubes *l* and *k* are always filled with material, no appreciable amount of air will enter the furnace during the charging operation, and care must be taken to always exclude air from the furnace. The temperature of the furnace in the hottest part should be about 1200°. The materials of the furnace charge should be proportioned so as to produce a slag which will form a temperature of 1000 or 1100° C. A slag containing approximately 30 per cent of lime, 30 per cent of iron oxide and 40 per cent of silica will have the desired character.

ELECTROLYTIC PROCESSES.

Electrolytic Precipitation of Precious Metals.—J. Snodgrass, 835,329, Nov. 6. Application filed Jan. 3, 1906.

This patent, coming from Johannesburg, is interesting, since it shows that work with electrolytic precipitation of gold has not been entirely abandoned in the Transvaal in favor of zinc precipitation. The precipitation box is shown in Fig. 4. The solution enters the box through 7 and leaves it through 8. 9 are the anodes, and 10 the cathodes. The latter rest on a sheet 11 of iron on the bottom, which is in electrical connection with all the cathodes. The special feature of the process is the construction of the porous electrodes. The cathodes 10 consist of very fine iron wire gauze in an iron frame. A screening of 10000 meshes per square inch is satisfactory. Inside of the gauze a woven fabric of fine texture may be used, which is rendered a conductor by a coating of plumbago. The conductivity may be increased by dipping the cloth into a solution of a lead salt and then into a solution of an alkaline carbonate or sulfate. By this means a deposit of carbonate or sulfate of lead is formed on the fibres of the cloth, and after the cathodes have been placed in position in the box, the first passage of the current reduces the lead to the metallic state, and so increases the conductivity. A combination of cloth and wire gauze may be used, woven fabrics being placed on both sides of the wire gauze network. The anodes are constructed with a wooden frame with pieces of porous cloth of very open texture, such as flax serim. The compartments thus formed are filled with an insoluble conductor in the form of powder or small lumps of gas carbon or coke. The solution flows slowly through the box, and when passing through the porous cathodes the precious metals are deposited on the same. The iron plate 11 at the bottom prevents the redissolution of any particles of

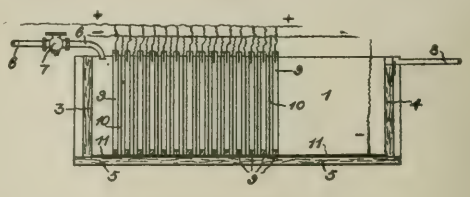


FIG. 4.—GOLD PRECIPITATION.

gold which may become detached from the cathodes 10 and come in contact with the plate 11.

Electrolytic Production of Bases of Alkali-Earth Metals.—A. Brochet and G. Ranson, 835,061, Nov. 13. Application filed Feb. 14, 1901.

The operation consists in the electrolysis of a mixture of the sulfide of an alkaline earth with a small quantity of a chloride. Under these circumstances, hydrate of the alkaline earth is formed at the cathode, and sulphur is deposited at the anode without any evolution of chlorine. The following equations represent the reduction:

$$SrS + 2H_2O = Sr(OH)_2 + H_2 + S$$
$$BaS + 2H_2O = Ba(OH)_2 + H_2 + S$$

When a diaphragm cell is used, the sulfide of the alkaline earth is placed in the anode compartment, and the alkali chloride in the cathode compartment. The chloride is made to flow in a continuous manner into the apparatus, and becomes charged with the base of the alkaline earth, and on passing out is led into the crystallizing apparatus in which the base of the alkaline earth becomes deposited by cooling. The latter is separated by centrifugal action from the mother liquid, which is returned to the apparatus. The anodic liquid consisting of a mixture of the sulfide of an alkaline earth and of a chloride of an alkali, may be when needed conducted from the apparatus and filtered, reheated and subjected to systematic lixiviation in the presence of the sulfide of an alkaline earth, whereby it may be kept saturated. The use of the chloride as catholyte enables one to obtain the base of the alkaline earth in a pure state, that is free from sulfide.

Extraction of Gold.—W. A. Hendryx, 836,380, Nov. 20. Application filed Sept. 20, 1904.

The mill is arranged on a hillside, so that the ore and solution are automatically transferred by gravity from one step of the process to the next one. At the top the ore supply and the stock solution tank are provided. The latter is filled with a weak cyanide solution. The ore, after being treated with rock breakers, is subjected to wet crushing in mortars, and then passed over an amalgamating table to recover any free particles of gold or silver. The cyanide ore-pulp solution is then run into a Chilian mill and is reground. It is then run over another amalgamating table and passed into settling and classifying tanks, where suitable chemicals are added, which thoroughly eliminate any deleterious acid salts the raw pulp may contain and which would interfere with the full action of the cyanide solution. The ore pulp flows from one compart

ment of the tank to another, and the pulp is separated from the cyanide and the cyanide is pumped back to the stock solution tank at the top of the mill. The ore pulp as it settles from the solution is divided and classified into different sizes by gravity, from the coarsest particles in the ore pulp to the finest slime. The heaviest particles settle in the first compartment, and the next largest in the second, and so on to the finest and lightest slime, which will be the last to settle. The several sizes are then drawn from the various compartments in sufficient quantity to make a charge of desired quantity and consistency of ore pulp in the agitating tank. This is the next step of the process, and represents its most important feature. The construction of the agitating tank will be described some-

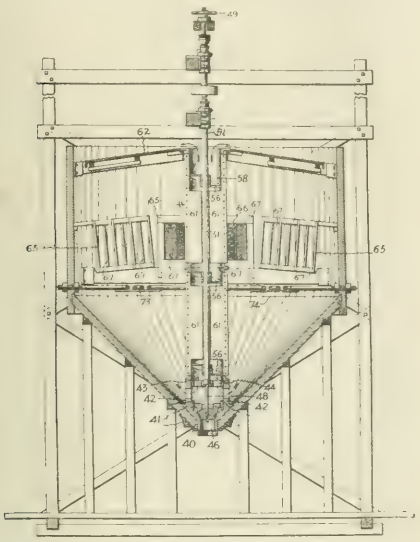


FIG. 5.—EXTRACTION OF GOLD.

what more in detail below. In this agitating tank the solution pulp is circulated, aerated, and the gold is deposited by electrolysis. After treatment in the agitating tank the ore pulp solution is passed into settling tanks. The clear solution is returned by pumping up to the stock-solution tank, while the settled ore pulp is discharged.

Fig. 5 shows the agitating tank with the discharge aperture 40 at the bottom, surrounded by the valve seat 41. The upward projecting legs 42, at substantially equal distances apart, are connected at the top by a flat ring 43, in which is secured the spider 44; 48 is a stem by means of which the valve 46 can be opened and closed with the aid of hand wheel 50. This valve stem is surrounded closely but loosely by a tubular shaft 51, carrying three screw propeller blades. The screw propellers cause circulation of the solution, as indicated by the arrows in the illustration. The solution pulp discharged at the top flows over the conical deflector plate 62 in a thin film, whereupon it drops downwards and is subjected to electrolysis between a system of anodes and cathodes. At four symmetrical places in the solution, electrode-supporting grames 65 are provided, which contain carbon anodes 66 and lead cathodes 67. These are so placed as to stand at an angle so that the surfaces of the cathodes are in the direct path of the ore pulp as it falls from the peripheral edge of the deflector into the tank.

Electrolytic Production of White Lead.—E. D. Chaplin, 836,177, Nov. 20. Application filed Feb. 3, 1906.

The inventor uses an electrolytic cell of three compartments, the electrolyte, which consists of a mixture of sodium nitrate [illegible] cathode compartment sodium hydrate is formed, and in the anode compartment lead oxychloride; the latter being an acid salt rendering impossible the formation [illegible] of lead. By a reaction between the lead oxychloride with sodium hydrate in a tank outside the electrolytic cell, lead hydrate is then produced. Some of the sodium hydrate is treated with carbon dioxide, with the result that sodium bicarbonate is formed. By reaction between the lead hydrate and the sodium bicarbonate in another tank, lead carbonate is finally produced. By means of this indirect process it is claimed that a white lead of color carrying and covering capacity is produced, which is greatly superior to that obtained by the direct process in which the soluble acid salt of lead is directly carbonated. The process is continuous, and, in addition to the lead and carbon dioxide consumed, requires only the addition from time to time of fresh quantities of neutral solution.

Electrolytic Production of Hypochlorites.—R. Kother, 832,983, Oct. 9. Application filed Nov. 3, 1905.

Mechanical details of construction of an electrolytic cell for making bleaching liquors. The construction is concisely summed up in the third claim, which reads as follows: "Electrolytic apparatus for the manufacture of bleaching liquor comprising compartments separated by non-conducting (vertical) partitions, a single electrode arranged horizontally in each non-conducting partition and projecting on either side thereof in such manner that each two adjoining half-electrodes lie horizontally one above the other forming a step-like arrangement, of the electrodes, the upper projecting half of an electrode in each compartment being perforated, the lower electrode portion in each compartment acting as an anode and the projecting half of the next electrode situated above it acting as a cathode."

Treating Cotton and Wool.—G. E. Burton, 827,293, July 31. Application filed Nov. 20, 1905.

In a vat containing an aqueous solution of sodium carbonate and sodium chloride (in the proportion of 2 to 1), with specific gravity 1.02, stirred and heated to about 80° F., wool or cotton fabrics are first exposed to electrolysis. The result is stated to be an opening, expansion and strengthening of the fibers of the wool, together with removal of the grease. The wool thereby gets a soft and velvet feeling. The solution is then drawn off into a separate vat and allowed to settle, and then brought back into the first vessel in which now an equal quantity of raw cotton is treated electrolytically for 15 or 20 minutes. The result is that the cotton takes on the appearance of wool, absorbing a certain amount of the wool grease and animal matter left in the solution from the first operation, and giving to the cotton an appearance and feeling of wool.

Galvanizing Wire.—G. L. Meaker, 830,093, Sept. 4. Application filed June 18, 1902.

To economize space in an electrolytic tank for galvanizing wire, drums are provided at the top and the bottom of the tank, and the wire is passed over them so as to form loops. Within each loop an anode is provided.

Electroplating Cylindrical Articles.—R. C. Totten, 827,473, July 31. Application filed Jan. 5, 1905.

The object is to electroplate cylindrical articles, such as metal working rolls, steels, [illegible] rolls, rods, mandrels, tubes, etc. The patent relates [illegible] mechanical details of a support on which the articles to be plated are slowly and uniformly rotated.

Anode.—J. Nelson, 830,018, Sept. 11. Application filed June 28, 1905.

In order to use metal in granular, lump or scrap form as anodes in electroplating vats, the inventor uses an anode

comprising a plate equipped with vertical marginal cleats, and a porous envelope comprising a sheet stretched across the cleated face of said plate and having its lower margin folded under the lower edge of the plate and its lateral margins folded and laced together."

Apparatus for Electroplating.—A. J. Leaver, 835,960, Nov. 13. Application filed June 11, 1906.

To agitate and circulate the plating solution, the inventor employs a screw propeller in a trunk immersed in the solution and extended as nearly to the bottom of the vat as may be desired. The trunk is bent through a right angle, the horizontal branch of it in which the propeller is placed being normally near the surface of the solution. Means are provided by which its depth can be varied. If it is desired to aerate the solution, the trunk is raised until the propeller is partly above the surface.

Electrolytic Production of Sodium.—C. F. Carrier, Jr., 830,051, Sept. 4. Application filed Jan. 30, 1905.

The construction of the cell is shown in Fig. 6, with the anode compartment in the center in form of a rectangular frame 4, 5, with graphite anodes 9. On each side of the anode compartment a cathode compartment is provided in form of a rectangular box or bell 13, containing the iron cathode 14, the lower surface of which is slightly inclined upwardly from the sides towards the center and towards one end. The middle portion of the bottom of the electrolytic cell (opposite the anode) is raised somewhat above the level of the two end portions (opposite the cathodes). The electrolyte 17 in the two cathode compartments is fused sodium hydroxide. The elec-

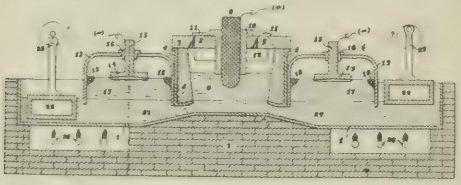


FIG. 6.—PRODUCTION OF SODIUM.

trolyte 6 in the anode compartment is fused sodium chloride, while below the electrolytes in these compartments a layer of fused lead 27 is provided which acts as a bipolar electrode and prevents the electrolytes in the anode and cathode compartments from mixing. The chemical reactions are exactly the same as in the Ashcroft process. In the anodic compartment chlorine is set free at the anode and conveyed away through the port 12, while sodium is set free at the lead 27 and alloys with it. When this lead-sodium alloy has been transferred to the cathodic compartments 17 the sodium passes into the sodium-hydroxide electrolyte and metallic sodium is set free at the cathodes 14 and is recovered. The sodium-hydroxide is not consumed. To circulate the lead-sodium alloy between the anodic and cathodic compartments, the two plungers 22 are put into a reciprocating motion, as is at once understood. In the cathode-compartment bells 13 an atmosphere of nitrogen or hydrogen is provided to prevent any oxidation of the metallic sodium which is set free. To prevent any short-circuiting between the cathode 14 and the bell 13, the cooling pipes 18 are provided, which extend entirely around the bell just below the surface of the electrolyte. In this way the electrolyte just around the cooling pipe is solidified and made a non-conductor.

BATTERIES.

Storage-Battery Plate. J. H. Robertson, 835,229, Nov. 6. Application filed Feb. 1, 1904.

By means of special tools, a series of horizontal oppositely communicating recesses are cut into the plate, "each being formed in the arc of a circle, whereby finely tapering parts are formed between the ends of the recesses and at each side of a central aperture, the bottom surface of each recess being longitudinally grooved." The object is to provide a large active surface for plate formation.

Storage Battery.—P. Schmitt and C. Fabre, 835,642, Nov. 13. Application filed Nov. 6, 1902.

A frame is made of strips of land somewhat in the form of a book-shelf, and the front and back are closed by perforated walls of celluloid or wood. The partition in this frame are filled with active material, and the special feature of the construction is that the active material is in the form of grains lying one on the other without being bound or soldered together. The horizontal sheets on which these grains rest act as conductors for the current.

Storage Battery.—E. W. Jungner, 836,261, Nov. 20. Application filed Sept. 11, 1905.

The patent refers to the manufacture of plates for his storage battery, which, in its chemical nature, is essentially the same as the Edison nickel-iron cell. The method comprises "feeding continuous metallic ribbons toward each other, perforating the ribbons, then feeding an active mass between said ribbons and finally uniting the ribbons and severing the electrodes thus formed."

Storage Battery.—T. A. Edison, 827,297, July 21. Application filed July 21, 1904.

In order to avoid the presence of active sulfides in his alkaline storage battery, vulcanized rubber insulators or supports are used, wherein all free sulphur capable of going into solution is preliminarily eliminated.

Primary Battery.—I. Kitsee, 827,914, 827,915, 827,917, Aug. 7. Application filed June 12, 1901, Feb. 20, 1904.

The inventor endeavors to avoid the use of a depolarizer in a primary cell, and tries to achieve this by using a platinized cathode which will absorb any hydrogen developed. His cathode consists of a rubber sheet on which rests a porous or perforated carbon plate, which is covered and filled with an "active material" consisting of platinized charcoal. The patent 827,917 refers to a combination of a number of primary cells in the form of a battery, whereby all the cells are connected to two reservoirs, one containing the electrolyte and the second the depolarizer, and means are provided for empyting and cleaning the cells.

Storage Battery.—W. Gardiner, 827,861, Aug. 7. Application filed Sept. 23, 1904.

The two plates are provided above one another. The lower plate is in form of a wire net, resting on the bottom of the cell, while the upper plate is made of two closely perforated metal sheets, the space between the two sheets being filled with active material. The construction seems to be specially intended for alkaline cells.

Storage Battery.—W. Gardiner, 827,968, Aug. 7. Application filed Aug. 31, 1904.

One electrode is formed of a sheet of pure lead closely perforated and folded transversely to provide a plurality of closely-arranged folds which are jammed close together, forming horizontal pockets adapted to contain active material. This plate is electrochemically formed. A silver-plated copper wire net on both sides of and below the lead plate forms the other electrode. The electrolyte is a solution of sulfuric acid and water containing sulfate of zinc, sulfate of lithium and bisulfate of mercury. In charging the metal constituents of the salts are deposited on the wire net, and constitute the negative pole-element, the wire screen serving but as a support for the deposit.

Dry Cell.—Angelica E. Post, 828,335, Aug. 14. Application filed May 17, 1905.

The ingredients are compounded in an absolutely dry condition and packed in the cell in such a way that they maintain their dry condition until required for use. Moisture is then introduced.

Galvanic Cell.—B. Jonas, 828,319, Aug. 14. Application filed April 2, 1904.

The cell contains horizontal carbon plates surrounded by granulated carbon, forming one electrode. The other electrode is granulated iron. Diaphragms are provided in form of porous cups and canvass walls. Mechanical details are provided by means of which the cell is periodically filled and emptied "with electrolytic fluid, which periodically drives out and fills the cell with air, which very greatly assists the cell in its generating power."

Separator for Storage Batteries.—L. H. Flanders, 836,107, Nov. 20. Application filed April 15, 1905. Assigned to Westinghouse Machine Co.

To separate the plates from each other, encircling insulating rings are used, provided with a downwardly-extending lug, arranged to coöperate with a recess formed in the electrode.

Dry Cell.—G. L. Tarver, 836,151, Nov. 20. Application filed March 5, 1906.

A cup of zinc sheet is amalgamated on the inside and lined with pasteboard paper. The carbon electrode in the center rests on the paper lining, and has packed around it a bottom layer of dry plaster of paris. On top of this is a thicker layer of silicious sand, and above this comes a mixture of powdered carbon and manganese dioxide. At the top is another layer of sand. The carbon-manganese dioxide mixture is brought to the consistency of a paste by adding a sufficient amount of solution, which consists of 3 parts by weight of water with 1 part salammoniac, combined with a cold mixture of 5 parts water and 2 sulphuric acid.

Dry Cell.—J. W. Brown, 836,480, Nov. 20. Application filed Sept. 14, 1906. Assigned to National Carbon Co.

To get a uniform internal resistance and thereby a uniform current density at all parts of a dry cell, the inventor makes the diameter of th ecylindrical zinc can larger at the top than at the bottom, and the carbon electrode thinner at the top than at the bottom. In this way the distance between the carbon electrode and zinc can is gradually shortened from the top downwards to compensate for the increasing distance along the carbon electrode from the binding post.

Discharges Through Gases.

Ozonizer.—M. Otto, 827,387, July 31, 1906. Application filed July 1, 1905. Assigned to American Ozone Co., Niagara Falls, N. Y.

Details of construction of an ozonizer, the elements of which are readily changeable. The particular feature is a metallic electrode forming an in-draft chamber for the air, and placed between two glass dielectrics, each of which is covered with a conducting layer connected with the ground. Several methods for producing the in-draft of the air are described. It is said that the apparatus should include in practice a cold-air generator. "Liquid air enriched by oxygen at a low temperature gives very good results."

Ozonizer.—A. Déchaux, 830,975, Sept. 11. Application filed April 28, 1904.

A battery of ozonizing units, each of which consists of three tubes. The outer and inner tubes are of metal, the middle one of a dielectrics. The three tubes are fitted within each other with slight friction. The outer and inner metallic tubes may have polygonal cross-section, while the middle glass tube may be cylindrical.

Notes on Electrochemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

The Power Problem in Relation to British Electrochemistry and Electrometallurgy.

Mr. W. B. Esson, in delivering his presidential address to the Civil and Mechanical Engineers' Society, dealt in an interesting way with the question of the supply of power to factories in general, taking as his basis a factory equipped with an engine giving 500 h.p. coupled to an electric generator working with at about two-thirds load during 2,800 hours, or 1,000,000-e. h. p. hours. Taking London prices for fuel and allowing 10 per cent for interest on capital outlay, the total costs work out as follows:

	Mond				
	Gas				
Works cost.....	0.228d.	0.294d.	0.258d.	0.294d.	0.36d.
Capital charges..	0.160d.	0.129d.	0.129d.	0.115d.	0.143d.
Total cost....	0.387d.	0.423d.	0.387d.	0.409d.	0.503d.

Of course, the above figures are for a very high load factor, much larger than that of any works, other than those devoted to electrochemistry and electrometallurgy, with which I am acquainted. Mr. Esson, however, draws a fair inference when he states that it will pay the large consumer better to use power generated at his own works instead of purchasing from a supply company.

One reservation should have been made in regard to the hours of the consumer's demand. If the industrial economy of his works permit it (in other than in cities stricken with sudden fogs), so that his power load would go off as the peak of the lighting load came on, the large consumer could undoubtedly get cheaper power from the supply company. The small consumer scores all along the line, and at 3 farthings and at 1 penny per unit, which are common charges in London, the isolated plant cannot possibly compete.

London can never be a large chemical or metallurgical center. The electroplating industries also must tend to center round Birmingham, where the works to which electroplating is subordinate are situated; electrometallurgical industries of the electric furnace type must be associated with works at the source of one or more of the component raw materials; electrochemical industries concerned in the preparation of electrolytic alkali and bleach must center around the Cheshire and Middlesborough salt districts.

Electrochemical and electrometallurgical processes of the intermittent type would probably obtain their power cheaper if it could be arranged that their operation was discontinued for 2 hours in each 24 during the peak of a lighting load. Where this is impossible the separate generating plant is absolutely essential. There is enough power going to waste in the gases of our blast furnaces at present used for firing boilers to supply power at a rate slightly cheaper even than that estimated by Mr. Esson.

Dr. Hutton's Lecture on the Electric Furnace.

The provincial engineering societies frequently begin their Winter session of papers before the London societies. The first paper of the Winter dealing with matters of interest to your readers was Dr. Hutton's missionary journey to Sheffield, to the local Engineering and Metallurgical Society, to read a paper upon "The Application of the Electric Furnace to the Metallurgy of Iron and Steel." One uses the word missionary advisedly, for it was a remarkable fact that in a paper presented to the Iron and Steel Institute, at their Autumn meeting in Sheffield last year, concerning the equipment of the Metallurgical Laboratories at Sheffield University, no mention was made of an electric furnace as part of the equipment.

Dr. Hutton's main thesis was to the effect that the advanced state of British mechanical engineering put us in the best position in regard to the possibility of power generation by steam, and that under conditions such as prevail for industries in which the electric furnace is used electrical energy could be obtained at 0.25 pence per unit in favorable localities and with

[1] It is very questionable whether the figures of Mr. Esson will be accepted on this side of the water. That steam power can be cheaper than gas power will certainly be most strongly disputed.—Editor.

large units. Taking this as his basis, Dr. Hutton proceeded to urge that the British steel maker should refrain from scoffing at new processes, just because they tried to do more than had been previously possible. Instead of propounding arguments and objections, which only served to show that the new process had not been properly understood, it would be worth while to go very thoroughly into the details of the whole question.

The cost of electricity was, after all, only a small fraction of the total costs, that even with steam power the proposition was not at all unfavorable. Sheffield was still the home of the highest class steel making, and British manufacturers had such an intimate acquaintance with the intricate technicalities of the subject, that they should be able to make the electric furnace go considerably further than those who had started with only a hazy knowledge of steel.

The Institution of Mining and Metallurgy.

The first ordinary meeting of this institution was held on Oct. 18, when three papers were discussed. The first by Mr. Malcolm Maclaren on "The Auriferous Rocks of India, Western Australia and South Africa," was mainly geological, the conclusions being set out in the following table and notes:

	India.	*West Australia.*	*South Africa.*
Cambrian or Pre-Cambrian....	Karnul beds. Cuddapah (Kadapa) beds. (Gwaliors and Bijawars of North India).	Oakover beds. Nullagine beds.	Witwatersrand system.
Archæan	Dharwar series (Aravalli, Chota, Nagpur and Shillong of North India). Fundamental gneissic granite.	"Auriferous series." Fundamental gneissic granite.	Swaziland schists. Bulawayo schists. Namaqualand schists. Malmesbury series. Gneissic granite.

Briefly summarizing, it may be stated emphatically for India and Western Australia, less so for South Africa, that: (a) In all three areas two distinct periods of auriferous deposition may be made out. (b) Both periods are connected with an original basic igneous rock, the older period occurring during the metamorphism of the igneous rock into a schist, the younger period being subsequent to the penetration of intrusive diabasic rock through the schists so formed. During the latter period the schists were mineralized anew, and, as in Western Australia and South Africa, overlying sedimentary rocks received some portion of the gold circulating in solution. (c) The younger periods may broadly be regarded as contemporaneous, and their basic eruptions as a Palæozoic prototype of the world-wide tertiary effusion of andesites with accompanying auriferous solutions.

The second paper, on "Sand Sampling in Cyanide Works," by Mr. Duncan Simpson, commenced by pointing out that "Important as is the sampling of the untreated and treated sand (respectively designated charge and residue) to the control of the recovery of gold in a cyanide works, consistent results seem to have been obtained in many cases more by accident than design. Carelessly-taken samples, at the end of a sufficiently long period, may give an average figure closely approximating to the true gold value, but instances are not unknown, more commonly when the slime separated from the mill pulp was run away to dams, of the figures representing the tonnage capacity of the treatment tanks being altered to bring the gold recovery within the limits of what has been considered normal extraction, making the tonnage treated a variable quantity dependent on the value of the sand." Eight methods of sampling were, therefore, described and discussed. Of these the author, speaking from his experience at the New Goch Gold Mine, described as unreliable the method of taking a sample by catching a sample from the delivery hose at regular intervals, and also the use of an automatic sampler. The only samples at present being taken are at the mill screens, at the discharged tailings belt, and from the slime filter presses, together with an occasional check by applying No. 4 method, in which half the sand was discharged from the tanks, leaving at the diameter a vertical face 7 feet high, which was divided into seven horizontal zones, each 1 foot wide; the face of each zone was scraped into a bucket, and the seven samples thus obtained were dealt with individually and collectively.

The third paper, on the "Treatment of the Precipitate and Manipulation of the Tilting Furnaces at the Redjang Lebong Mine, Sumatra," by Mr. S. J. Truscott, does not lend itself to abstracting in these notes, on account of the concise manner in which it is written.

An Annealing Furnace for Tool Makers.

One feature of an electrical exhibition held at Sheffield this month was an electric furnace for annealing and hardening, which was exhibited by the Electrical Co. (the English branch of the A. E. G.). Its essential features consist of two wrought iron electrodes in a rectangular crucible, a bath of a fused electrolyte of barium chloride or barium and chloride of lime, and a regulating transformer step-down transformer having a range of from 2 to 50 volts. Tests with a pyrometer indicate that the temperature is equable throughout. The following interesting table gives the energy consumption per cubic inch of electrolyte for a medium sized furnace to maintain a constant temperature:

750° C. is about 4.10 watts.	1,000° C. is about 23,000 watts.
800° C. is about 7.05 watts.	1.150° C. is about 36.10 watts.
850° C. is about 9.85 watts.	1,300° C. is about 49.20 watts.

Of course upon starting there is a heavy flow of current for a few minutes until the electrolyte becomes fluid.

Improved Conditions in the Electrolytic Alkali Trade.

The annual report of the director of the Electrolytic Alkali Co., Ltd. (who work under the Hargreaves-Bird patents) for the year ended Aug. 31 last, states that the net profit, after allowing for depreciation, mortgage debenture interest and expenditure on renewals, repairs, up-keep of buildings, plant, etc., is £12,605, added to £528 brought forward, making £12,623. Three years' dividend on the preference capital having now accrued, the directors propose to pay one and a half years' dividend upon these shares, which will absorb £10,496, leaving £2,127 to be carried forward. The favorable result is attributable to increased output and reduced cost of production.

Inventions, Inventors and the New Patent Act.

A paper read by Mr. E. Dunbar Kilburn before the Junior Institution of Engineers of Oct. 16, on the "Protection of Inventions," was mainly devoted to an attack upon the law which came into force on Jan. 1, 1905, which was chiefly notable for the introduction of an official search into British Patent Office administration. Speaking as a patent agent the author regarded the act as having been followed by a disillusionment of the popular mind, and that the inventor is not benefiting financially and otherwise as had been expected. The chief point urged is that as the official search is only undertaken upon the filing of the completed specification, it will not compare in utility with a search upon the filing of the preliminary specifica-

tion. One result may, indeed, be the disappearance of the preliminary specification.

Turning now to another question in connection with the British patent practice which had recently come in for a good deal of discussion, namely, as to whether it should be made compulsory for a holder of a British patent to manufacture in Great Britain the article protected, the author strongly held the view that the compulsory working of patents is wrong. The general tendency abroad, in those countries where the law renders working compulsory, is towards relaxation of the requirements, and it may be that at no distant date it will be found that it has become obsolete. Again, theory is opposed to practice. It sounded very nice to say that if a man owns a patent in a country he must manufacture in that country the articles covered by the patent, or at any rate a substantial proportion of what he sells in the country, in order to benefit local industries. In practice, however, it frequently happened that those who are struggling to start an invention found the continual drain on their pockets, in consequence of the recurring need for proving workings on their foreign patents, in excess of their income in the early stages, with the result that they let these patents lapse and leave the establishing of the industry or the introduction of the article into that country to take care of itself. If, on the other hand, they had been free to import articles manufactured in the home country, and so start the market for the new article, they would probably in the end have found a purchaser who would establish a local factory, and the ultimate result would be what the present legal requirement has in view. No doubt in the case of England there is much importation of patented goods made abroad, but the reason in many cases will probably be found to be either that the expenses of manufacture in England are much greater than in the inventor's home country, or that the slowness, which is so unfortunately apparent in many English firms to take up a new article and risk something in putting it on the market, drives the inventor to importing and building up a demand on his own account if he wished to get any sale at all outside his home country. Compulsory working, in the author's opinion, got at the question in the wrong way. If they wanted to give the home manufacturer a chance in competition with the foreigner, which is after all what those who advocate compulsory working are aiming at, it would seem that they must do it in a different way, and get at the foreigner when he imports. Let him make abroad as much as he liked, and let him import as much as he liked, but on terms which would prevent him from competing unfairly with the home manufacturer. Also let the facilities for the granting of compulsory licenses be made use of.

Market Quotations.

Prices are still ruling on the high side, led by the exceptional strength of the metals. A large business was done in copper sulphate, which is now strong at £31.10 per ton, an increase of £1 per ton in the past fortnight. Shellac has reached the high price of 220/ per cwt., and Para rubber is at 5/1½ to 5/2½ per pound. Copper remains firm at £99. Tin again shows an upward tendency, closing at £196.12.6 cash, and £197.10 three months, having risen £4 and £3 respectively since the end of October. Zinc was £27.10 on Oct. 31, and had fallen slightly by the time of writing (Nov. 6) to £27.5. Lead shows little variation from £19.10, a slightly upward tendency if anything.

London, Nov. 7, 1906.

BOOK REVIEWS.

The Mineral Industry, Vol. XIV. Edited by W. R. Ingalls. New York: *Engineering and Mining Journal*. Price, $5.

We have received the Mineral Industry for 1905. The general excellence of the series is as well maintained as ever. Mr. Richard P. Rothwell, who founded the journal. The scope has been well carried out by Mr. W. R. Ingalls, editor of the *Engineering and Mining Journal*. By considerable effort the volume has been brought out [illegible] in the past two or three years. The general excellence of the book is beyond question and it certainly surpasses any other technical annual published in any language. American mining men and metallurgists have a right to be proud of this book as representative of the best in American technology.

It is useless to tabulate the general articles and their respective authors. Among these might be mentioned James Douglas, chief engineer of the Phelps-Dodge copper interests and well known as a graceful writer on general subjects as well as an expert copper smelter. Prof. H. O. Hofman and Prof. Rob. H. Richards, of the Massachusetts Institute of Technology. The fourteen volumes of the Mineral Industry comprise a complete up-to-date encyclopedia of the mining and smelting business, valuable alike to the capitalists, engineer and young student.

We could have wished that the articles on electrochemistry and electrometallurgy had not been dropped, and that short biographies with photographs of the different authors had been inserted as was the custom "Consule Planco." But these are but captious criticisms in such a fine production.

Metallurgy of Cast Iron. By Thomas D. West. Fully illustrated. Eleventh edition. Cleveland, Ohio: The Cleveland Printing Company. Price, $3.00.

This book is so well known to foundrymen, and its intrinsic value is so well indicated by the appearance of the eleventh edition in less than ten years after the first edition, that little need be said concerning its technical value.

The book is well characterized on the front page as "a complete exposition of the processes involved in the treatment of cast iron, chemically and physically, from the blast furnace through the foundry to the testing machine—a practical compilation of original research."

The first part treats of manufacture and use of coke; properties in ores; operations of blast furnaces; the different brands of pig iron and how to purchase and use them intelligently.

The second part deals with the elements in cast iron and their physical effects; the utility of chemical analyses, and how to use them in making the different mixtures of irons employed in making gray and chilled castings.

In the third part, after discussing the properties of and methods for testing molten iron, the author discloses phenomena in the activities of cooling metal, etc., and presents results of tests in all kinds of iron and best methods for testing.

The book is concluded by a series of selected useful tables for furnace and foundry work.

Aside from the technical value of the book, the eleventh edition, under review, is most interesting as a human document. The chief objects for which the author has been persistently fighting in his long professional life have now been accomplished.

His standard test drillings of cast iron have been taken over by the Bureau of Standards in Washington, and it may surely be hoped that this will tend toward greater uniformity in the work of chemists in making pig iron and the mixing of iron by analysis, for making castings, etc. Much progress has been made in establishing standard methods for making chemical determinations of the different constituents of iron, as well as in grading pig iron by analysis.

These are only a few points for which the author of this book has been successfully contented, and the foundry industry owes a lasting tribute to him for his indefatigable work. We only wish that the book may be as productive of good results in the future as it has been in the past.

The Copper Hand-Book.—A manual of the copper industry of the world, Vol. VI. Houghton, Mich.: Horace J. Stevens. 1,116 pages. Price, $5.00.

This is the sixth annual edition of the well-known Copper Hand-Book. It is to be acknowledged that the author endeavors to keep the volume fully up to date. For the present edition the last two chapters have been entirely rewritten.

Of these the one dealing with the copper mines of the world represents, as heretofore, the most important portion of the book. This chapter now comprises 910 pages, and gives concise information on about 5,000 copper mining companies. While the information given is concise and to the point, there are occasionally delightful touches of humor. Thus in the glossary of mining terms an apex is defined as "that part of an ore vein at or nearest surface; usually requires opposing experts and several lawsuits to determine; in case of litigation the apex is usually owned by the litigant having the most money." In speaking of the Trinity Copper Co., the company of Mr. Lawson, of Boston, the complete working forces of the concern are summed up as follows: "Forces, one $10,000-a-year-general manager, and two $3.00-a-day watchmen."

The book in its new edition will undoubtedly be welcome to the many friends which it has won in former years.

Concentration and Briquetting of Iron Ores

We have repeatedly called attention in these columns to the fact that improvements in metallurgical methods often create new commercial values whenever they render possible the economical treatment of materials which formerly were practically valueless.

A most interesting example is presented in the progress made in recent years in concentrating low-grade iron ores and in briquetting the concentrates or fine ores for further treatment.

In the following notes the processes of Mr. Gustaf Gröndal shall be described, which have already proven commercially successful in various plants, notably in Sweden and Norway, and seem liable to land extended application also in this country. The American patents are owned by the American Cröndal-Kjellin Co., 45 Wall Street, N. Y.

Crushing and Concentrating of Magnetic Ores

The Gröndal crushing and concentrating process is commercially applicable to poor magnetites containing down to 25 per cent iron as well as magnetites with a high percentage of phosphorus and copper not chemically combined with the iron. The crude ore is reduced to about ½ inch cube in a crusher, and is then treated together with water in a Gröndal ball mill, from which it escapes as pulp in a finely ground condition. Each mill requires 20 to 25 hp. and treats 50 to 100 tons of ore in 24 hours.

The ore thus crushed is then passed through a magnetic separator. In order to get rid of the bulk of the non-magnetic slimes it is often advisable to pass the pulp coming from the ball mill through a slime box before charging it into the separator proper.

This slime box consists of an ordinary V-shaped box receiving a stream of clear water from the bottom. Two of these slime boxes are arranged under a horizontal electromagnet, either pole of which terminates in a hatchet-shaped pole piece, the edge of which nearly touches the level of the pulp in the box.

The dimensions of the box and the velocities of the pulp and clear-water currents are so arranged that everything except the finest slimes settles in the box, and passes through a pipe to the magnetic separator, the fine slimes overflowing and being drawn off to waste. Any magnetic matter contained in these slimes is arrested when it comes within the magnetic field produced by the wedge-shaped pole piece and accumulates at the surface of the water until it forms masses of such size as to drop down and be carried away through the abovementioned pipe by the issuing stream of water; the strength of the electromagnet is such as just not to lift any of the magnetic particles out of the water.

The pulp freed from slime now passes to the separator proper. A magnet with pole pieces of the same shape as those used for the slime boxes is surrounded by a drum composed of alternate bars of soft iron and brass. This drum rotates about 1 inch above the surface of the pulp, which traverses a pyramidal box, this being divided into two compartments by a partition reaching nearly to the top of the box. The pulp enters at the side opposite to the partition and a stream of clear water entering from the bottom and rising up on the same side of the partition, carries all the pulp well over the edge of the latter, and thus immediately under the drum.

The bars of iron composing the latter become powerful magnets so long as they are within the very strong magnetic field of the pole pieces, and they therefore pick up all the magnetic particles from the stream of pulp. The remainder of the pulp drops down on the other side of the partition and is carried off by the stream of water through the waste outlet.

The pure magnetite is lifted by the drum to the very edge of the magnetic field, where it is thrown off by the speed at which the drum revolves. The middlings, consisting of particles that are in part magnetite and in part barren rock, are flung off before they reach the weak part of the field. The capacity of such a two drum separator is about 50 tons in 24 hours. It requires about 6 amperes at 120 volts, or say 1 e. h. p., while less than ¼ e. h. p. will drive the drums.

Fig. 1.—Gröndal Magnetic Separator.

Some of the results lately obtained from the new type of drum separator at Herrang, give concentrates 60 per cent to 61.5 per cent iron, middlings 7 per cent to 10 per cent iron, and

tailings 5 per cent to 8 per cent iron, and from 70 to 100 tons of ore have been treated per double-drum separator per 24 hours.

BRIQUETTING.

The Gröndal process of briquetting, which has attracted wide-spread attention, serves to prepare the concentrates obtained by the above described processes for treatment in the blast furnace, but is, of course, also applicable to the treatment of fine ores and sands.

The powdered ore is pressed into briquettes of suitable size without the use of any binding material whatever, the moisture in the powder being so adjusted as to obtain briquettes sufficiently firm to be removed from the press to the cars used in the furnace. These are made with a frame of iron covered with fire-brick. The briquettes are placed on edge on the cars, the number of layers depending on the material to be briquetted.

The loaded cars are transferred one by one to a long tunnel-shaped furnace, which is gas fired, the combustion chamber being situated about the middle of the furnace. All the cars are made with a groove along one end and a projecting rib at the other, and as they are advanced by being pushed in, these fit gas-tight; the longer sides of the cars are fitted with a deep flange, with dips into a channel filled with sand and running the full length of the furnace.

The row of cars thus constitutes an air-tight horizontal partition, below which the air needed for combustion is admitted, keeping the wheels and framework of the cars cool. The row of cars does not reach quite to the discharge end of the furnace, and the air current passes below the cars to that

FIG. 2.—BRIQUETTE PRESS.

end and returns above them towards the combustion chamber, traversing the mass of burnt briquettes, cooling these and at the same time becoming itself heated.

The products of combustion pass over the entering cars of briquettes as these advance towards the combustion chamber, thus heating these and becoming themselves so thoroughly cooled that they escape into the stack at a temperature of under 100° C. The cars of briquettes are drawn at about the same temperature, when a car is taken out regularly every half-hour.

Owing to these applications of the regenerative principle, the thermal efficiency of the furnace is high, the chief source of loss of heat being the evaporation of the water contained in the raw briquette. When briquetting iron concentrates, and using producer gas, the consumption of coal in

FIG. 3.—BRIQUETTE FURNACE.

the producer has been found to average 7 per cent of the weight of briquettes burnt.

The temperature in the combustion chamber reaches 1,300 or 1,400° C., and at this heat the particles agglutinate sufficiently to form a firm, hard briquette, able to stand rough treatment and long transport. The briquettes, though hard, are very porous, and are consequently far more easily reduced in the blast furnace than ordinary lump ore.

Briquettes made by this process from pyrites residues and purple ore have proved to be entirely satisfactory for use in the open-hearth steel furnace.

The output of one furnace varies from 30 to 150 tons in 24 hours, according to the class of ore used and the degree of desulphurization required, for in addition to its mechanical action, the briquetting furnace acts as an exceptionally efficient calciner for removing practically the whole of the sulphur contained in the pulverized ore, as shown by the following analysis of ore before and after briquetting, the figures representing per cents:

Crude ore	39.3 Fe.	1.13 S.	0.006 P.
Concentrates	62.0	.27	.003
Refuse	11.4	1.58	.017
Briquettes	64.1	.008	.003
Pig iron from briquettes		.005	.042

These results were obtained at Herrang, in Sweden.

Recording Types of Le Chatelier Pyrometer.

The Le Chatelier pyrometer, with its platinum rhodium-platinum thermo couple and indicating galvanometer, has won an enviable position for the thermo-electric pyrometer in general. But the indicating form of pyrometer does not meet the requirements of many superintendents and managers. They want to know what has been going on during the hours they were not at the particular place in the works where the pyrometer is installed. They want also a record to be made and filed away which will show the exact heat treatment that was given in each particular run, so that it can be duplicated some other time or so that it may be noted and avoided if the product be unsatisfactory. They do not wish to repeat their failures, but they do wish to repeat their successes.

The result has been that there have been enough demands for a good recording pyrometer to enlist the services of the masters of that type of instrument building, and it is now

strange that the most complete and serviceable recording pyrometers have come over to us from Paris, where it has been Le Chatelier himself that has given the stimulus, with the mechanical details and design worked out by the most clever and finished instrument maker in France.

In Fig. 1 is shown the recorder which has its record sheet mounted upon a drum, and each sheet accommodates a record of twenty-six hours' run. The instrument is in a solid milled aluminium case, which, of course, cannot warp. The needle carries a reservoir pen, and the drum, counterbalanced and controlled by clockwork, comes up to meet the pen at short intervals. The record is very distinct, and its variations from a straight line show the changes in temperature. An excellent feature is the very open scale secured by having a width of 4¼ inches for the record. But we shall refer to another form of this recorder, and in connection with it shall speak of some features that are common to both.

This second very beautiful instrument (Fig. 2) is made to look a little ungainly because we illustrate it with the case open so as to show somewhat of the interior. As with the other, the case is of solid aluminium, thick, strong and dust-proof. Projecting down from the galvanometer needle is a stylus. Under the stylus travels a carbon belt, and under that is the record paper. Over the needle there is a beam, which is one piece, with the two long arms supporting it. This beam is raised by clockwork, and is allowed to fall of its own weight once a minute. The needle, which has been free to swing to its proper position, is depressed by the beam, and the stylus causes the carbon belt to make a dot on the record paper.

The record paper is thin, but opaque, and is in the form of a very long ribbon wound upon an aluminium spool. A single spool carries enough of the paper to last a whole year of continuous running, makes this instrument particularly valuable for use where a process of long duration is to be put under control.

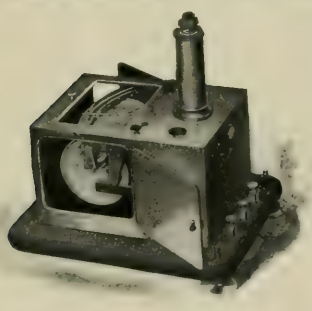


FIG. 1.—THERMO-ELECTRIC RECORDING PYROMETER.

The current generated in the thermo-couple, and which it is the duty of the recorder to measure, is the only electric current that has any function in the instrument, as the whole recording operation is controlled by an eight-day clock. By not employing any relay currents all danger of direct or induced changes in the light current to be measured is avoided. Each instrument has two different internal resistance values, either of which may be employed by using the proper binding posts, of which there are three.

In the usual course of operation, the case should be closed and locked. The clock can be wound and started or stopped without lifting the case. The record feeds out of the instrument as does the tape from a stock "ticker," and as there is a new roll of paper needed but once a year, if the instrument is continuously run, it can readily be seen how little chance there is for anyone to tamper with the works.

Of no less importance in the pyrometer is the thermo-couple, whose wires must be homogenous to avoid generation of parasitic currents along the wires. The Wilson-Maeulen Co., of New York City, who are supplying these recorders, have obtained wires for thermo-elements that are strictly homogeneous and which are sold much more reasonably than couples have been supplied for heretofore that have had certificates from the national laboratories of the European countries.

The couples this firm supply are tested, when the purchaser desires, by the Bureau of Standards at Washington. We have seen some of the certificates from the Bureau on their couples, which show that though made from wires not contiguous they yet all have the same values in electromotive force at the same temperature.

The firm announce that they can supply wires of which exact duplicates or other quantities having the same values can

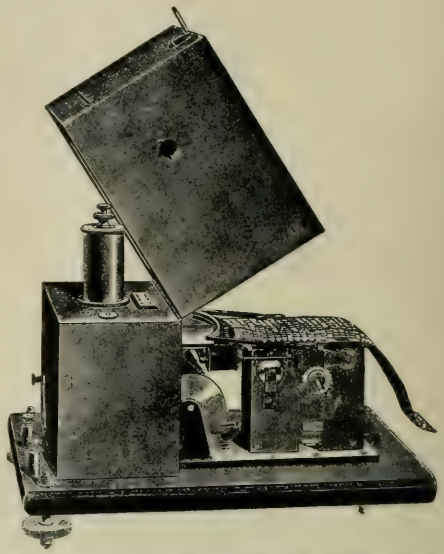


FIG. 2.—LATEST TYPE OF THERMO-ELECTRIC RECORDING PYROMETER.

be supplied for some years to come. The Bureau's report on the homogeniety of the wires shows them to be of the highest quality.

One element is of platinum and the other of rhodio-platinum, but platinum iridio-platinum will be supplied to anyone preferring it, although nowhere but in England does the latter enjoy much favor. It is perfectly reliable up to 900° C., but should not be used above that. When it is understood that platinum and irridium alloy do not hold constant above 900° C., though both metals and all their alloys melt at over 1,700° C., it is not difficult to understand why couples of base metals and their alloys are proving false in their operations when in use over 750° C. The recorders above mentioned are supplied with base metal couples when 750° C. is the upper limit at which readings will be taken.

Notes.

Tube Mills in Gold Metallurgy.—The United States Reduction & Refining Co. of Colorado Springs, have recently installed eight Abbé tube mills, 5 feet by 23 feet, equipped with "Ideal" spiral feed and discharge for regrinding in the cyanide process. Another mill of the same make, measuring 4 feet by 14 feet, has been installed at the plant of the Silver Peak Valcade Gold Mining Co., at Silver Peak, Nev.

New Publications.—Within the past fifteen or sixteen months, the Publicity Department of the Allis Chalmers Co. have covered with new bulletins about 80 per cent of the products manufactured by the company, and it has taken nearly 80 publications, averaging about 24 pages in size, to do so. This is in itself a good illustration of the variety and extent of the products of the Allis-Chalmers Co. When their list of

publications is completed, the number will be considerably in excess of 100. They have recently completed a list of those publications which were in force on Oct. 1.

Seattle Exposition.—Much interest is now taken in the West in the Alaska-Yukon-Pacific Exposition, to be held in Seattle in 1909 from June 1 to Oct. 15. The primary purpose of the fair is to exploit the resources and potentialities of Alaska, Yukon and the Pacific Northwest, and to make known and foster the vast importance of the trade of the Pacific Ocean. It is estimated that the fair will cost $10,000,000. The exposition site comprises 255 acres of the campus of Washington University, and the plan of the Exposition includes the erection of permanent buildings, which are to be taken over later by the University for educational purposes.

The End of the Municipal Hypochlorite Plant in Philadelphia.—Several years ago we had occasion to visit the small municipal hypochlorite plant in Philadelphia. It was located in an old house not far from the Public Building and was at that time only intermittently operated. It was stated, however, that the hypochlorite solution had proven quite satisfactory for disinfecting houses after contagious diseases. From a technical point of view the construction was quite primitive. The cathodes were made of zinc—a peculiar thing since an attendant might reverse the current. In the recent report of the Chief of the Electrical Bureau, Mr. John C. Sager, to the Mayor of Philadelphia, we find the following note: "The disinfecting plant for the manufacture of electrozone, which had been operated by this Bureau since 1895, having been discontinued during the last administration, was dismantled and brought into the office."

Pressed and Seamless Drawn Steel Articles.—We have received from Messrs. Janney, Steinmetz & Co., of Philadelphia, Pa., their illustrated article on pressed and seamless drawn steel specialties, such as tanks, shells, cylinders, bottles for compressed gases, etc. Each article is worked out of circular discs or rectangular pieces of plate by means of a series of operations through dies. The material is carefully annealed and worked at each operation, thereby maintaining the original vitality of the metal. The steel is low in carbon, sulphur and phosphorus, has a high limit of elasticity, and a tensile strength of 55,000 pounds to the square inch. In manufacture the circular discs or plates are pressed into cups of such diameter and length as the finished product demands. The cup is then cold-drawn into a finished product of required size. Any defect in the raw material shows itself in the early drawing operations. Such material is immediately rejected. The cold drawing makes the product smooth inside and out, and greatly increases its tensile strength as well as making it dense and tough. The gradual working of the material makes it possible to produce light and very strong articles, accurate in dimensions, free from seams, welds and riveted joints.

Gas Power.—We have received from the Backus Water Motor Co., of Newark, N. J., their catalogue on Backus suction gas producers and gas engines. In this catalogue we find the following comparative operating costs of different engines:

Type Engine.	*Fuel.*	*Price.*	*Fuel Consumption per hp. Hour*	*Cost per hp. Hour*	*Cost per hp. Month*
Gasoline engine	Gasoline	10 cents per gallon	½ gallon	.125	[illegible]
Gas engine	Illuminating gas	$1 per 1,000 cubic feet	18 cubic feet	.018	[illegible]
Gas engine	Natural gas	25 cents per 1,000 cu. feet	13 to 15 cubic feet	.00325 to .00375	50 to 58
Ordinary steam engine	Bituminous coal	$3 per ton	6 to 15 lbs.	.0075 to .0375	*2.25 to 5.5
Producer gas engine	Anthracite pea coal	$4 per ton	1 to 1¼ lbs.	.002 to .00225	47 to 58
Producer gas engine	Gas coke	$5 per ton	1¼ to 1½ lbs.	.00281 to .00375	73 to 88

* Cost of engineer and other attendance should be added to this; also large amount of water necessary.

Total amount of water used by producer is approximately 4 gallons per horse-power per hour.

Jubilee of the Coal-Tar Industry.—By a mistake of the printer the following passages were omitted from our report of the Jubilee of the Coal-Tar Industry. They should be inserted on the top of page 432: Dr. W. F. Hillebrand, president of the American Chemical Society, [illegible] cabled to Sir William the [illegible] American Chemical Society. [illegible] President N. M. Butler, of Columbia University, Dr. [illegible] P. Cadman (who spoke in an extremely [illegible] pressive way); Prof. H. Schumacher, one of the German Exchange professors who is now lecturing at Columbia, Dr. H. Remsen, of Johns Hopkins, Prof. Walter Nernst, of Berlin, Prof. H. W. Wiley (in his typical humorous style), Comptroller H. A. Metz. In his reply, Sir William first expressed his sincere thanks to all former speakers, and then gave at length an account of his struggles in his early days, how his father wanted him to become an architect, how he was attracted by chemistry, and became one of the [illegible] students of Thomas Hall in the City of London School, to enter later, at the age of 15, the Royal College of Chemistry in London, where the celebrated W. A. Hoffman was professor of chemistry. There he invented mauve at the age of 18. He described how his father and brother helped him to introduce his invention into the industry, and gave an account of his further work as a manufacturer, which ceased in 1873. After this date Sir William occupied himself purely with scientific research.

Graphite Electrodes for Electric Furnace Work. The great activity in electrochemistry in this country is indicated by the fact that the total amount of graphite electrodes supplied by the International Acheson Graphite Co., of Niagara Falls, for use in electric furnaces and similar work during the year ending July 1, 1906, was 2,404,171 pounds. It is, of course, to be understood that this embraces only the quantity of material supplied in electrode form, since very large quantities of artificial graphite are sold by the Acheson Co. for other purposes, for instance, for paint pigment. It is also known that the consumption of amorphous carbon electrodes is also very large, since for a wide range of electric furnace work the cheaper amorphous carbon electrodes will do equally well. There have been recently some very interesting devolpments in the manufacture of very large amorphous carbon electrodes by the National Carbon Co., of Cleveland, the electrodes produced being larger than any which have been made before.

Calendar.—Messrs. C. W. Leavitt & Co., in New York, importers of foreign ores, metals and alloys for iron, steel and brass works and dealers in chemicals for glass works, have sent us their calendar for 1907, with the picture of an automobile, broken down, and being hauled home by a horse under the guidance of a grinning farmer.

Applications of Ultraviolet Light.—By a misprint, which our readers will undoubtedly have recognized as such, the term efflorescent is used instead of fluorescent in the report of Dr. Baskerville's lecture on page 435 of our last issue.

T. Shriver & Co., manufacturers of filter presses, formerly of New York City, announce the removal of their office and works to Harrison, N. J., where they have increased manufacturing facilities and beter railroad accommodations for immediate shipment. This concern manufactures filter presses for all classes of industrial filtration and also small presses for laboratory work.

Public Lectures. Among the long list of free public lectures to be held in the season of 1906-7 under the auspices of

the Department of Education in the City of New York, we find the following: Prof. J. N. Gray, eleven lectures on General Physics; Prof. E. R. von Nardroff, eight lectures on Heat as a Mode of Motion; Dr. Charles Baskerville, on Radium and its Application; Mr. W. W. Ker, eleven lectures on Principles and Practice of Electrical Engineering; Prof. C. L. Harrington, eight lectures on Electricity; Prof. Morris Loeb, five lectures on the Chemistry of Carbohydrates; Prof N. A. DuBois, three lectures on Chemistry.

Aluminium.—According to the United States Geological Survey, the production of aluminium in this country was 61,281 pounds in 1890, 920,000 pounds in 1895, 7,150,000 pounds in 1900, and 11,347,000 in 1905. The production of bauxite in this country amounted to 48,129 gross tons in 1905, which is the maximum annual output ever achieved. In addition to the home production 11,726 gross tons of bauxite were imported during 1905.

Electric Furnace for Hardening and Annealing Steel.—In the London *Electrical Review*, Nov. 23, an electric furnace is described which was exhibited by S. E. Fedd at an electrical exhibition in Sheffield. It is specially adapted for hardening and annealing steel. It consists of a rectangular crucible built up of firebrick and clay within a cast iron casing. At opposite sides electrodes of wrought iron are fixed with massive connections well screened within the firebrick lining, which is of such thickness that the outer casing remains cool and but little loss of heat takes place. The bath consists of crystals of pure barium chloride, which is molten by the passage of the current, and can be made uniform at any temperatures between 750° and 1,400° C. For the higher temperatures above 1,100° C., barium chloride alone is used, but for the lower range it is necessary to mix it with potassium chloride. The cost of the charge is insignificant. Alternating currents are used with pressure regulation by means of a transformer. Experiments have shown that the temperature of the molten mass is perfectly uniform at every part of the bath and at any depth, with the exception of a layer from 10 to 15 millimeters thick at the top surface, which is in contact with the air. The rate of energy consumption is stated to vary from 4.10 watts at 750° C. to 49.2 watts at 1,300° C. per cubic inch of charge in furnaces of medium size. Over twenty of these furnaces are at present in operation, and it is stated that with gas at 30 cents per thousand cubic feet, and electric power at 2 cents per kilowatt-hour, the actual cost of hardening is about the same for the latest forms of gas ovens and for the electric furnace, and the output per month is greatly increased with the latter.

Screening and Sizing.—Bulletin 21 of the Colorado Iron Works Company describes in detail this company's patent "impact screen." The motion of this screen is perpendicular, or at right angles to the plane of the screen surface, while in all other flat screens the motion is nearly in the same plane in which the material is traveling. The motion of the impact screen, combined with the impact or vibration given to the screen cloth at each stroke, prevents small particles of ore from lodging in the meshes. By thus keeping the screen cloth open, its capacity is enormously greater than any form of revolving screen. Another advantage is the small space occupied. The impact screen of the Colorado Iron Works fills the requirements between the limits of about ¼ inch and 8 or 100 mesh, sizing pulp between these extremes in perfect manner. This screen was first brought out a few years ago, and has been improved in the meantime in some details. The same bulletin contains notes on cylindrical and conical revolving screens or trommels, on octagonal revolving screens, on hydraulic classifiers, on spitzlaesten, and on the Dow classifier, which separates the sands from the slimes in cyanide mills.

Niagara Falls. One of the strongest and ablest arguments in favor of industrial Niagara Falls is made in a letter of Mr. F. W. Haskell, president of the Carborundum Co., to Capt. Chas. W. Kutz, and recently published in the *Niagara Falls News*. The letter points out the misconceptions and misrepresentations which are at the bottom of the strong public sentiment against the industrial development of Niagara. He agrees with Mr. Buck that Niagara, as a mighty saver of labor and coal, is a vastly greater asset to the people than as a mere show place, "but while I have this belief I realize that it is Utopian." Under existing conditions the line will be drawn somewhere in the development of the waterpower. But Mr. Haskell insists that it should not and cannot honestly be drawn in such a way as to interfere with the contracted or expected power supply of any concern which has invested funds and developed an industry with the expectation, based on the good faith of the State of New York, that that investment would be protected and not strangled just as it attains maturity.

Digest of U. S. Patents.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington D. C.

Ozone and Miscellaneous Gas Reactions (*Concluded.*)

No. 652,081, June 19, 1900, J. W. Chisholm, of San Francisco, Cal.

Produces gas for heating and illumination by supplying air and steam to ignited carbonaceous fuel and conducting the products through a series of heated regenerator chambers, each filled with superposed crucibles containing "iron and copper or other material of opposite polarities." Said to be an improvement on Hall patents 494,198-9-200.

No. 659,236, Oct. 9, 1900, A. C. Johnson, of Baltimore, Md.

Passes electric sparks through the lead chambers of sulfuric acid plants, to precipitate the acid-laden moisture in suspension. The spark is said to have an effect in converting the steam and sulfur dioxid into sulfuric acid. Oxyhydrogen gas may be simultaneously introduced, the explosion assisting the precipitation.

No. 665,266, Jan. 1, 1901, F. Purdy, of Chicago, Ill.

Produces gas for fuel and illumination by passing air through ignited bituminous coal, mixing the product with oil and superheated steam, and passing through a chamber containing a filling of refractory material. An electrode having a series of horizontal iron shelves, embedded, except at the ends, in refractory material, depends centrally into the chamber, and a similar annular electrode of opposite polarity constitutes the lining of the chamber. The highly heated products of combustion passing through the refractory material are said to constitute a part of the electric circuit.

No. 667,099, Jan. 29, 1901, E. C. Paramore, of Philadelphia, Pa.

Treats chlorin gas to eliminate its odor, change its color and enhance its bleaching action, by passing the moist chlorin from a generator containing manganese dioxid and dilute hydrochloric acid, through a tube wherein it is subjected to an electric discharge. The chlorin is caused to move backward and forward through the tube by a pump, so as to be repeatedly subjected to the action of the electric current.

No. 667,100, Jan. 29, 1901, E. C. Paramore, of Philadelphia, Pa.

Apparatus for carrying out the process of the preceding patent. The glass tube through which the chlorin moves has lateral branches into which extend wire-brush electrodes, hermetically sealed in glass bulbs.

No. 671,507, April 9, 1901, R. J. Yarnold, of Streatham, England.

A number of spaced dielectric sheets, for example, of glass or mica, are interposed between two electrodes, the air or gas to be treated passing between them. The dielectrics and electrodes may be flat and arranged in a pile or tubular and concentric.

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